P
Techno
Mtm Dept
C
Colliery Engineer

Mines and Minerals

VOLUME XXXI

August, 1910, to July, 1911

122531
17/6/12

SCRANTON, PA.
INTERNATIONAL TEXTBOOK CO.
1911

INDEX

INDEX A—Authors, Subjects, and Titles of Articles

(Separate Indexes of "Book Reviews," "Personals and Obituaries," "Catalogs Received," and "Trade Notices" follow "Index A")

Page

A B C OF ELECTRICITY IN MINES..... 692
Accidental Taylor Air Compressor....... 458
Accident Investigations, Mine.......... 282
Accident Prevention, Mine.......... 276
Accidents, Liability for Industrial.......... 501
Accidents, Prevention of Mine.......... 611
Accidents to Working Men.......... 227
Acid Specific Gravity Tests, The.......... 62
Action of CO and H, or CO_2 and H on Iron Oxides.......... 454
Adams, Thomas K., James Blick, 486—Mine Fires.......... 274
Aerial Tramway, Utah Consolidated.......... 150
Aerolith Rescue Apparatus.......... 521
Air From a Hydraulic Compressor.......... 561
Air Compression, Hydraulic.......... 129
Air Compressor, An Accidental Taylor.......... 458
Air Compressor, Electric.......... 263
Air Compressors at Mines, Small.......... 477
Air Compressors, Turbine.......... 285
Air, Desiccation of Furnace.......... 723
Air Explosions, Disastrous.......... 683
Air Locomotives, Two-Stage.......... 365
Air-Nitrate Fertilizers.......... 320
Alabama Coal Mines, Convict Labor in.......... 679
Alabama Inspectors' Warning Card.......... 763
Alabama, Kellerman Mine.......... 204
Alabama, Mulga Mine Explosion.......... 40
Alabama, Natural Gas and Petroleum in.......... 175
Alaska, Bonanza Copper Mine.......... 441
Alaska Coal, Cost of Mining and Marketing.... 221
Alaska Copper Deposits, Chitina Valley.......... 315
Alaska Copper Mining, 1909.......... 318
Alaska Gold Production, 1909.......... 339
Alaskan Coal Business, Possible Effect of Panama Canal on.......... 175
Alaskan Coal Mining, No Progress in.......... 272
Alaskan Coals, Obstacles to Development of .. 657
Alaska Oil Field, Katalla.......... 731
Alaska's Coal Lands, Value of.......... 401
Alaska's Gold From Lode Mines, 1909.......... 326
Alaska, Transportation in.......... 409
Alaska, Valdez Creek Placer District.......... 572
Alchemist in Modern Industry, The.......... 688
Aldrich Pump Department.......... 506
Alkalinity, Determination of.......... 478
Alkalinity, Testing for Protective.......... 13
Alkalinity, Titrating for Protective.......... 479
Allis-Chalmers Co., The Largest Ore Crusher.. 432
Altar, Mexico, Mining Cement Gravel at.......... 229
Aluminum Bronze.......... 584
Aluminum for Prevention of Scale in Boilers... 719
American Blower Co., Test of Double-Inlet Sirocco Fan.......... 541
American Institute of Mining Engineers, One Hundredth Meeting.......... 754
American Institute Panama Meeting.......... 329
American Mining Congress..........226, 585
American Mining Congress, September, 1910 Meeting..........105, 231
American Safety Lamp and Mine Supply Co., Safety Lamps on Battleships.......... 535
American Society of Engineer Draftsmen 105
American Tin Mining in Bolivia.......... 568
American *vs.* European Coal Mines.......... 195
American Zinc, Lead and Smelting Co., Report of.......... 293
Amethyst Coloration of Glass.......... 573
Ammonium-Nitrate Explosives, Firing.......... 767
Analyses, Grading.......... 344
Announcement, Win G. Conley, Webster H. Taylor.......... 162
Anthracite and Bituminous Output, Pennsylvania.......... 485
Anthracite Cryptogram.......... 417
Anthracite, Thermochemistry of..........37, 166
Arizona, Los Pilares Mine.......... 106
Arizona, Miami Concentrating Mill.......... 1
Arkansas, Zinc and Lead in.......... 10
Asbestos.......... 469
Ashworth, James, Bellevue Explosion, Alberta, 399—Fernie Slack Bin Fire, 293—Protective Value of Humidity 188
Assay for Copper, Proof.......... 149
Assay of Copper Bullion, Combination.......... 240
Assay of Gold, Wet..........143, 423
Assayers in British Columbia, Examination for 769
Assays, Standardization of Bullion.......... 690
Aston, T. J., Rules for Mine Foremen.......... 293
Aston, W. C., Elizabeth Tunnel.......... 102
Atkinson, A. S., Restoring Dredged Ground... 422
Aubrey, Louis E., Fake Oil Companies.......... 424
Austin, C. C., Pumping at Bisbee, Ariz.......... 132
Australia, Restoring Dredged Ground.......... 422
Austria, Radium in.......... 6
Automatic Coal Sampler (Statt).......... 85
Automatic Transportation Co., New Transportation System.......... 39

BAILEY, E. E., COAL AND COKE SAMPLING, 89—Sampling Coal and Coke. .209, 190
Bailey, E. W., Can Water Vapor Be Resolved Into Its Gases in an Explosion, 225—Lamp Burning in Water, 166—Pumping.......... 38
Bailing Water at Coleman Shaft.......... 631
Balaklala Copper Co., Fume-Controlling Device 561
Baldwin Camp Lighting Outfit.......... 33
Banner Mine Explosion.......... 675
Barbour, Percy E., Developments in Cyanide Practice.......... 597
Bartlett & Snow Co., C. O., Self-Dumping Car Hauls.......... 536
Bath-House, Marianna Mine.......... 525
Battleships, Safety Lamps on.......... 535
Bavarian Graphite Industry.......... 504
Beard, J. T., Black Powder, 225—The Siphon in Mining, 427—Pumping and Siphoning Hot Water.......... 663
Bellevue Explosion, Alberta.......... 399
Belgium, Use of Breathing Appliance.......... 440
Bingham, Yampa Smelter at.......... 14
Bisbee, Ariz., Pumping at.......... 132
Black Powder..........165, 225
Blair's Zinc-Lead Pigment Plant.......... 698
Blakeley, Abram G., Gravity Determination of Coal.......... 499
Blast Furnaces, Circular Copper.......... 20
Blasting Holes Charging.......... 44
Blasting, Knox System.......... 36
Blasting, Principles of Electric.......... 393
Blasting Supplies.......... 222
Blatchley, Raymond S., Petroleum Mining Engineering.......... 4 2
Blick, James.......... 486
Blossburg Plant, Automatic Transportation Co. 39
Blowers in Coyote Mine, Gas.......... 361
Blowers, Turbine.......... 285
Blowpipe, The Oxygen.......... 163
Blumenstein, W. H., Rules for Handling Kanite.......... 681
Boiler Tests, Mechanical Stokers and Hand Firing.......... 42
Bolivia, American Tin Mining in.......... 568
Bonanza Copper Mine, Alaska.......... 441
Bonanza Mine.......... 315
Bonding in Mines, Rail.......... 673
Booth, W. H., Clay Shrinkage.......... 615
Boquillas Zinc Deposits.......... 479
Boring, Cost of Core.......... 323
Bowman, F. C., Power for Concentrating Mill.. 19
Brackett, F. Ernest, Bailing Water at Coleman Shaft.......... 631
Brady, F. Webster, Bunker Hill Coal Storage Plant.......... 166
Brattice Cloth, Use Care in Purchase of.......... 44
Brazil, Discovery of Coal in.......... 500
Brazil's Iron Ore Deposits.......... 57
Breaker, The Taylor Concrete.......... 272
Breathing Apparatus, European Laws Regarding.......... 413
Breathing Apparatus, New.......... 759
Breccia, Duty on.......... 213
Breccia, Tariff on.......... 380
Brier Hill, Pa., First-Aid Graduates.......... 491
British Coal Mining Notes.......... 666
British Columbia, Examination for Assayers in 769
Broadwater, C. C., Filter Presses.......... 682
Brokaw, Albert D., Surface Alterations of Gold Ores.......... 687
Broken Hill, N. S. W., Minerals.......... 439
Bronze, Aluminum.......... 584
B. T. U., How He Explained the.......... 82
Buffalo-Susquehanna Sagamore Mine.......... 645
Buker, H. W., Mining Iron Under the Sea..... 569
Bunker Hill Coal Storage Plant.......... 166
Bureau of Mines, The Director of.......... 162
Bureau of Mines Rescue Station.......... 68
Bureau of Mines, The United States.......... 79
Burton, Howard E., Leadville, Colo., Zinc Deposits.......... 436
Buskett, Evans W., Metal Mine Ventilation, 19—Tests for Copper Minerals.......... 430
By-Product Coke Ovens, Koppers.......... 185

CACTUS MINE, SOUTH UTAH MINE AND MILL.......... 592
Caging Machine, Mine Car.......... 413
Calama, Chile, Concentration at.......... 605
Calama Copper District, Chile.......... 473
Calcium Carbide Production in 1908.......... 369
Calcium Chloride, Desiccation of Furnace Air.. 723
Calculating the Tonnage of Ore Piles.......... 158

Page

Calculations, Portland Cement... 25
Calderwood, J. M., Hand Jig for Small Mines... 591
California, Electric Iron Smelting in... 221
California, Geology of the Coalinga Oil Fields... 4
California, Iron Ores of the Southwest... 574
Cameron Combination, Ingersoll-Rand... 462
Canada, William W., Mexican Territory of Quintana Roo... 425
Canadian Iron Ore Industry... 455
Canadian Mining Institute Meeting (March, 1911)... 596
Canal, The Panama... 241
Can Water Vapor be Resolved Into Its Gases in an Explosion... 225
Capacity of Revolving Screen... 181
Carbic Cakes... 174
Carbon Dioxide and Vegetation... 163
Carbon Dioxide in French Mines, Outbursts... 392
Carbon Monoxide, Tests for... 33
Care of Mine Mules... 650
Car Hauls, Self-Dumping... 536
Carraher, T. A., Oxygen Helmets Used at Mine Fire... 161
Carruth, J. A., New Mexico Gold Gravels... 117
Carter, T. Lane, Mining Industry of Nicaragua... 566
Catalog of Books Worth While... 612
Cement Calculations, Portland... 25
Cement Concrete Vats and Tanks... 364
Cement-Lined Pumps and Column Pipes... 506
Cement Mill, Coal Dust Explosion in a... 417
Cerium, Tariff on Alloy of Iron and... 23
Chance, Edwin M., Gravity Determination of Coal, 499—Thermochemistry of Anthracite... 166
Charging Blasting Holes... 44
Cheddite... 97
Chemistry of Coal Dust Explosions, Lievin, France... 264
Chemistry of the Cyanide Process... 689
Chemist's Place in Mining, The... 683
Chile, Calama Copper District... 473
Chilian Coal... 638
China, Tin Mining in... 131
Chitina Valley Copper Deposits... 315
Circular Copper Blast Furnaces... 20
Clancy Cyanide Process... 433
Classification of Fuels, Commercial... 397
Clay Shrinkage Under London... 615
Clifford, William, The Coefficient of Friction, 176—Warnings to Colliers... 562
Coahuila Coal Co., Explosion at Palau No. 2 Mine... 202
Coal and Coke Sampling...89, 190, 209
Coal and Coke Shipments, Pocahontas... 630
Coal and Other Exports... 526
Coal Consumption in the United States... 426
Coal Discovered in Territory of Magellan... 531
Coal Dust, Explosibility of... 369
Coal-Dust Explosion in a Cement Mill... 417
Coal-Dust Explosions...623, 746
Coal-Dust Explosions, Chemistry of, Lievin, France... 264
Coal, Export of... 494
Coal Fields of Iowa and Missouri... 80
Coal Field, The Illinois... 54
Coal, Gravity Determination of... 499
Coal in Boilers, Combustion of... 492
Coal in Brazil, Discovery of... 500
Coal in Chile... 638
Coal in Dan River District, North Carolina... 177
Coalinga Oil Field... 164
Coalinga Oil Field, Geology of... 4
Coaling Station in Southern Pacific... 700
Coal Lands, Value of Alaska's... 401
Coal-Loading Machine...66, 206
Coal Market, French... 766
Coal Mine Act, A Model (Canadian)... 546
Coal Mines of Dawson, New Mexico... 653
Coal Mine Transportation... 408
Coal Mining in India... 64
Coal Mining in Oklahoma... 193
Coal Mining Institute of America, December, 1910, Meeting... 407
Coal Mining Institute of America, June, 1911, Meeting... 38
Coal Mining News... 93
Coal Mining Notes...373, 404
Coal Mining Notes, British... 666
Coal Notes...48, 212, 281, 481, 649
Coal Pillars, Methods of Removing... 495
Coal Plants, Engineering of Modern... 57
Coal Production for 1910, United States... 417
Coal Production of the World, 1908... 538
Coal Sampler, Automatic (Statt)... 85
Coal Sampler, Mechanical... 169
Coal, Sampling... 98
Coal Seam, Working a Steep... 77
Coal Statistics, Illinois, 1910... 357
Coal Storage Plant, Bunker Hill... 166
Coal Trusts and Safe Mining (?)... 633
Cobalt, Air From Hydraulic Compressor... 561
Cobalt Ore Buying by Smelters... 705
Cobalt Ores, Hydrometallurgy of... 303
Cobalt Ores, Treatment of... 693
Cobalt, Ragged Chutes Air Compression... 129
Cobre Copper Mine... 235
Cobre Mines, Cuba... 449
Coefficient of Friction... 176
Cokedale, Colo., Explosion...547, 658
Cokedale Mine, Inspection Table at the... 752
Cokedale Plant, Falcone Oven Tile-Laying Machine... 287
Coke Oven Gas... 266
Coke Ovens in Mexico, Retort... 257
Coke Ovens, Koppers By-Product... 185
Coke, Sampling Coal and...190, 209
Coleman Shaft, Bailing Water at... 631
Coll, Aloysius, Mining Cement Gravel at Altar... 229
Colleges, Federal Aid for Mining... 743
Collieries, Power Production at... 33
Colliers, Warnings to... 562
Colliery Survey Notes, Rocky Mountain Fuel Co... 61
Colliery Warnings... 482
Collingridge, Bede, Testing for Protective Alkalinity... 13
Collins, Alonzo G., Siphon... 101
Colorado Coal Mine Commission... 418
Colorado Coal Mine Law, Proposed... 482
Colorado Fuel and Iron Co., Starkville Explosion... 261
Colorado Mine Law, The Proposed... 487
Colorado, Report of Coal Mine Inspector... 665
Colorado, Rio Blanco County... 328
Colorado School of Mines Thesis... 35
Combination Assay of Copper Bullion... 240
Combustion of Coal in Boilers... 492
Combustion of Explosives, Heat of... 429
Commercial Classification of Fuels... 397
Commission, Report of South African Mining Regulation... 337
Common Casting Copper... 421
Compensating Grades... 768
Compensation for Injury... 410
Competition Among Smelting Companies... 293
Competition, 1909, Transvaal Drill... 459
Compressor, Electric Air... 263
Concentrating Mill, Miami... 1
Concentrating Mill, Power for... 19
Concentration at Calama, Chile... 605
Concrete Breaker, The Taylor... 272
Concrete Floors, to Prevent Dusting of... 500
Concrete, Properties of... 636
Concrete Shafts in Iron Ore Mines... 740
Concrete Vats and Tanks, Cement... 364
Conical Mill, Fine Crushing With... 604
Conservation Undesirable... 384
Consolidated Fuel Co., Utah... 385
Contract, Selling Zinc Ore on... 550
Converter Flames, Copper... 718
Convict Labor in Alabama Coal Mines... 679
Copper Blast Furnaces, Circular... 20
Copper Bullion, Combination Assay of... 240
Copper, Common Casting... 421
Copper Converter Flames... 718
Copper Deposits, Chitina Valley... 315
Copper Deposits, Scientific Study of... 684
Copper District, Chile, The Calama... 473
Copper Minerals, Tests for... 430
Copper Notes, Lake Superior... 717
Copper, Proof Assay for... 149
Copper Refining, Laurel Hill Works, Brooklyn 690
Copper Smelting, Reverberatory... 557
Correction... 227
Correlation of Thacker Field... 83
Correspondence. Black Powder, 165, 225—Blick, James, 486—Can Water Vapor be Resolved Into Its Gases in an Explosion, 225—Clay Shrinkage, 615—Coal-Dust Explosions, 746—Determination of Meridian, 37, 682—Explosive Mixture, 225—Fernie Slack Bin Fire, 293—Filter Presses, 682—Haulage, 425—Howell-White Furnace, 225—Lamp Burning in Water, 166—Lamp Burns in Water, 37—Method of Laying Off a Curve, 165—Mining Around Gas Wells, 486—Mining in San Domingo, 745—Miscellaneous Questions, 293—Monazite, 101—Problem of Education, The, 745—Pumping, 37—Putting Out a Mine Fire, 101—Railroad Rates, 101—Recollections of William Smurthwaite, 165—Rolling Friction, 683—Rope Haulage, 563—Rules for Mine Foremen, 293—Safety in Mines, 562—Siphon, 101—Size of Shaft Pillar, 615—Steam in the Air-Current, 486—Steel Supports in Mines, 563—Storing Coal, 745—Surveying, 37, 166—Thermochemistry of Anthracite, 37, 166—To Find Percentage of Grade, 101—Ventilation, 101—Warnings to Colliers... 562
Coslettising, Rust-Proof Finish for Iron... 225
Cost of Core Boring... 323
Cost of Cyanide Process... 726
Cost of Mining and Marketing Alaska Coal... 221
Cost of Sinking a Mexican Mine Shaft... 275
Cottrell Fume Controlling Device... 561
Coxe & Fritz Laboratories... 35
Coyote Mine, Gas Blowers in... 364
Crawford, Robert, Power Production at Collieries... 33
Cresson Mine... 735
Crown Mines, Ltd... 691
Crusher, The Largest Ore... 432
Cryptogram, An Anthracite... 417
Cuba, El Cobre Mines... 449
Cuba, Iron Ores of Santiago... 245
Cuban Gold Mines... 240
Cuban Mining Development... 244
Cunningham, F. W., Methods of Removing Coal Pillars... 495
Curve, Method of Laying Off a... 165
Curves, Compensating Grades... 768
Cutting Machine, A Wire Rope... 85
Cyanide Practice, Developments in... 597
Cyanide Process, Chemistry of the... 689
Cyanide Process, Cost of... 726
Cyanide Process, The Clancy... 433
Cyanide Treatment of Ores in Mexico... 604
Cyanide Trust, The International... 568
Cyaniding, Lime Reaction in... 475

DAN RIVER DISTRICT, NORTH CAROLINA, COAL IN... 177
Data of Petrodynamics...210, 505
Davis, F. Jerome, Wet Assay for Gold... 143
Dawson, N. Mex., Coal Mines of... 653
Delagua, Colo., Explosion... 374
Delagua, Colo., Explosion, Notes on the... 641
Del Carmen Mine... 479
Del Carmen Zinc Mine... 437
Derailing Switches, Stop Blocks and... 753
Desiccation of Furnace Air... 723
Determination of Alkalinity... 478
Determination of Latitude... 119
Determination of Meridian...37, 682
Determining Vertical Depths in Drill Holes... 288
Developing Poverty Gulch Claims... 727
Developments in Cyanide Practice... 597
Dewaterer, Slime Settler or... 339
De Wilde Precipitation Process... 342
Diamonds in German South Africa... 426
Director of the Bureau of Mines, The... 162
Disastrous Air Explosions... 683
Divers in Mines, Submarine... 446
Dixon Crucible Co., Joseph, Lubricating Bearings of Heated Rollers... 127
Doors, Extra Mine... 216
Dreadnaught Breathing Apparatus... 759

Page

Dredged Ground, Restoring 422
Drifting Records, Vekol Mining Co. 606
Drifting with a Stoping Drill 697
Drill Competition, 1909, Transvaal 459
Drill, History of the Rock 18
Drill Holes, Determining Vertical Depths in 288
Drilling in the Upper Peninsula 295
Drill Steels, Stoping 717
Dry Placer Mining Machines 589
Dry Time in Seward Peninsula 293
Duck, Geo. F., Cokedale, Colo., Explosion, 658—Notes on the Delagua, Colo., Explosion, 641—Report of Coal Mine Inspector of Colorado, 665—Starkville, An Explanation, 355—Submarine Divers in Mines, 446—The Delagua, Colo., Explosion 374
Dump, Jeffrey-Griffith Cross-Over 622
Du Pont Powder Co., Explosives for Tunnel Driving, 159—Solid Nitroglycerine, 711—Blasting Supplies, 222—Weight of Fulminate of Mercury 59
Durable Triangulation Station 23
Duralium 314
Durand, C. S., Geology of Kolar Gold Field, 350—Yerington District 24
Dust Dampening, Kruskoff Process 630
Dust, Explosibility of Coal 369
Dust Explosion in a Cement Mill, Coal 417
Dust Explosions, Chemistry of Coal, Lievin, France 264
Dust Explosions, Coal 623, 746

EARTH COLORS OF SARDINIA 472
Earthquakes and Colliery Explosions 609
Eavenson, Howard N., Engineering of Modern Coal Plants, 57—Power Production at Collieries 189
Edgecomb, H. R., Asbestos 469
Editorial, 34, 98, 162, 226, 290, 354, 418, 482, 546, 610, 678, 742
Educating the Anthracite Miner 291
Educational 426
Education, The Problem of 745
Edwards, Edward, Tin Sluicing in Tasmania 309
Electrical Horsepower, Formula for 652
Electrically Driven Pumps 288
Electric Air Compressor 263
Electric Blasting, Principles of 393
Electric Circuit Problems 507
Electric Furnace for Zinc Smelting 793
Electric Hoists 361
Electric Iron Smelting in California 221
Electricity in Gaseous Mines, Safe Use of 126
Electricity in Mines, The A B C of 692
Electricity in Mining 756
Electric Mine Lamp, Hubbell 127
Electric Motors for Rope Haulage 174
Electric Motors Save Mine 603
Electric Service Supplies Co., Rail Bonding in Mines 673
Electric Shock in Mines 493, 543, 637
Electric Signal to Shaft Bottom, Engine and Weigh Room 353
Electric Transmission, Losses in 53
Elizabeth Tunnel 102, 137
Elliptical *vs.* Rectangular Shafts 167
Employers' Liability 354
Engineering of Modern Coal Mines 57
Engineering Societies, Library of 662
English, Teaching Foreigners 35
English to Foreigners, Teaching 60
English, Standard 99
Equalization of Fuels 405
Ernestine Mining Co., Electric Motors Save Mine 603
Estimation of Pulp, Specific Gravity 715
Eucalyptus Trees, Growing 396
European Coal Mines, American *vs.* 195
European Laws Regarding Breathing Apparatus 413
Evolution of Hoisting, 153, 251, 298—Calculating Capacity of Electric Hoists and of Shafts, 444—Deep Mines, Electric Hoists 358
Evolution of Mine Haulage, 45—Locomotives, 71—Ropes, Aerial Tramways, Cableways 45

Page

Examinations:
Illinois, Selected Questions, Springfield, 1909 182, 219
Indiana, Fire Boss and Hoisting Engineers, Terra Haute, August, 1910, 381—Mine Bosses, Terre Haute, August, 1910 279
Iowa, Hoisting Engineers', Des Moines, December, 1910, 674—Hoisting Engineers', Des Moines, October, 1910, 643—Mine Foremen's, Des Moines, October and December, 1910 539
Pennsylvania, Anthracite Mine Foreman, April, 1911, 770—Mine Foremen's and Fire-Bosses' Bituminous, April, 1910, 509—Mine Inspectors' Scranton, May 25, 1910 51, 95
Washington, Northwestern Improvement Co. 741
West Virginia, Mine Foremen's, Fairmont, November, 1910 414
Examination for Assayers in British Columbia 769
Examination of Mine Officials 741
Examination of Steels by Corrosion 703
Exeter Machine Works, The Organ Feedwater Heater 371
Explosibility of Coal Dust 369
Explosion at Bellevue, Alberta 399
Explosion at Palau No. 2 Mine 202
Explosion, Banner Mine 675
Explosion, Can Water Vapor be Resolved Into Its Gases in an 225
Explosion, Cokedale, Colo. 658
Explosion, Mulga Mine 40
Explosion, Notes on the Delagua, Colo. 641
Explosion, The Delagua, Colo 374
Explosion, Starkville, Colo 261
Explosions, Coal-Dust 623, 746
Explosions, Chemistry of Coal-Dust, Lievin, France 264
Explosions, Disastrous Air 683
Explosions, Earthquakes and Colliery 609
Expositions, Mining Industrial 593
Explosive Mixture 225
Explosives, Firing Ammonium-Nitrate 767
Explosives for Tunnel Driving 159
Explosives, Heat of Combustion of 429
Explosives, Permissible 206
Exports, Coal and Other 526
Export of Coal 494
Extra Mine Doors 216

FAHRIG, ERNEST, MINING IN SAN DOMINGO 745
Fairmont Coal Co., Testing of Miners' Oil 764
Fake Oil Companies 424
Falcone Oven Tile-Laying Machine 287
Falcone Tile-Laying Machine 187
Fans, Method of Installing 760
Fast Tunnel Driving 9
Federal Aid for Mining Colleges 743
Feedwater Heater, The Organ 371
Fernie Slack Bin Fire 293
Ferrocyanide, Solubility of Gold in Potassium 477
Ferrosilicon, Poisonous Gases From 591
Filling Stopes 465
Filter Presses 682
Fine Crushing With Conical Mill 604
Fire, Fernie Slack Bin 293
Fire, Oxygen Helmets Used at Mine 161
Fire, Putting Out a Mine 191
Fire, The Leyden, Colo., Mine 425
Fires, Mine 274
Firing Ammonium Nitrate Explosives 767
Firing, Mechanical Stokers and Hand 42
First-Aid Contest in Washington 545
First-Aid Contests (Reading, Pa., Red Cross) 197
First-Aid for Injured Spines 538
First-Aid Graduates, Brier Hill, Pa. 491
First-Aid Work at Coal Mines 490
Fitch, H. H., Handling Mine Supplies 267
Fitch, Jr., Thomas W., Method of Installing Fans 760
FitzGerald, Francis A. J., Electric Furnace for Zinc Smelting 793
Flames, Copper Converter 718
Fluorspar in Colorado 435
Foreman of Mine Rescue Station 160
Foremen, Rules for Mine 293

Page

Franz Screen, The 126
Fraser, Lee, Surveying 166
Fraser, Jr., Robert, Mining in Spain 429
French Coal Market 766
Friction, Rolling 683
Friction, The Coefficient of 176
Fuel Values, Practical 178
Fuels, Commercial Classification of 397
Fuels, Equalization of 405
Fulminate of Mercury 59
Fulminating *vs.* White Phosphorus for Igniters 76
Fume Abatement, Kennett Smelter 140
Fume-Controlling Device, Cottrell 561
Furnace Air, Desiccation of 723
Furnaces, Circular Copper Blast 20

GALLOWAY, A. A., GAS BLOWERS IN COYOTE MINE 364
Gardner, James H., Mammoth Cave of Kentucky 720
Gary, W. Va., Mine Laboratory Work at 217
Gas and Briquets From Mine Refuse 421
Gas and Its Development, Power 640
Gas Blowers in Coyote Mine 364
Gas, Coke Oven 266
Gas in Very Hot Flames 522
Gas Leaks in Coal Mines, Natural 615
Gas Wells, Mining Around 486
Gases in Water, Solubility of Rare 552
Gasoline Locomotive 158
Gasoline Locomotive for Mine Use 542
Gasoline Mine Locomotives, German 30
Gasoline Motor Haulage 629
General Electric Co., Induction Motor Runs Under Water 448
General Ore Mining Notes 584
Geochemistry 26
Geography of Petroleum 607
Geologists and Nature Fakirs 547
Geology of Herrin Quadrangle 527
Geology of Kolar Gold Field 350
Geology of the Coalinga Oil Field 4
Geology of the La Veta Coal Field 66
Georgia Brown Iron Ore Washeries 294
Glass, Amethyst, Coloration of 573
Globe, Arizona, Superior and Boston Mine, 112—Triangulation Station 23
Globe Mine, Rock Handling Arrangement at 753
Gogebic Range, Mining on the 712
Gold Deposits of San Juan, Colo 335
Gold Field, Russian Far Eastern 447
Gold Gravels, New Mexico 117
Gold in Potassium Ferrocyanide, Solubility of 477
Gold Mines, Cuban 240
Gold Mines, Philippine 464
Gold Mining in Alaska, 1909 250
Gold Mining, Southern 98
Gold Ores, Surface Alterations of 687
Gold, Wet Assay of 143, 423
Government Oil Lands Suit 419
Grade, To Find Percentage of 191
Gradenwitz, Dr. Alfred, Oxygen Process for Melting Iron 146
Grades and Screens, Standard 345
Grades, Compensating 768
Grading Analyses 344
Graduates, Lehigh University 726
Grady, P. A., Mine Law of West Virginia 370
Grahamite, A Solid Native Bitumen 458
Grapevine Tunnel 137
Graphical Notebook, A 332
Graphite Industry, Bavarian 504
Graphite, Lubricating Bearings of Heated Rollers 127
Gravels, New Mexico Gold 117
Gravity Determination of Coal 499
Gravity Estimation of Pulp, Specific 715
Gravity Tests, The Acid Specific 62
Gray, F. W., A Graphical Notebook 332
Greece, Minerals of 157
Green Cañon Mine, Stop Blocks and Derailing Switches 753
Gregory, Newman B., Zinc Mining at Yellow Pine 340
Guests on Mining Institute Excursions 355
Gunsolus, F. H., Blasting Supplies 222

Page

HALEY, M. M., ELECTRIC SIGNAL TO SHAFT BOTTOM 353
Hall, R. Dawson Norris, Compensating Grades, 768—Data of Petrodynamics, 210, 505—Sagamore Mine 645
Hancock, R. T., Calculating the Tonnage of Ore Piles, 158—The Determination of the Meridian, 37—Thermochemistry of Anthracite 37
Hand Jig for Small Mines 591
Handling Mine Supplies 267
Hanger, Trolley Wire for Mines 32
Hardinge, Fine Crushing With Conical Mill 604
Hartman, W. E., Koppers By-Product Coke Ovens 185
Harvey, Thomas C., First-Aid for Injured Spines 538
Haulage 425
Haulage, Electric Motors for Rope 174
Haulage, Evolution of Mine 45, 71
Haulage, Gasoline Motor 629
Haulage Rope Strains 39
Haussler, C. A., Mining Laws in Santo Domingo 580
Haworth, Erasmus, A Warning to Coal Men 672
Hazleton Plunger Jig 621
Heat of Combustion of Explosives 429
Hecla Mine Hoist 28
Helmet for Mine Fire Fighting, A New Smoke 281
Helmets Used at Mine Fire, Oxygen 161
Henderson, Charles W., Developing Poverty Gulch Claims, 727—Prospecting Poverty Gulch Claims, 694—Report on Poverty Gulch Mine 586
Hendrie & Bolthoff, Electric Motors for Rope Haulage 174
Herrick, R. L., Miami Concentrating Mill, 1—Old Dominion Pumping System, 324—Superior and Boston Mine Pumping Plant, 112—The Cresson Mine, 735—To Friends, 34—Tunneling on Los Angeles Aqueduct 135
Herrin Quadrangle, Geology of 527
Hiawatha Mine 385
Hinds, Henry, Coal Fields of Iowa and Missouri 80
Hirschberg, Charles A., History of the Water Leyner Drill 148
History of the Rock Drill 18
History of the Water Leyner Drill 148
Hoist, A Simple Hydraulic 157
Hoist, Hecla Mine 28
Hoisting, Evolution of, 251, 298—Deep Mines, Electric Hoists, 358—Calculating Capacity of Electric Hoists and of Shafts 444
Holden Mine, Coal-Loading Machine 206
Holes, Charging Blasting 44
Holes, Determining Vertical Depths in Drill 288
Holloway, George T., Geochemistry 26
Holmen Coaling Stations, Modern 639
Holt, Theodore P., Lime Reaction in Cyaniding 475
Hospital, Underground 490
Houses, Miners' 115
How About This 291
Howell-White Furnace 225
How He Explained the B. T. U 82
Hoy, Austin Y., Working a Steep Coal Seam 77
Hubbell Electric Mine Lamp 127
Hudson, W. G., Principles of Electric Blasting 393
Humidity, Protective Value of 188
Hunter Jig 131
Huntley, Rollo E., Slime Settler or Dewaterer 339
Hunt, J. M., Three-Wire Mine Service 402
Hutchins, Neill, Kellerman Mine 204
Hyder, F. B., Specific Gravity Estimation of Pulp 715
Hydraulic Air Compression 129
Hydraulic Compressor, Air From a 561
Hydraulic Hoist, A Simple 157
Hydrometallurgy of Cobalt Ores 303

I. C. S. ANNIVERSARY BANQUET 754
Idaho, Metal Mining Fatalities in 700
Idaho's Metal Production, 1910 686
Ifft, George Nicolas, Bavarian Graphite Industry 504
Igniters, Fulminating vs. White Phosphorus for 76
Ilgner Hoists 363
Illinois Coal Field 54
Illinois Coal Statistics, 1910 357
Illinois, Geology of Herrin Quadrangle 527
Illinois Mining Law, New 761
Illinois Rescue Station Commission 609
Illinois, Rescue Stations in 214
Important Oil Land Decision 751
India, Coal Mining in 64
Indian Coal Mines, Timbering in 179
India's Mineral Resources 763
Induction Motor Runs Under Water 448
Ingersoll-Rand-Cameron Combination 462
Ingersoll-Rand Co., History of the Rock Drill 18
Injury, Compensation for 410
Innoko District, Alaska 288
Inspection Table at the Cokedale Mine 752
Institution of Mining and Metallurgy, Report of Committee on Standardization of Assays 690
Interlocking Safety Switch Co., Safe Use of Electricity in Gaseous Mines 126
Interlocking Steel Mine Supports 664
International Cyanide Trust 568
International Smelting Co., The Tooele Smelter 321
Inventions, Simultaneous 3
Investigations, Mine Accident 282
Iowa and Missouri, Coal Fields of 80
Ireland, Peat Bogs of 212
Iron and Cerium, Tariff on Alloy of 23
Iron Mining, Gogebic Range 712
Iron Ore Deposits, Brazil's 7
Iron Ore Industry, Canadian 455
Iron Ore in Spain, Production of 577
Iron Ore Mines, Concrete Shafts in 740
Iron Ore Notes, Lake Superior 714
Iron Ores of Santiago, Cuba 245
Iron Ores of the Southwest 574
Iron Ore Washeries, Georgia Brown 294
Iron Oxides, Action of *CO* and *H* on 454
Iron, Oxygen Process for Melting 146
Iron, Rust-Proof Finish for 225
Iron Under the Sea, Mining 569
Irrigated Lands 196

JACOBS, E., CHITINA VALLEY COPPER DEPOSITS 315
Jawbone Tunnel 136
Jeffrey-Griffith Cross-Over Dump 622
Jeffrey Mfg. Co., 128—Mining Industrial Expositions 503
Jig for Small Mines, Hand 591
Jig, The Hazleton Plunger 621
Jones, C. Colcock, Iron Ores of the Southwest 574
Joplin District, The Year in 352
Joplin District, Timbering in 144
Joplin District Zinc and Lead Ores 31, 327
Joplin Zinc and Lead Production, 1910 435
Joplin Zinc Ore Notes 472
Junction Shaft Pumping Plant, Bisbee 132

KANITE, RULES FOR HANDLING 681
Kansas, A Warning to Coal Men 672
Kansas Coal District, Southern 668
Katalla, Alaska, Oil Field 731
Kellerman Mine, Kellerman, Ala 204
Kellum, William, The Problem of Education 745
Kemmerer Coal Co., Room-and-Pillar Method 88
Kempton, W., A B C of Electricity in Mines 692
Kennett Smelter Fume Abatement 143
Klier, Vitus, Mine Laboratory Work at Gary, W. Va 217
Klondike, Yukon Territory of Canada 463
Knox Blasting System 36
Koepe Hoists 361
Kolar Gold Field, Geology of 350
Koppers By-Product Coke Ovens 185
Kruskoff Process of Dampening Dust 630

LABORATORY WORK AT GARY, W. VA., MINE 217
Lakes, Arthur, Geology of La Veta Coal Field 466
Lake Superior Copper Notes 717
Lake Superior Copper Ore 149, 341
Lake Superior Iron Ore Notes 714
Lake Superior Iron Ores 152, 297, 440
Lambert, Thomas E., Circular Copper Blast Furnaces 20
Lamp Burning in Water 166
Lamp Burns in Water 37
Lamp, Hubbell Electric 127
Land Classification, Progress in 355
Langerfeld, A., The Acid Specific Gravity Tests 62
Laramie-Poudre Tunnel, Fast Tunnel Driving 9
Largest Ore Crusher, The 432
Latitude, Determination of 119
La Veta Coal Field, Geology of 466
Law, New Illinois Mining 761
Law, New Legislation Will Affect Mining Shares 748
Law, New Ohio Mine 201
Law of West Virginia, The Mine 370
Law, Pennsylvania's Bituminous Mine 742
Law, Possibilities of a New Liability 532
Law, Proposed Colorado Coal Mine 482
Lawrence, Benjamin B., El Cobre Copper Mine 235
Laws in Santo Domingo, Mining 580
Law, The Proposed Colorado Mine 487
Lead in Arkansas, Zinc and 10
Lead Production in 1910, Joplin Zinc and 435
Lead Refinery, Trail Smelter and 121
Lead Smeltery, Webb City 709
Leadville, Colo., Zinc Deposits 436
Leakage in Coal Mines, Natural Gas 615
Legalized Slavery 354
Legislation Will Affect Mining Shares, New 748
Lehigh University, Coxe & Fritz Laboratories 35
Lehigh University Graduates 726
Leonard, C. M., Cyanide Treatment of Ores in Mexico 604
Leschen & Sons Rope Co., A., Tramway at Del Carmen Zinc Mine 437
Lesson of the Price-Pancoast Disaster 611
Lessons from the Price-Pancoast Fire 678
Lethbridge, Alberta, Mining at 635
Levy, Donald M., Copper Converter Flames 718
Leyden, Colo., Mine Fire 425
Leyner Drill, History of the Water 148
Liability, Employers' 354
Liability for Industrial Accidents 501
Liability Law, Possibilities of a New 532
Library of the Engineering Societies 662
Lighting Outfit, Tent and Shack 33
Lime Reaction in Cyaniding 475
Limit of Proportionality 349
Lloyd, Bert, New Tile-Laying Machine 187
Loading Machine, Coal 66, 206
Locomotive, Gasoline 158
Locomotives, Gasoline Mine 30
Locomotives, Two-Stage Air 365
Log Washer in Zinc Mining 423
Los Angeles Aqueduct, Elizabeth Tunnel 102
Los Angeles Aqueduct, Tunneling on 135
Los Pilares Mine 106
Losses in Electric Transmission 53
"Lo the Poor Prospector" 546
Lubricating Bearings of Heated Rollers 127

MAGELLAN, COAL DISCOVERED IN TERRITORY OF 531
Magnetic Separation of Wolframite 462
Maikop Oil Field 468
Mammoth Cave of Kentucky 720
Manganese Deposits, Uruguay 286
Marianna Coal Mines 523
Marting, W. Va., Mining and Coking Plant 171
Massachusetts, New Legislation Will Affect Mining Shares 748
McBride, Harry A., Production of Iron Ore in Spain 577
McChotka, James, A Simple Hydraulic Hoist 157
McColl, Jay R., Method of Installing Fans 760
McCrossin, E. F., Georgia Brown Iron Ore Washeries 294
McGaw Shaft Pumping Plant 112
McGregory, C. M., The Tooele Smelter 321
McOlvin, E. P., Mining Around Gas Wells 486
McQuiston, J. C., Hecla Mine Hoist 28
Mechanical Coal Sampler 169
Mechanical Stokers and Hand Firing 42
Mee, G. H., The Panama Canal 241
Meetings of Institutes 38
Menne-Oxygen Melting Process 708
Mercury Production in Russia, 1910 600
Mercury, Weight of Fulminate of 59

Page
Meridian, Determination of ...37, 682
Merrill Metallurgical Co., Filter Presses ... 682
Metal Mine Ventilation ...19, 337
Metal Mining Fatalities in Idaho ... 700
Metal Mining News ... 116
Metallurgical Sleuthing ... 476
Metallurgy of Cobalt Ores ... 303
Method of Installing Fans ... 760
Method of Laying Off a Curve ... 165
Methods of Removing Coal Pillars ... 495
Mexican Rival to the Panama Canal ... 244
Mexican Situation, The ... 290
Mexican Smokeless Powder Factory ... 308
Mexican Territory of Quintana Roo ... 425
Mexico, Coal in ... 741
Mexico, Cyanide Treatment of Ores in ... 604
Mexico, Explosion at Palau No. 2 Mine ... 202
Mexico, Oil Operations in Tampico ... 406
Mexico, Petroleum Deposits About Tampico ... 50
Mexico, Retort Coke Ovens in ... 257
Mexico, Tepic Territory ... 331
Meyer, Ralph A., Porcupine Gold Fields ... 701
Meyers, H. K., Equalization of Fuels ... 405
Miami Concentrating Mill ... 1
Midvalley Coal Co., Gasoline Locomotive for Mine Use ... 542
Miller, L. F., Radium and Radioactivity ... 732
Milwaukee Locomotive Mfg. Co., Gasoline Locomotive for Mine Use ... 542
Mine Accident Investigations ... 282
Mine Accident Prevention ... 276
Mine Car Caging Machine ... 413
Mine Cave Commission, Mine Support Tests ... 749
Mine Cave Enquiry, Scranton ... 620
Mine Fires ... 274
Mine Inspectors' Institute, June, 1910, Meeting 38
Mine Laboratory Work at Gary, W. Va ... 217
Mine Law of West Virginia ... 370
Mine Promotion Literature ... 291
Minerals of Greece ... 157
Miner, Educating the Anthracite ... 291
Mine Rescue Car No. 4, Route of ... 573
Mine Rescue Car Schedule ... 621
Mine Rescue Work ... 184
Mines of Missoula County, Mont ... 581
Mines Trails Committee, Standard Grades and Screens ... 344
Mine Support Tests ... 749
Mining Around Gas Wells ... 486
Mining at Lethridge, Alberta ... 635
Mining Cement Gravel at Altar ... 229
Mining, Chemist's Place in ... 683
Mining Engineer, Civil Service Examination ... 203
Mining Engineers as Citizens ... 419
Mining Industrial Expositions ... 503
Mining Industry of Nicaragua ... 566
Mining in San Domingo ... 745
Mining in Spain ... 429
Mining in the Malay States ... 448
Mining in the Tintic District of Utah ... 553
Mining Iron Under the Sea ... 569
Mining Laws in Santo Domingo ... 580
Mining Method, Room-and-Pillar ... 88
Mining on the Gogebic Range ... 712
Mining, Power Used in ... 86
Mining, Preparing and Coking Coal, Marting, W. Va ... 171
Mining Safety Device Co., Mine Car Caging Machine ... 413
Mining Wrinkles ... 752
Miscellaneous Questions ... 293
Missoula County, Mont., Mines of ... 581
Missouri, Coal Fields of Iowa and ... 80
Model Coal Mine Act, A (Canadian) ... 546
Modern Holmen Coaling Stations ... 639
Moffit, Fred H., Valdez Creek Placer District ... 572
Moisture, Protective Value of ... 188
Monazite ... 101
Montana, Mines of Missoula County ... 581
Moon, H. E., Drifting with a Stoping Drill ... 697
Moores, Harold, Power Production at Collieries 33
Moral of Starkville, The ... 391
Moyer, Albert, Cement Concrete Vats and Tanks, 364—To Prevent Dusting of Concrete Floors ... 500
Mt. Jessup Coal Co., Peck's Shaft Breaker ... 513

Page
Mules, Care of Mine ... 650
Mulga Mine Explosion ... 40

NATURAL GAS AND PETROLEUM IN ALABAMA ... 175
Natural Gas Leakage in Coal Mines ... 615
Nevada, Yerington District ... 24
New Book Catalog ...83, 548
New Breathing Apparatus ... 759
Newfoundland, Pebble Industry in ... 249
Newfoundland, Wabana Iron Mines ... 569
New Illinois Mining Law ... 761
New Inventions, 64, 128, 192, 256, 320, 384, 448, 512, 576, 644, 708 ... 772
New Legislation Will Affect Mining Shares ... 748
New Mexico Gold Gravels ... 117
New Ohio Mine Law ... 201
New Smoke Helmet for Mine Fire Fighting ... 281
New Tile-Laying Machine ... 187
New Transportation System ... 39
New Western Pacific Railroad ... 170
Nicaragua, Mining Industry of ... 566
Nitroglycerine, Solid ... 711
North Carolina, Coal in Dan River District ... 177
Northwestern Improvement Co., Examination of Mine Officials, 741—First-Aid Contest ... 545
Northwestern Ores ... 434
No "Smoke Farmers" There ... 603
Notebook, A Graphical ... 332
Notes, Colliery Survey ... 61
Notes on the Delagua, Colo., Explosion ... 641

OBSTACLES TO DEVELOPMENT OF ALASKAN COALS ... 657
Oceania, Phosphate in ... 454
Ohio Brass Co., A B C of Electricity in Mines, 692—Electric Circuit Problems ... 507
Ohio Mine Law, New ... 201
Ohio Mines, Paraffine Oil in ... 499
Oil Companies, Fake ... 424
Oil Field, Geology of the Coalinga ... 4
Oil Field, Katalla, Alaska ... 731
Oil Field, The Maikop ... 468
Oil in Ohio Mines, Paraffine ... 499
Oil in Venezuela ... 441
Oil Land Decision, Important ... 751
Oil Operations in Tampico ... 406
Oil Notes ... 134
Oil, The Testing of Miners' ... 764
Oklahoma, Coal Mining in ... 193
Old Dominion Pumping System ... 324
Ontario, Canadian Iron Ore Industry ... 455
Ore Crusher, The Largest ... 432
Ore Mining Notes...11, 250, 318, 351, 564, 584, 706
Ore Piles, Calculating the Tonnage of ... 158
Ores, Hydrometallurgy of Cobalt ... 303
Organ Feedwater Heater ... 371
O'Toole, Edward, Power Used in Mining ... 86
Outbursts of Carbon Dioxide in French Mines ... 392
Oxygen Blowpipe, The ... 163
Oxygen Helmets Used at Mine Fire ... 161
Oxygen Melting Process, The Menne ... 708
Oxygen Process for Melting Iron ... 146

PACIFIC COAST COAL CO., WORKING A STEEP COAL SEAM ... 77
Paint, Earth Colors of Sardinia ... 472
Palau No. 2 Mine, Explosion at ... 202
Palmer, Leroy A., Mining in Tintic District, 553—South Utah Mine and Mill, 592—Utah Consolidated Aerial Tramway, 150—Utah Metals Co. Tunnel, 296—Yampa Smelter at Bingham ... 14
Panama Canal, Mexican Rival to ... 244
Panama Canal, The ... 241
Panama Exposition, World's ... 213
Panama Meeting, American Institute ... 329
Pan American Mining Co ... 128
Paraffine Oil in Ohio Mines ... 499
Parker, E. W., American Institute Panama Meeting ... 329
Parral, Mexico, Cyanide Plant ... 604
Paving Blocks, English Slag ... 273
Payne, Henry M., American vs. European Coal Mines ... 195
Peat Bogs of Ireland ... 212

Page
Pebble Industry in Newfoundland ... 249
Peck's Shaft Breaker ... 513
Pennsylvania, Anthracite and Bituminous Output ... 485
Pennsylvania's Bituminous Mine Law ... 742
Pennsylvania-Hillside, First-Aid Contest ... 197
Percentage of Grade, To Find ... 101
Perkins, Frank C., A Wire Rope Cutting Machine ... 85
Permissible Explosives ... 206
Perth Amboy Plant, American Smelting and Refining Co ... 603
Petrodynamics, Data of ...210, 505
Petroleum Deposits About Tampico, Mexico ... 50
Petroleum in Alabama, Natural Gas and ... 175
Petroleum, Geography of ... 607
Petroleum Industry, Venezuela ... 154
Petroleum Mining Engineering ... 442
Petroleum, Tayabas ... 697
Phelps, Hartley M., Marianna Coal Mines ... 523
Philippine Gold Mines ... 464
Phosphate in Oceania ... 454
Phosphorus for Igniters, Fulminating vs. White 76
Phosphorus Poisoning ... 693
Pictou Coal Field Lore ... 179
Pilares Mine, Los ... 106
Pillars, Methods of Removing Coal ... 495
Pittsburg-Buffalo Co., Marianna Mine ... 523
Placer Mining Machines, Dry ... 589
Platinum in Vermilion River District, Canada 443
Pneumelectric Machine Co., Trolley Wire Support for Mines ... 32
Pocahontas Coal and Coke Shipments ... 630
Porcupine Gold Fields ... 701
Portable Electric Safety Light Co., Hubbell Lamp ... 127
Porter Co., H. K., Two-Stage Air Locomotives 365
Portland Cement Calculations ... 25
Possibilities of a New Liability Law ... 532
Possible Effect of the Panama Canal on Alaskan Coal Business ... 175
Poverty Gulch Claims, Developing, 727—Prospecting, 694—Report on ... 586
Powder, Black ...165, 225
Powder-Thawing Device ... 640
Power for Concentrating Mill ... 19
Power Gas and Its Development ... 640
Power Plant, A Sectionalized ... 636
Power Production at Collieries ...33, 180
Power to Ventilate a Mine ... 771
Power Used in Mining ... 86
Practical Fuel Values ... 178
Precipitation Process, De Wilde ... 342
Prevention of Mine Accidents ... 611
Prevention of Scale in Boilers ... 719
Priamur Province, Russian Far Eastern Gold Field ... 447
Price-Pancoast Disaster, Lesson of ... 611
Price-Pancoast Disaster, The ... 616
Price-Pancoast Fire, Lessons from the ... 678
Price, Thomas H., Extra Mine Doors ... 216
Principles of Electric Blasting ... 393
Problem of Education, The ... 745
Production of Iron Ore in Spain ... 577
Progress in Land Classification ... 355
Promotion Literature, Mine ... 291
Proof Assay for Copper ... 149
Properties of Concrete ... 636
Proportionality, Limit of ... 349
Proposed Colorado Coal Mine Law ... 482
Proposed Colorado Mine Law ... 487
Prospecting Advice ... 435
Prospecting for Silver ... 289
Prospecting Poverty Gulch Claims ... 694
Prosser, Warren C., Gold Deposits of San Juan, Colo ... 335
Prosser, W. T., Katalla, Alaska, Oil Field ... 731
Protective Alkalinity, Testing for ... 13
Protective Alkalinity, Titrating for ... 479
Protection of Surface Over Coal Mines ... 610
Protective Value of Humidity ... 188
Pulp, Specific Gravity Estimation of ... 715
Pumping and Siphoning Hot Water ... 663
Pumping at Bisbee, Ariz ... 132
Pumping ... 37
Pumping System, Old Dominion ... 324

Page

Pumps, Electrically Driven 288
Putting Out a Mine Fire 101

QUENNER DRY PULVERIZER 229
Quintana Roo, Mexican Territory of 425

RADIANT MINE, ROPE THROW AT 752
Radium and Radioactivity 732
Radium in Austria 6
Ragged Chutes Air Compressor Plant 129
Ragged Chutes, Hydraulic Compressor, Air from 561
Rail Bonding in Mines 673
Railroad Rates 101
Raymond's Resignation, Dr. R. W. 588
Reade, Frederick, Titrating for Protective Alkalinity 479
Reading Coal and Iron Co., First-Aid Contest 197
Reclaiming Zinc Lead Fines 131
Recollections of William Smurthwaite 165
Red Cross, First-Aid Contest 197
Redpath, C. H., The Tooele Smelter 321
Reeder, E. C., Sectional Slag Pot 149
Report of American Zinc, Lead and Smelting Co. 293
Report of Coal Mine Inspector of Colorado 665
Report on the Poverty Gulch Mine 586
Rescue Apparatus, Aerolith 521
Rescue Apparatus, New Breathing Apparatus 759
Rescue Station, Bureau of Mines 68
Rescue Station Commission, Illinois 609
Rescue Station, Foreman of Mine 160
Rescue Stations in Illinois 214
Rescue Work, Mine 184
Resignation, Dr. R. W. Raymond's 588
Restoring Dredged Ground 422
Retort Coke Ovens in Mexico 257
Reverberatory Copper Smelting 557
Reybold, H. W., Electric Motors for Rope Haulage 174
Reynolds, Sim, Coal Trusts and Safe Mining (?), 633—The Moral of Starkville, 391—Possibilities of a New Liability Law 532
Rhea, Vincent, Rail Bonding in Mines 673
Rice, George S., Mine Accident Investigations 282
Rightmire, R. E., Commercial Classification of Fuels 397
Rio Blanco County, Colo. 328
Roane Iron Co., Gasoline Motor Haulage 629
Robb, Edward M., Los Pilares Mine 106
Roberts, Peter, Teaching English to Foreigners 60
Robinson, A. E., Room-and-Pillar Method 88
Rock Drill, History of the 18
Rock Handling Arrangement at Globe Mine 753
Rocky Mountain Fuel Co., Colliery Survey Notes 61
Rolling Friction 683
Rolls, Rule for Figuring Capacity 468
Room-and-Pillar Method 88
Rope Haulage 563
Rope, Sag and Stretch of Power Transmission 228
Rope Strains, Haulage 39
Rope Throw at Radiant Mine 752
Ross, Clyde P., Modern Holmen Coaling Stations 639
Rowe, C. E., Determination of Latitude 119
Rowe, J. P., Mines of Missoula County, Mont. 581
Roy, Andrew, Recollections of William Smurthwaite 165
Ruby, The 319
Rule for Figuring Capacity of Rolls 468
Rules for Handling Kanite 681
Rules for Mine Foremen 293
Russia, Maikop Oil Field 468
Russian Far Eastern Gold Field 447
Rust-Proof Finish for Iron 225
Rutledge, J. J., Mine Accident Prevention 276

SAFETY IN MINES 562
Safety Lamps on Battleships 535
"Safety the First Consideration" 743
Safe Use of Electricity in Gaseous Mines 125
Sagamore Mine, Buffalo-Susquehanna 645
Sag and Stretch of Power Transmission Rope 228
Sale, A. J., Scientific Study of Copper Deposits 684
Sampler, Automatic Coal (Statt) 85

Page

Sampler, Mechanical Coal 169
Sampling Coal 98
Sampling Coal and Coke 89, 190, 209
San Domingo, Mining in 745
San Juan, Colo., Gold Deposits of 335
Santiago, Cuba, Iron Ores of 245
Santo Domingo, Mining Laws in 580
Sardinia, Earth Colors of 472
Saunders, W. L., History of the Rock Drill 18
Savage, T. E., Geology of Herrin Quadrangle 527
Scandium 478
Scientific Study of Copper Deposits 684
Scott, C. E., Testing of Miners' Oil 764
Scott, S. M., Jr., Combination Assay of Copper Bullion 240
Scranton Mine Cave Commission, Mine Support Tests 749
Scranton Mine Cave Enquiry 620
Screen, Capacity of Revolving 181
Screen, The Franz 126
Screens, Standard Grades and 345
Seager, James A., Electric Air Compressor 263
Seamon, W. H., Chemistry of the Cyanide Process 689
Sectionalized Power Plant 636
Sectional Slag Pot 149
Self-Dumping Car Hauls 536
Selling Zinc Ore on Contract 550
Servus Rescue Equipment Co., New Breathing Apparatus, 759—Smoke Helmet for Mine Fire Fighting 281
Settler or Dewaterer, Slime 339
Shaft Construction, Steel Mine 516
Shaft, Cost of Sinking a Mexican Mine 275
Shaft Pillar, Size of 615
Shafts, Elliptical *vs.* Rectangular 167
"Sheelite" in Nova Scotia 665
Sheridan, Jo. E., Coal Mines of Dawson, New Mexico 653
Shock in Mines, Electric 493, 543, 637
Shot Towers 68
Shrinkage of Clay Under London 615
Shumway, Ralph, Colliery Survey Notes 61
Shurick, A. T., Mining at Lethridge, Alberta 635
Signal to Shaft Bottom, Engine and Weigh-Room, Electric 353
Silicon Steel 552
Silver, Prospecting for 289
Simmons Co., John, Tent and Shack Lighting Outfit 33
Simple Hydraulic Hoist, A 157
Simultaneous Inventions 3
Sinking a Mexican Mine Shaft, Cost of 275
Siphon 101
Siphoning Hot Water, Pumping and 663
Siphon in Mining, The 427
Sirocco Fan, Test of Double-Inlet 541
Size of Shaft Pillar 615
Skinner, Robert P., International Cyanide Trust 568
Slack Bin Fire, Fernie 293
Slag Paving Blocks, English 273
Slag Pot, Sectional 149
Slags 152
Slavery, Legalized 354
Sleuthing, Metallurgical 476
Slime Settler or Dewaterer 339
Sluicing in Tasmania, Tin 309
Small Air Compressors at Mines 477
Smelter and Lead Refinery, Trail 121
Smelter at Bingham, Yampa 14
Smelter Fume Abatement, Kenhett 143
Smelters, Cobalt Ore Buying by 705
Smelter, The Tooele 321
Smeltery, Webb City Lead 709
Smelting Companies, Competition Among 293
Smelting, Reverberatory Copper 557
Smith, Sion B., Liability for Industrial Accidents 501
Smurthwaite, Recollections of William 165
Society Meetings 286, 734
Societies:
American Institute of Electrical Engineers, June Meeting, 708—American Institute of Mining Engineers, June Meeting, 708—American Institute of Mining Engineers, One Hundredth Meeting, 754—American Institute of Mining Engineers, Panama Meeting, 286, 329—American Mining Congress, September, 1910, Meeting, 105, 231—American Mining Congress, 226, 615—American Society of Engineer Draftsmen, 105—American Society of Engineering Contractors, September, 1910, Meeting, 189—American Society of Mechanical Engineers, 549, 708—Appalachian Engineering Association, June, 1911, Meeting, 734—Canadian Mining Institute, 286, 549—Canadian Mining Institute Meeting, March, 1911, 596—Coal Mining Institute of America, 286—Coal Mining Institute of America, December, 1910, Meeting, 407—Coal Mining Institute of America, June Meeting, 38, 708—Kentucky Mining Association, 615—Massachusetts Institute of Technology Anniversary, 549—Mine Inspectors' Institute, June, 1910, Meeting, 38—National Civic Federation, 353—National Gas and Gasoline Engine Trades Association, 549—West Virginia Coal Mining Institute, 286, 615—West Virginia Mining Association, December, 1910, 417—West Virginia Mining Institute, December, 1910, Meeting 356

Page

Sodium Carbonate as an Oxidizer 704
Solubility of Gold in Potassium Ferrocyanide 477
Solubility of Rare Gases in Water 552
South Africa, Diamonds in German 426
South African Mining Regulation Commission, Report of 337
Southern Gold Mining 98
Southern Kansas Coal District 668
South Utah Mine and Mill 592
Spain, Mining in 429
Spain, Production of Iron Ore in 577
Spaulding, Charles F., Rules for Figuring Capacity of Rolls 468
Specific Gravity Estimation of Pulp 715
Specific Gravity Tests, The Acid 62
Spines, First-Aid for Injured 538
Spitzbergen Coal Mining 101
St. Lawrence Power Project 120
Stables, Underground, Care of Mine Mules 650
Stadler, H., Grading Analyses 344
Stag-Cañon Fuel Co., Coal Mines of Dawson, New Mexico 653
Standard English 99
Standard Grades and Screens 345
Standardization of Bullion Assays, Report of Committee to Institution of Mining and Metallurgy 690
Starkville—An Explanation 355
Starkville, Colo., Explosion 261
Starkville, The Moral of 391
Station, Durable Triangulation 23
Steam in the Air-Current 486
Steel Mine Shaft Construction 516
Steel Mine Supports, Interlocking 664
Steel, Silicon 552
Steels, Stoping-Drill 717
Steel Supports in Coal Mines 387
Steel Supports in Mines 563
Steel, Vanadium 334
Stephens, J. R., Method of Laying Off a Curve 165
Stoek, H. H., Illinois Coal Field 54
Stokers and Hand Firing, Mechanical 42
Stop Blocks and Derailing Switches 753
Stopes, Filling 465
Stope Timbering 29
Stoping Drill, Drifting With a 697
Stoping-Drill Steels 717
Storage Plant, Bunker Hill Coal 166
Storing Coal 745
Stow, Audley H., Correlation of Thacker Field 83
Strains, Haulage Rope 39
Stripping Coal Beds 69
Strontium, Metallic 703
Submarine Divers in Mines 446
Sullivan Machinery Co., Drifting With a Stoping Drill, 697—Small Air Compressors at Mines 477
Sulman, H. L., Titrating for Protective Alkalinity 479
Sundt, F. A., Concentration at Calama, Chile 605

Page
Superior and Boston Mine.................. 112
Supplies, Handling Mine.................. 267
Support of Surface, Report of Scranton Mine Cave Commission.................. 620
Supports in Coal Mines, Steel.................. 387
Surface Alterations of Gold Ores.................. 687
Surface Over Coal Mines, Protection of.................. 610
Surface Support, Report of Scranton Mine Cave Commission.................. 620
Surveying..................37, 166
Survey Notes, Colliery.................. 61
Swedish Pig Iron Production, 1909.................. 126
Swedish Steel Rails.................. 372
Sylvester, Geo. E., Gasoline Motor Haulage.... 629

TAILINGS, UTILIZING ZINC.................. 601
Talc, Tariff on Sawed.................. 111
Tampico, Mex., Petroleum Deposits About.... 50
Tampico, Oil Operations in.................. 406
Tanks, Cement Concrete Vats and.................. 364
Tariff on Alloy of Iron and Cerium.................. 23
Tariff on Breccia.................. 380
Tariff on Precious Stone.................. 404
Tariff on Sawed Talc.................. 111
Tarr, Raymond P., Compensation for Injury.. 410
Tasmania, Tin Sluicing in.................. 309
Tayabas Petroleum.................. 697
Taylor Air Compressor.................. 129
Taylor Air Compressor, An Accidental.................. 458
Taylor Concrete Breaker.................. 272
Teaching English to Foreigners.................. 60
Teaching Foreigners English.................. 35
Technical Societies.................. 615
Technical Supply Co., Catalog of Books Worth While.................. 612
Tehauntepec, Mexican Rival to Panama Canal 244
Telephone Train Dispatching.................. 308
Temiskaming Mine, Cobalt.................. 331
Tent and Shack Lighting Outfit.................. 33
Tepic Territory, Mexico.................. 331
Terry Core Drill Co., Cost of Core Boring...... 323
Testing for Protective Alkalinity.................. 13
Testing of Miners' Oil, The.................. 764
Test of Double-Inlet Sirocco Fan.................. 541
Tests for Carbon Monoxide.................. 33
Tests for Copper Minerals.................. 430
Tests, Mine Support.................. 749
Texas Lignite.................. 284
Thacker Field, Correlation of.................. 83
Thawing Device, Powder.................. 640
Thermochemistry of Anthracite..................37, 166
Thesis Prize, Colorado School of Mines.................. 35
Thesis, Report on Poverty Gulch Mine, 586, ...694, 727
Thomson, Francis A., Steel Supports in Mines.. 563
Thomas, Willis P., Coal Mining in Oklahoma.. 193
Three-Wire Mine Service.................. 402
Tile-Laying Machine, A New.................. 187
Tile-Laying Machine, Falcone Oven.................. 287
Timbering in Indian Coal Mines.................. 179
Timbering in the Joplin District.................. 144
Timbering, Stope.................. 29
Tin Mining in Bolivia, American.................. 568
Tin Mining in China.................. 131
Tin Sluicing in Tasmania.................. 309
Tintic District, Utah, Mining in.................. 553
Titrating for Protective Alkalinity.................. 479
To Friends.................. 34
Tonnage of Ore Piles, Calculating.................. 158
Tooele Smelter.................. 321
To Prevent Dusting of Concrete Floors.................. 500
Trail Smelter and Lead Refinery.................. 121
Tramway at Del Carmen Zinc Mine.................. 437
Tramway, Utah Consolidated Aerial.................. 150
Transmission, Losses in Electric.................. 53
Transportation, Coal Mine.................. 408
Transportation in Alaska.................. 409
Transportation System, New.................. 39
Transvaal Drill Competition, 1909.................. 459
Treatment of Cobalt Ores.................. 693
Trenching, Value of Surface.................. 686
Trenton Iron Co., Rope Haulage.................. 563
Triangulation Station, Durable.................. 23
Trolley Wire Support for Mines.................. 32
Trusts and Safe Mining, Coal (?).................. 633
Tungsten Ore in Washington.................. 307

Page
Tunnel Driving, Explosives for.................. 159
Tunnel Driving, Fast.................. 9
Tunneling on Los Angeles Aqueduct.................. 135
Tunnel, The Elizabeth.................. 102
Tunnel, Utah Metals Company.................. 296
Turboblowers and Compressors.................. 285
Turnbull, J. M., Trail Smelter and Lead Refinery.................. 121
Turner, R. J., Consolidated Fuel Co.................. 385
Tuttle, E. G., El Cobre Mines.................. 449
Two-Stage Air Locomotives.................. 365

U. S. BUREAU OF MINES, THE.................. 79
United Mine Workers' Expense Account 504
United States Coal and Coke Co., Description of Plant.................. 57
United States Coal Production, 1910.................. 417
University of Illinois, Apparatus Illustrating Method of Charging Holes.................. 44
University of Washington, Mines Building.... 426
Uruguay, Coal Discoveries in.................. 485
Uruguay Manganese Deposits.................. 286
Use Care in Purchase of Brattice Cloth.................. 44
Utah Consolidated Aerial Tramway.................. 150
Utah Metals Co., Tunnel.................. 296
Utah Mine and Mill, The South.................. 592
Utah, Mining in the Tintic District.................. 553
Utilizing Zinc Tailings.................. 601

VALDEZ CREEK PLACER DISTRICT.... 572
Value of Surface Trenching.................. 686
Vanadium in Cast Iron.................. 716
Vanadium Steel..................234, 334
Vegetation, Carbon Dioxide and.................. 163
Venezuela, Oil in.................. 441
Venezuela, Petroleum Industry.................. 158
Ventilation.................. 101
Ventilation, Metal Mine..................19, 337
Vermilion Making.................. 161
Verner, John, Coal-Dust Explosions.................. 623
Volume of Air Delivered by Fan.................. 486
Volumetric Capacity of a Fan.................. 416

WABANA IRON MINES, NEWFOUNDLAND.................. 569
Wagner, John L., Trolley Wire Support for Mines.................. 32
Wagner, Walter C., Electricity in Mining..... 756
Walker, Sydney F., Electric Shock in Mines, ...493, 543, 637
Walker, William H., The Alchemist in Modern Industry.................. 688
Wanviz, John D., Durable Triangulation Station.................. 23
Ward, Morris R., The Ruby.................. 319
Warning to Coal Men, A.................. 672
Warnings, Colliery.................. 482
Warnings to Colliers.................. 562
Washed His Home for Gold.................. 677
Washeries, Georgia Brown Iron Ore.................. 294
Washington, Compensation for Injury in, 410—Examination of Mine Officials, 741—First-Aid Contest in, 545—Tungsten Ore in...... 307
Water at Coleman Shaft, Bailing.................. 631
Water, Lamp Burning in..................37, 166
Waters, Surface Alterations of Gold Ores..... 687
Water Vapor Be Resolved Into Its Gases in an Explosion, Can.................. 225
Webb City Lead Smeltery.................. 709
Weight of Fulminate of Mercury.................. 59
Weldin, William Archie, Elliptical vs. Rectangular Shafts.................. 167
Wellman-Seaver-Morgan Co., Hecla Mine Hoist 28
Wells, Mining Around Gas.................. 486
Western Electric Co.................. 128
Westinghouse Electric & Mfg. Co., Mine Hecla Hoist.................. 28
West Virginia Institute Meeting.................. 356
West Virginia Mining Association.................. 417
West Virginia, The Mine Law of.................. 370
Wet Assay of Gold..................143, 423
Wetherill Magnetic Separator.................. 462
Whaley, William, Coal-Loading Machine...... 206
Whitcomb Co., Geo. D., Gasoline Motor Haulage.................. 629
White, George W., Howell-White Furnace.... 225

Page
Wilhelm, U. H., Bonanza Copper Mine, Alaska 441
Williams, Percival S., Mining on the Gogebic Range.................. 712
Williams, R. Y., Rescue Stations in Illinois.... 214
Wilson, E. B., Coal Mine Transportation, 408—Cuban Gold Mines, 240—Dry Placer Mining Machines, 589—Evolution of Hoisting, 153, 251, 298, 358, 444—Evolution of Mine Haulage, 45, 71—Hydraulic Air Compression (Ragged Chutes), 129—Hydrometallurgy of Cobalt Ores, 303—Iron Ores of Santiago, 245—Kennett Smelter Fume Abatement, 143—Marting, W. Va., Plant, 171—Metallurgical Sleuthing, 476—Peck's Shaft Breaker, 513—Retort Coke Ovens in Mexico, 257—Reverberatory Copper Smelting, 557—Taylor Concrete Breaker.................. 272
Wind Rock Coal Mine.................. 65
Wind Rock Mine, Coal-Loading Machine...... 206
Wire Rope Cutting Machine.................. 85
Wiring, Three-Wire Mine Service.................. 402
Wisconsin Zinc Ores.................. 480
Witteveen, G., De Wilde Precipitation Process 342
Wittich, L. L., Joplin District Zinc and Lead Ores, 31—Joplin Zinc and Lead Production, 1910, 435—Log Washer in Zinc Mining, 423—Reclaiming Zinc Lead Fines, 131—Selling Zinc Ore on Contract, 550—Southern Kansas Coal District, 668—The Year in the Joplin District, 352—Timbering in the Joplin District, 144—Utilizing Zinc Tailings, 601—Webb City Lead Smeltery, 709—Zinc and Lead in Arkansas.................. 10
Wolf, August, Northwestern Ores, 434—Tungsten Ore in Washington.................. 307
Wolframite in Argentina.................. 432
Wolframite, Magnetic Separation of.................. 462
Woodworth, R. B., Steel Mine Shaft Construction, 516—Steel Supports in Coal Mines, 387—Steel Supports in Mines.................. 563
Wooley, Monroe, Philippine Gold Mines...... 464
Working a Steep Coal Seam.................. 77
World's Panama Exposition.................. 213
World's Production of Zinc in 1910.................. 596

YAMPA SMELTER AT BINGHAM.................. 14
Year in the Joplin District, The.................. 352
Yellow Pine, Nev., Zinc Mining at.................. 340
Yerington District.................. 24
Young, C. M., Coal-Dust Explosions.................. 746
Yukon Placer Yield, 1909.................. 147
Yukon Territory of Canada.................. 463

ZINC AND LEAD IN ARKANSAS.................. 10
Zinc and Lead Ores, Joplin District...31, 327
Zinc and Lead Production, Joplin, 1910....... 435
Zinc Deposits, Boquillas.................. 479
Zinc Deposits, Leadville, Colo.................. 436
Zinc in 1910, World's Production of.................. 596
Zinc Lead Fines, Reclaiming.................. 131
Zinc-Lead Pigment Plant, Blair's.................. 698
Zinc Mine, Del Carmen.................. 437
Zinc Mining at Yellow Pine, Nev.................. 340
Zinc Mining, Log Washer in.................. 423
Zinc Ore Notes.................. 431
Zinc Ore Notes, Joplin.................. 472
Zinc Ore on Contract, Selling.................. 550
Zinc Ores, Wisconsin.................. 480
Zinc Smelting, Electric Furnace for.................. 703
Zinc Tailings, Utilizing.................. 601

Index B—Book Reviews

ABSTRACTS OF REPORTS OF FRENCH COAL DUST EXPERIMENTS.................. 421
Accidents from Gas in Coal Mines of Belgium, 1891 to 1909.................. 36
Act to Consolidate and Amend the Coal Mines Regulation Act, British Columbia.................. 549
Address of Neil Robinson to West Virginia Mining Association.................. 613
Alpha Portland Cement Co.................. 164
American Red-Cross Abridged Textbook on First-Aid.................. 420
Annales Des Mines de Belgique.................. 36

Page

Annual Report of Minister of Mines, British Columbia, 1909 164
Annual Report of Water Supply Commission of Pennsylvania 164
Appalachian Mine Record 36
A Study in Heat Transmission 292

BADLAND FORMATION OF BLACK HILLS REGION 681
Biennial Report of State Geologist of Missouri 612
British Columbia Bureau of Mines, Bulletin No. 2, 1910 421
Bulletin of Bureau of Labor, November, 1910 612
Bulletin of the Engineer of Mines of Peru, No. 75 36
Bulletins of Massachusetts Institute of Technology 421
Bulletin 2, 1911, West Virginia Geological Survey, Levels and Coal Analyses 548
Bureau of Mines 744
Bureau of Mines Publications 681
Bureau of Mines, Washington, Bulletin 292
Bureau of Mines, Washington, D. C. 485
Bureau of Standards, Washington, Circular 292
Business Prospects Year Book, 1911 356

CANADA BUREAU OF MINES, REPORT FOR 1910 421
Canada Department of Mines, Production of Coal and Coke, 1909 421
Canada Department of Mines 292
Canada Department of Mines, Iron Ore Deposits of Bristol Mines 164
Canada Department of Mines, Report 1909, 228, 485
Carnegie Foundation for Advancement of Teaching, Report 1910 613
Cement Age and Concrete Engineering 484
Child Labor Legislation in Europe, Department of Commerce and Labor 421
Chrysotile Asbestos 613
Clays and Clay Industry of Washington 485
Coal Cutting With an Electric Puncher 100
Coal Mines Regulation Act, Victoria, B. C., 1911 613
Coal Mining in Arkansas, Part I 744
Coal Report Illinois Bureau of Labor Statistics, ... 28, 292
Colliery Guardian Co., Abstracts of American and Austrian Coal Dust Experiments 548
Colorado Geological Survey 228
Commemorative Bulletin, Ohio State University 292
Compressed Air Plant 291
Contributions to the Economic Geology of the Republic of Argentina 681
Corrosion and Preservation of Iron and Steel .. 484
Cost of Mining 36
Crop Statistics, Bureau of Labor, Nebraska ... 421
Cyanide Handbook, The 35

DEPARTMENT OF COMMERCE AND LABOR 485
Department of Geology and Natural Resources, Indiana, Report, 1909 485
Department of Mines, New South Wales, Australia, Report 1909 228
Department of Mines, Ottawa, Canada, Mineral Production for 1910 681
Department of Mines, Ottawa, Canada, Report 1910 613
Department of Mines, Victoria, Australia, 1909 228
Dictionary of Metallurgical and Chemical Material 164

ECONOMIC GEOLOGY, THIRD EDITION 227
Electric Circuit Problems 483
Elementary Chemistry for Coal Mining Students 612
Elementary Crystallography 420, 484
Elements of Geology 612
Esperanto Grammar 227

FARMERS' HANDBOOK OF EXPLOSIVES 485, 549
Features of Pro[illegible] Power Plant Development in Europe 548
Final Re[illegible] Commission, Tra[illegible] 228
Forestry [illegible] 5, West Virginia Geo[illegible] 612
Friction in [illegible] 292

GEOGRAPHICAL REPORT ON THE FRANZ JOSEF GLACIER 681
Geology of Iron River District, Michigan 584
Geological Map of Cobalt 356
Geological Notes, Inspector of Mines, Norway. 421
Geological Survey of Georgia, Public Roads of Georgia 164
Geological Survey of Michigan, 1908, Report .. 228
Geological Survey of Ohio 681
Glimpse of Utah 164
Gold Dredging in California 355
Gold Hill Mining District of North Carolina ... 36
Gold Mine in Tennessee Clays, A 420
Guide to Technical Writing 680
Gurley's Manual 420

HEATON'S HANDBOOK FOR 1911 484
High Efficiency Centrifugal Pump 355
How to Operate a Pneumelectric Coal Puncher 485
Hurd's Iron Ore Manual 744

ILLINOIS BUREAU OF LABOR STATISTICS COAL REPORT FOR 1909 228
Iowa Geological Survey, Volume 20 613
Irrigated Lands of the State of Washington ... 421

JOURNAL OF THE IRON AND STEEL INSTITUTE, VOL. LXXXI 228

LOGICAL POINT, THE 421

MAGNETIC PROPERTIES OF HEUSLER ALLOY 744
Manual for Assayers and Chemists 420, 484
Manual of Practical Assaying, Furman 356
Map of Cobalt-Nickel-Arsenic-Silver Area 292
Map of Mineral Claims in Orient Mining District 164
Map of West Virginia 100
Map of Wood, Ritchie, and Pleasants Counties Oil and Gas Fields of West Virginia 549
Map of Whittier-Olenda Oil Fields, California .. 549
Maryland Geological Survey, Vols. 7 and 8 36
Mechanical World's Pocket Diary, 1911 484
Mechanical World's Electrical Pocketbook, 1911 484
Metal Statistics, 1911 484
Meteorite Studies 292
Michigan Geological Survey, Monroe Formation of Southern Michigan 485
Mine and Quarry for October 292
Mineral Industries of Canada 681
Mineralogy of Arizona 484
Mineral Resources of Philippine Islands ... 292, 421
Mineral Production of Ontario, 1910 .... 613, 681
Mining Laws of Ohio, 1910 36
Minnesota Iron Ranges 228
Modern Method of Pumping 292
More Recent Cyanide Practice 420
Motion Study 612

NATURAL RESOURCES OF COLORADO, UTAH, AND NEW MEXICO 164
New York State Museum, Bulletin 228
New Zealand Geological Survey, Bulletin 10 ... 485
New Zealand Geological Survey, Bulletin No. 11 421
New Zealand Geological Survey 228
North Carolina Geological and Economic Survey 164, 228, 292, 681

OIL RESOURCES OF CALIFORNIA 549
Ontario & Western Railway Co., Report of President, 1910 228
Ore Mining Methods 292

PAINT MANUFACTURERS' ASSOCIATION OF THE UNITED STATES .. 164, 228
Pennsylvania Department of Mines Report, Part II, Bituminous, 1909 228
Pennsylvania State College, Bulletin No. 4 421
Permissible Explosives, Miners Circular No. 2 .. 549
Philippine Journal of Science 228
Pocketbook of Mechanical Engineering 484
Practical Data for the Cyanide Plant 227
Practical Mineralogy Simplified 548
Practical Shaft Sinking 292
Practical Stamp Milling and Amalgamation ... 420
Preliminary Review and Estimate of Mineral Production of Mines, Victoria, B. C. 549
Preliminary Report of Oil and Gas Developments of Tennessee 549
Prevention of Coal Dust Explosions by Zone Systems 549
Principles of Chemical Geology 100
Proceedings and Collections of Wyoming Historical and Geological Society, Vol. 11 .. 612, 681
Proceedings of Colorado Scientific Society 228
Proceedings of Engineering Association of N. S. W., 1908-9 100

QUELQUES MOTS SUR LA QUESTION DES POUSSIRES AU CONGRESS DE DUSSELDORF, 1910. 228

RECORD OF FIRST SERIES (1908-09) BRITISH COAL DUST EXPERIMENTS 421
Register School of Mines, New Mexico, 1911 ... 681
Report of Bureau of Mines, Colorado, 1910 ... 681
Report of Bureau of Mines, Ontario, Canada, 1910 292
Report of Chief Inspector of Mines of Ohio, 1910 421
Report of Cherry Mine Disaster, State of Illinois, Bureau of Labor 421
Report of Colorado State School of Mines 421
Report of Committee Transvaal Stope Drill Competition 485
Report of Consolidated Mercur Gold Mines Co .. 292
Report of Department of Mines, Pennsylvania, Parts 1 and 2, 1909 421
Report of Geological Survey, State of Washington, 1909-11 421
Report of Government Mining Engineer, Transvaal, 1910 681
Report of Mining Industry of Natal 1909 421
Report of Minister of Mines, New Zealand 421
Report on Mineral Production of Canada, 1910. 613
Report of Mining Department of Tennessee 485
Report of Mining Investigation Commission, Illinois 613
Report of Mine Rescue Station Commission, Illinois, 1910 613
Report of Secretary of Internal Affairs, Harrisburg, Pa. 421
Report of Smithsonian Institution, 1909 548
Report of State Geologist, New Jersey, 1909 .. 292
Report of State Geologist of North Carolina, 1910 613
Report of State Mine Inspectors of Iowa, 1910 421
Report of Superintendent of Coast and Geodetic Survey, 1910 681
Report of Tests of Babcock & Wilcox Boiler, Battleships Wyoming and Arkansas 421
Review of the Joplin District Folio 681
Rock Drills 420

SCHOOL OF MINES AND METALLURGY, UNIVERSITY OF MISSOURI, BULLETIN 1910 484
Scope and Progress of Mining Industry in Colorado 292
Smoley's Tables 291
Some Relations Between the Composition of a Mineral and Its Physical Properties 549
State Geological Survey Bulletin No. 16, Illinois 613

TABLES FOR DETERMINATION OF MINERALS 680
Tables for the Determination of Minerals by Physical Properties 36
Temperature-Entropy Diagram 744
Tennessee State Geological Survey Report 681
Tennessee State Geological Survey 164
Tennessee White Paint Tests 421
Testing for Metallurgical Processes 356
The Coal Trade 680
The Logical Point (Panama Exposition) 228
Twentieth Century Sheet-Metal Worker 227
Twenty-five Years of Engine Building 485

UNITED STATES DEPARTMENT OF AGRICULTURE, FORESTS OF ALASKA 164
United States Geological Survey Press Bulletins 744

	Page
United States Geological Survey Publications, ...36, 164, 227, 292, 420, 485, 549, 612,	681
University of Idaho, Catalog 1909 and 1910..	228
University of Illinois, Bulletin No. 42........	421
University of Illinois, Bulletin 43, Freight Train Resistance..............................	485
University of Missouri......................	164
University of Pittsburg, Cooperative System..	164
VOLATILE MATTER OF COAL, THE....	355
WATER-JACKET FURNACE, A..........	164
Washington Geological Survey, Bulletin No. 1..............................	421
Who's Who in Mining and Metallurgy, 1910...	100
West Virginia Geological Survey Reports of Pleasants, Wood, and Ritchie Counties.....	36
Wisconsin Engineer, The..................	485

Index C—Personals and Obituaries

	Page
ADAMS, GEORGE I......................	726
Anderson, J. C.........................	667
Armstrong, Charles G....................	390
Armstrong, Francis J....................	390
Ashworth, James.........................	667
BAILEY, A. C..........................	343
Baker, Charles W........................	273
Balmey, Luther W........................	726
Bart, John..............................	726
Bauder, W. R............................	549
Bellis, Thomas (Obit.)..................	352
Bement, A...............................	273
Blick, James (Obit.)....................	486
Boardman, H. P..........................	161
Bond, G. B. (Obit.).....................	744
Borie, A. E.............................	218
Boyle, P. M. (Obit.)....................	744
Brinton, Walter.........................	343
Brown, A. R.............................	667
Burbidge, Frederick.....................	105
Bush, B. F..............................	667
CALVIN, SAMUEL (OBIT.)................	652
Caraway, Dr.............................	667
Clagett, Thomas H.......................	463
Cobleigh, Henry R.......................	667
Cole, David.............................	463
Conner, Eli T...........................	218
Craig, W. R.............................	622
Cuntz, William C........................	273
Cushman, Allerton L.....................	343
DANIELS, JAMES B. (OBIT.).............	614
Darton, N. H............................	390
Davies, Ralph E.........................	549
Davis, D. D.............................	667
De Bardeleben, Henry F..................	463
De Bardeleben, Henry Fairchild (Obit.)....	352
Delano, Frederick A.....................	273
Dennison, Charles T. (Obit.)............	545
Dietz, J. H.............................	50
Draper, Marshall D......................	161
Duck, George F..........................	273
EMMONS, SAMUEL F. (OBIT.)............	614
English, Thomas.........................	343
Enzian, Charles.........................	218
Epperson, James.........................	549
Evans, Joseph E. (Obit.)................	614
Evans, William A........................	218
FRASER, NORMAN........................	726
Frost, Harwood..........................	726
GAMBILL, G. T.........................	667
Gardner, Henry A........................	343
Garrison, F. Lynwood....................	622
George, H. C..........................105,	463
Gross, John..........................161,	273
HABERSTICK, PETER.....................	390
Hamilton, H. H..........................	667
Hammond, Reese..........................	726
Herzig, C. S............................	390
Hitchcock, C. K.........................	105
Hoffmann, Ross..........................	105
Holbrook, W. R..........................	667
Holden, E. C............................	105
Holloway, George T......................	105
Hopkins, N. Monroe......................	343
Huntoon, Prof. Louis D..................	667
Hutchins, J. P.......................50,	105
JENKINS, W. J.........................	667
Jessup, A. B............................	726
Jones, J. F.............................	105
Jones, W. J.............................	622
KANE, JOHN I..........................	622
Key, J. W...............................	667
Knight, Walter (Obit.)..................	614
Knipp, C. T.............................	273
Koester, Frank......................161,	549
LAKES, ARTHUR.........................	218
Lambert, Thomas E.......................	105
Lancaster, Henry M......................	549
Langhans, Arthur........................	390
Lathrope, W. A..........................	390
Lincoln, John J.........................	726
Locke, Edward G.....................390,	463
Lord, Nathaniel W. (Obit.)..............	744
Ludlow, Edwin...........................	726
MAIZE, RICHARD........................	390
McCrorie, Harry.........................	667
McGough, Miss Kathryn...................	463
McKay, Hood.............................	726
Meadows, Bert M.........................	105
Moffatt, David H. (Obit.)...............	614
Moore, F. Cushing.......................	105
NEWBY, J. W...........................	667
Norman, Sidney..........................	549
O'NEALE, M. L.........................	726
Osborn, Dr. Harold......................	390
PARKER, H. C..........................	463
Parsons, Lewis A........................	622
Payne, Edward F. (Obit.)................	318
Perry, John R. (Obit.)..................	614
Pierce, Walter L. (Obit.)...............	352
Prior, Wright S. (Obit.)................	82
Probert, Frank H........................	463
RAMSEY, ERSKINE.......................	726
Ramsay, George Sharp (Obit.)............	545
Rice, G. S..............................	726
Roberts, Warren R.......................	549
Rogers, A. P............................	105
Rountree, W. S..........................	667
Routenlusch, William....................	667
Rowe, Fremont S.........................	105
Russell, James M........................	622
Rutledge, J. J......................218,	667
Ryan, Daniel............................	667
SAUNDERS, WM. L.......................	667
Scott, George H.........................	549
Seaman, A. E............................	105
Seaver, Kenneth.........................	273
Shaw, John..............................	343
Shelby, Charles Francis (Obit.).........	463
Shields, H. W...........................	463
Shields, Dr. M. J.......................	218
Shroter, George A.......................	726
Smurthwaite, William (Obit.)............	82
Soper, Ellis............................	273
Stephens, K. B..........................	105
Stines, Norman..........................	105
Stoek, H. H.............................	273
Sylvester, George E.....................	549
TAYLOR, HARRY N.......................	390
Thompson, Heber S. (Obit.)..........545,	613
Thornby E. M............................	726
Torrey, H. G............................	463
Tucker, Charles H. (Obit.)..............	352
Tuttle, Edgar G...................549, 50,	105
Tyrrell, J. B.............50, 105, 218, 390,	726
VAN WINKLE, C. T......................	667
WALKER, SYDNEY F......................	218
Warren, H. M............................	726
Watson, Charles.........................	343
Watt, Stewart (Obit.)...................	352
Weed, Walter Harvey.....................	463
Welch, J. C.............................	726
Weldon, J. D............................	667
Wiley, W. Murdoch.......................	390
Wilke, Charles..........................	161
Woakes, Ernest Russell..................	50
Wren, Mr................................	549
YODER L. E............................	50
ZIMMERMAN, W. F.......................	218

Index D—Catalogs Received

ABENAQUE MACHINE WORKS..........	772
Ackroyd & Best, Ltd................43,	256
Aldrich Pump Department...........281, 357,	401
Allis-Chalmers Co.........357, 512, 628, 671,	772
Aloe Co., A. S..........................	256
Alteneder & Sons, Theo..................	671
American Asphaltum and Rubber Co.........	671
American Blower Co................43, 256,	671
American LaFrance Fire Engine Co..........	160
Austin Separator Co.....................	357
American Ship Windlass Co................	357
American Spiral Pipe Works...............	544
BALDWIN LOCOMOTIVE WORKS...281,	401
Barco Brass and Joint Co................	772
Bartley Crucible Co., Jonathan...........	256
Betterman, Reinhold.....................	772
Bristol Co.........44, 281, 357, 401, 512, 628,	671
Buettner & Shelburne Machine Co..........	43
Buffalo Forge Co........................	671
CARB-OX CO., THE......................	160
Carnegie Steel Co. 160, 544,	671
Christy Box Car Loader Co................	401
Chicago Pneumatic Tool Co...............281,	401
Colorado Iron Works Co...........160, 256,	401
Colorado Portland Cement Co..............	401
Conveying Machinery Co..................	401
Crawford & McCrimmon Co.................	160
Crocker-Wheeler Co......................	772
Cutler-Hammer Mfg. Co...................43,	512
Cyclone Drill Co........................	281
DAVIS & SON (DERBY), LTD., JOHN.... ....160, 357, 401, 628,	772
De Laval Steam Turbine Co....... ....44, 160, 401, 512, 628,	772
Dempcy-Degener Co.......................	671
Denver & Rio Grande Railroad.............	43
Detroit Leather Specialty Co.............	160
Dixon Crucible Co., Joseph..............43,	357
Dodge Mfg. Co.........................160,	401
DuPont de Nemours Powder Co......401, 544,	772
EAGLE FOUNDRY AND MACHINE CO... 357,	512
Economic Machinery Co...................	772
Electric Railway Equipment Co............	401
Electric Service Supplies Co.............544,	628
Excelsior Drill & Mfg. Co................	772
FALKENBURG & LAUCKS..................	772
Flory Mfg. Co., S.......................	512
Foote Mineral Co........................	43
Foust Concentrator Co...................	671
GARDNER CRUSHER CO...................	772
General Engineering Co., 43, 160, 256, 281, 357, 401, 512, 544, 628, 671,.......	772
Goldschmidt Thermit Co.................256,	281
Goodman Mfg. Co.........................	160
HAZARD MFG. CO.....................401,	544
Hendrie & Bolthoff Mfg. & Supply Co...	43
Hendryx Cyanide Machinery Co..........544,	628

Page
Hill-Clutch Co. 160, 512
Holland Trolley Supply Co. 43
Holman Bros. 545
Hunt Co., C. W. 160
Hunt Filter Co. 44

INTERNATIONAL INSTRUMENT CO. 772
Industrial Instrument Co. 256, 628
Ingersoll-Rand Co. . . 43, 160, 256, 401, 512, 628, 671
Ironsides Co. 401

JEANESVILLE IRON WORKS CO. 628
Jeffrey Mfg. Co. 43, 512, 628
Johns-Manville Co., H. W. 43

KEUFFEL & ESSER CO. 401, 671, 772
Kolesch & Co. 256

LESCHEN & SONS ROPE CO., A. 357
Leyner Engineering Works Co., J. Geo. 160, 512
Lindsay Bros. 401
Linforth & Milburn 401
Link-Belt Co. 160, 357, 512, 544
Lippincott Co., J. B. 512

MACKINTOSH HEMPHILL & CO. 160
Maloney Oil & Mfg. Co. 401
McKiernan-Terry Drill Co. 401
Merck & Co. 772
Merritt & Co. 43
Mesta Machine Co. 671
Milwaukee Locomotive Mfg. Co. 544, 628
Mine and Smelter Supply Co. 281
Montague Compressed Air Co. 544
Montgomery Machinery Co., J. H. 628
Morris Machine Works 401
Mulconroy Co. 160

NATIONAL BRAKE AND ELECTRIC CO. . . 256
National Electric Lamp Association 160, 401, 512, 772
National Pulverizer Co. 772
New Way Motor Co., The 160

OHIO BRASS CO. 43, 256
Ore Concentration Co. (1905) Ltd., The . . 43

PACIFIC TANK CO. 628
Paint Manufacturers Association 43
Pennsylvania Flexible Metallic Tubing Co. 772
Pilley Mfg. Co. 772
Pneumelectric Machine Co. 160, 512
Portable Electric Safety Light Co. 160
Powell Co., Wm. 772
Precision Instrument Co. 401
Pulsometer Steam Pump Co. 772

QUEEN & CO., INC. 43

RAYMOND BROTHERS IMPACT PULVERIZER CO. 281
Ridgway Dynamo and Engine Co. 160
Rockwell Co., W. S. 256
Rockwell Furnace Co. 256, 544
Roebling's Sons Co., John A. 43, 160, 281, 401
Roessler & Hasslacher Chemical Co. 401
Robinson, J. R. 671

SAEGER ENGINE WORKS 772
Sauveur & Boylston 772
Scranton Steam Pump Co. 772
Servus Rescue Equipment Co. 628
Simmons Co., John 544
Smith, Emery & Co., Inc. 772
Standard Tool Co. 357
Starrett Pump & Mfg. Co. 772
Stephens-Adamson Mfg. Co. 512
Stocker, Geo. J. 671
Stromberg-Carlson Telephone Mfg. Co. 44, 281, 357, 401, 512, 545, 628
Sturtevant Co., B. F. 401
Sturtevant Mill Co. 160, 357
Sullivan Machinery Co. 160, 281, 357, 628, 671

TAYLOR INSTRUMENT COS. . . . 160, 281, 401
Taylor Iron & Steel Co. 160
Thew Automatic Shovel Co. 671
Toledo Pipe Threading Machine Co. 772
Trenton Iron Co. 544
Tyler Co., W. S. 160

ULMER CO., J. C. 772

VULCAN IRON WORKS 256
Vulcanite Portland Cement Co. 401, 544

WARREN-EHRET CO. 401
Watt Mining Car Wheel Co. 160, 401
Weber Subterranean Pump Co. 772
Webster Mfg. Co. 628
Wellman-Seaver-Morgan Co. 256
Weston Dodson & Co. 401
Western Electric Co. 43, 160, 256, 357, 401
Western Metallurgical Co. 772
Whitcomb Co., Geo. D. 628
Williams & Co., J. H. 43
Wing Mfg. Co., L. J. 772
Wisconsin Engine Co. 44
Wood Ore Testing Co., Inc., Henry E. 772
Wright Mfg. Co. 256

YORK BRIDGE CO. 401

Index E—Trade Notices

ALLEN & CO., LTD., EDGAR 407
American Blower Co. 747
American Locomotive Co. 632
American Safety Lamp and Mine Supply Co. . . 576
Atlantic Equipment Co. 632

BALL ENGINE CO. 208
Bartlett & Snow Co., C. O. 128
Borden Co., The 208
Bristol Co. 278

CARNEGIE STEEL CO. 208
Chamberlain, Wm. M. 748
Crawford & McCrimmon Co. 576

DE LAVAL STEAM TURBINE CO. 748
Denver & Rio Grande Railroad 748
Dixon Crucible Co., Joseph 128, 278, 407
Dodge, Day & Zimmerman 407
Dorr Cyanide Machinery Co. 407

ELECTRIC SERVICE SUPPLIES CO. . . 632, 748
Erdle Perforating Co. 632

FLORY MFG. CO., S. 676

GARDNER CRUSHER CO. 748
General Chemical Co. of California 208
General Electric Co. 189, 677
Goodman Mfg. Co. 632, 747
Goulds Mfg. Co. 747
Goyne Bros. 278

JEFFREY MFG. CO. 189, 208, 407, 508
Johns-Manville Co., H. W. 128, 208, 407, 576, 747

KOLESCH & CO. 189

LEROY MINING & MILLING CO. 632
Link-Belt Co. 407, 508, 576, 632
Lunkenheimer Co. 632

MCKIERNAN-TERRY DRILL CO. 278
Mineralized Rubber Co. 677
Mining and Coking Equipment Co. 677
Minneapolis Steel and Machinery Co. 208
Moore & Co., Charles C. 677

NELSON IRON WORKS 128

OGDEN ASSAY CO. 748
Ohio Brass Co. 208

PEYTON CHEMICAL CO. 208
Pilley Mfg. Co. 677
Pittsburg Testing Laboratory 208

SANFORD-DAY IRON WORKS 508
Scranton Acetylene Lamp Co. 576
Servus Rescue Equipment Co. 278
Simonds & Burns 632
Stine Co., J. C. 189
Stromberg-Carlson Tel. Mfg. Co. 208
Sturtevant Co., B. F. 407
Sullivan Machinery Co. 508, 632

TAYLOR INSTRUMENT COS. 128
Terry Core Drill Co. 278
Toomey, Inc., Frank 278

WAGNER ELECTRIC MFG. CO. 128
Weber Subterranean Pump Co. 677
Western Electric Co. 407, 676, 748
Westfalia Engineering Co. 508, 677
Westinghouse Co. 407, 677
Westinghouse Electric and Mfg. Co. . . 189, 278, 576
Williams Patent Crusher and Pulverizer Co. 632, 747

Mines and Minerals

Vol. XXXI—No. 1. SCRANTON, PA.—AUGUST, 1910—DENVER, COLO. Price, 25 Cents

MIAMI CONCENTRATING MILL

Written for Mines and Minerals, by R. L. Herrick

Features of Construction that Admit of Low Cost and Tend to Economy of Operation

The 2,000-ton concentrating mill of the Miami Copper Co., at Miami, in Gila County, Ariz., is now in course of construction. It is the design of H. Kenyon Burch, who constructed the Moctezuma concentrating mill at Nacozari, Sonora, Mex., which during the last year recovered 85 per cent. of the copper mineral in the ore, thus achieving a record for this kind of mill work. Preliminary to preparing plans for the Miami mill, Mr. Burch, bent on absorbing the good points in mills treating copper sulphide ores, visited Cananea, Mex., Anaconda, Mont., Garfield, Utah, and Ely, Nev. The Miami mill, therefore, embodies the best constructive features of previously built copper concentrating mills, besides original features and improvements by the designer. Fig. 1 shows the mill to be situated on a side hill, to the south of No. 4 main hoisting shaft, whose steel head-frame towers above the mill frame in the middle foreground. By constructing the mill on this steep hill, head room is obtained in the mill and a tailing dump in the gulches below, two advantages not always available. Adjacent to the No. 4 head-frame are the run-of-mine ore receiving bins of 1,000 tons capacity. Below the bins the coarse crushing machinery is to be placed, and between the coarse crushers and the crushed-ore bins is the mechanical sampler. One end of the building, in line with the fine crushing department, is to be devoted to a machine shop where repairs can be made. To move the heavy machinery in the fine grinding department to the repair shop, one 5-ton hand-moved crane and one 15-ton electrically operated power crane are installed. Heavy girders extend the length of the building for these cranes to operate on, so that in case of a breakdown, or if repairs are needed, the heavy parts of the machinery can be removed and new parts substituted with little delay. This is one of the original features of the plant, as the heavy machinery is either above or on the machine-shop floor.

Fig. 1. Miami Concentrator

Fig. 2. Rear of Miami Concentrator

The mill is divided into three units of two sections each. Fig. 2 shows the rear of the mill to have six circular concrete foundations on which will rest six steel ore bins, two for each section, and from which coarsely crushed ore will be fed to rolls that crush to $\frac{1}{8}$ inch in diameter. There are four floor levels to the main mill building, the uppermost being for screens and the fine rolls mentioned; the next lower floor is to be occupied by the Chilean mills; the third floor will carry the classifiers and Deister concentrating tables, and the fourth floor will be used for the conical settling tanks and Deister slime tables.

Above the third and fourth floors will be suspended narrow platforms, paralleling the tables, which will support the hydraulic classifiers and the launders that are to feed the tables. This arrangement is intended to promote dryness and cleanliness by preventing the splashings from falling to the table floor. The three products from the concentrating tables flow through separate pipes to their respective launders beneath the floor. There is a vault for each mill section centrally situated between the product ends of two lines of tables and in these the launders are placed. Fig. 4 shows the vault openings in the lowest retaining wall. The roof of the vault is the floor of the table room and is formed of iron in which are cemented thick pieces of glass. This floor is similar to a glass sidewalk and permits the daylight that enters from skylights above the tables to penetrate the vault beneath the floor. This is the original introduction of transparent floors into mill practice.

Another feature of importance in connection with the construction of this mill is the absence of elevators. The nearest approach to this kind of machinery is one inclined conveyer belt and several small centrifugal pumps. The flow of ore and pulp through the plant, with these exceptions, is accomplished by gravity.

The only hand work at this operation from the time the ore is broken in the mine stopes to the time the concentrate is going to its bin, consists in moving, loading, and starting levers and the adjustment of machinery. Shoveling and tramming are eliminated, making the mill the only one of its kind arranged in the world.

Construction Notes.—By taking advantage of the side-hill slope, which is hard cemented breccia of the Gila conglomerate, the mill terraces and tunnels are excavated from it. Little hard rock work was necessary and the concrete work consisted in facing up the excavations with retaining walls, etc. In this

way the elaborate concrete arched columns for foundations that have been a feature of several previously constructed concentrating mills were made unnecessary, with a consequent great saving. It is interesting to note that several score Apache "bucks" from the near-by Indian reservation condescended to work with pick and shovel on this foundation work and proved about as efficient as the Mexicans. The terrors of the Southwest a generation back were evidently tempted by the unusual chance to acquire the wampum for fire-water.

Fig. 5 shows the method of constructing the concrete retaining walls. Instead of building frames and tearing them down as fast as the wall was completed, the main walls were largely constructed by building sectional forms which were transported from place to place by means of a gin pole tackle. The illustration shows the forms in the foreground, the one immediately in front laid flat with its toe pointed vertically and about to be hauled into the correct position occupied by the adjacent form behind it. By using these forms built about 16 feet long, considerable time and expense were saved over the usual method, although the form method requires a greater employment of skilled labor and supervision to insure the avoidance of mistakes.

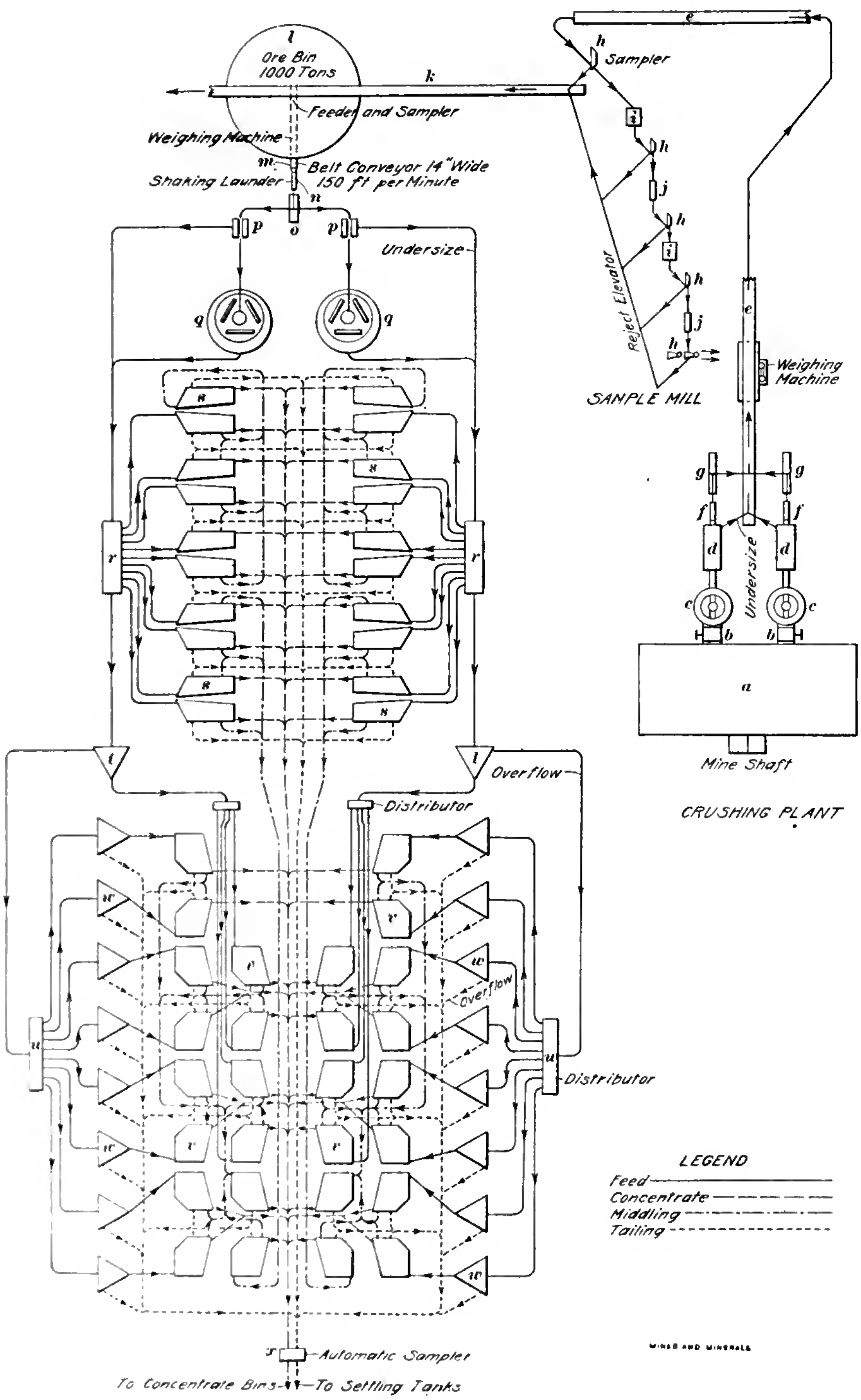


FIG. 3. FLOW SHEET, MIAMI CONCENTRATOR

The frame of the concentrator is of structural steel erected under contract by the Minneapolis Steel and Machinery Co. The sheathing will be corrugated iron with rubberoid or some similar roofing. With the exception of the wooden feed-launders and supporting platforms, nothing but concrete and steel enters into the construction of the entire plant.

The proposed mill treatment, as based upon the results achieved in the experimental mill, is approximately as indicated by the flow sheet, Fig. 3. The run-of-mine ore from bin *a* passes through the automatic ore feeder *b* to No $1\frac{1}{2}$ gyrating rock breakers *c*, whose average capacity is about 50 tons per hour through a 2-inch diameter ring. The material passes from the crushers to 4'×10' rotary screens *d*, which permit all that will pass through $\frac{3}{4}$-inch holes to go to a belt conveyer *e*. The oversize from the screens goes to the shaking launders *f* which feed the 54"×24" coarse rolls *g* that break the ore to $\frac{1}{2}$-inch cubes. From the coarse rolls the ore goes by gravity to the 30-inch wide conveyer belt *e* that travels at the rate of 300 feet per minute up an incline of 15° 30'. This delivers to the mechanical sampling department. On reaching the sampling mill the ore is cut out by samplers *h*, mixed by machines *i*, and crushed by rolls *j* until the final duplicate samples, weighing 40 pounds each, are obtained. The discard from the various samplers goes to the 30-inch wide conveyer belt *k*, and this, traveling at the rate of 250 feet per minute, delivers the ore by means of a tripper to one of the six large circular ore bins *l*. Reference to the flow sheet will show how the mill section is divided into two parts from here on. Fine rolls *o* of Mr. Burch's design, reduce the ore to $\frac{1}{8}$-inch cubes in the first application of wet crushing. The product is screened at *p* to 22 mesh or coarser, the screens' oversize going to Chilean mills *q*, the undersize to hydraulic classifiers *r*. The mills grind to 30 mesh and combine product with the screen undersize to the classifiers. The latter will probably be of the Richard vortex type, or the Swain individual classifier for each table. The coarse product of each classifier is split between nine Deister No. 2 concentrating tables *s* whose tailing goes direct to waste, and whose middling is treated on a tenth table. The fine slime product of the classifier, however, is settled in a conical tank *t* and the thickened pulp product distributed between four Deister slime tables *v*. The overflow from this first conical settling tank, however, flows to a distributor *u* which supplies eight conical settling tanks *w*, each individual tank serving to thicken the slime for one Deister slime table. All of these slime tables make a middling and tailing product in addition to concentrate. The

middling product from the 10 concentrating tables *s* on the floor above is combined and retreated on one Deister slime table which in turn gives its middling to a second slime table. The latter, in addition, treats the combined middling of the 12 previously mentioned slime tables and makes only concentrate and tailing.

The overflow water of the eight conical settling tanks *w* is combined with the tailing from the concentrating tables *s* and slime tables *v* and flows outside the mill to a system of concrete settling tanks. This will be similar to the system built by Mr. Burch at Nacozari, the clear overflow water from the tanks being pumped back to the head of the mill, and the sediment flushed to waste.

It will be noted that this proposed system provides for the accurate classification and uniform sizing of the pulp grains fed to every concentrating and slime table, thus insuring the early elimination of barren tailing and reducing the retreatment of middling to a minimum. By providing adequate capacity of settling tanks, every concentrator is likewise fed a pulp sufficiently thickened to insure the best results. And finally, by installing a tailing-settling system of adequate capacity, tanks of non-leaking concrete, etc., the continued reuse of the mill water is made possible, thus reducing its waste to a minimum, a noteworthy feature in such a desert country where the water expense is an important item.

Concentration Results.—While the ratio of concentration is not yet fully determined, it is likely that about 20 to 1 will be adopted. From the experimental mill tests it seems likely that the extraction will be somewhere between 75 and 80 per cent., or from an ore averaging 2.75 per cent. copper, the recovery will be from about 40 to 44 pounds of metal per ton milled.

This will give a product expected to contain about 40 per cent. copper, 35 per cent. silica, 6 per cent. iron, and the balance in sulphur and alumina.

With the depth of ore proved to the 620-foot level, thus giving it a thickness of 400 feet, and making 14,000,000 tons in sight, a production of 700,000 tons of ore annually can be maintained for at least 20 years. This does not take into consideration the possibilities of either prolonged life or enlarged production, based upon the other 150 acres of unexplored ground now thought to be mineralized.

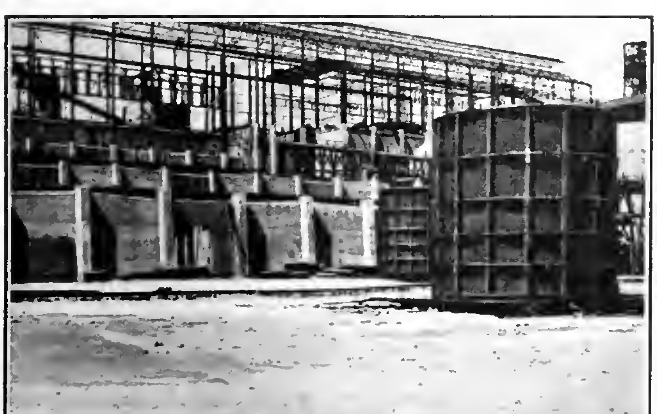


Fig. 4. Lower Portion of Miami Concentrator

Water Supply.—In addition to two valuable water rights of its own which will supply considerable water, the Miami company has secured from the Old Dominion company a supply of 1,000,000 gallons per day. This latter water will flow by gravity to the company ranch on the lower end of Miami wash at which point the pumping plant will be located. From here the water will be pumped through a 14-inch diameter cast-iron pipe to the storage tank located on the hill above the concentrating mill. Since the choice of mill site involved the problem of whether it was cheaper to pump water than to transport ore, it was solved in favor of the pumping.

Power Plant.—The power plant is situated at the foot of the hill, just east of the valley settlement of Miami. The containing building is constructed wholly of steel and concrete, and built for more power machinery than the present mill requires in order to provide for future installations. The boiler plant consists of three water-tube boilers burning fuel oil, each boiler of 600-horsepower nominal capacity with steam at 185 pounds pressure and 100 degrees of superheat. The steam engines, three in number, are of four-cylinder triple-expansion type, size 21″×40″×40″×40″×48″ stroke. Each engine is direct-connected to a 1,000-kilowatt, alternating-current generator producing three-phase current of 25 cycles at 6,600 volts pressure.

Fig. 5. Movable Forms for Concrete Work

The electric current will run the underground haulage system, the mill shops, and the lighting system, but compressed air will operate the mine drills and the hoisting engines.

The air-compressor plant consists of two machines, of the four-cylinder, triple-expansion, two-stage type, each with a capacity of 4,000 cubic feet of free air per minute compressed to 90 pounds per square inch pressure. This air will be piped a half mile from the power plant to No. 4 shaft where, by use of fuel oil, the air will be preheated before use in the hoisting engines.

SIMULTANEOUS INVENTIONS

That "nimble thoughts can jump both sea and land" is exemplified by the invention in Australia and the United States, granted almost simultaneously, of processes for the separation and recovery of zinc sulphides from intermixed sulphides or gangue. Scarcely had the full text of the process invented by Mr. H. J. Horwood, mine manager of the Broken Hill Proprietary, and which was published in our issues of November 10 and 17 last, been circulated throughout the world than the news of a similar patent taken out in the United States came to hand. The specifications of this patent are not available, but *The Mining Magazine* states that a patent covering exactly the same ground has been granted in the United States to H. A. Wentworth, who has assigned it to the Huff Electrostatic Separator Co., of Boston. This patent is numbered 938,732, and is dated November 2. The specification describes how complex sulphides are ground sufficiently fine, and subjected to a roast for a few minutes at a dull red heat. In this way the particles of pyrite and galena are covered with oxide or sulphate, while the blende remains unaffected. Subsequently the mixture is treated by a flotation process, whereby the blende is floated off while the iron and lead compounds sink. With the exception that the respective inventors differ as to the length of time required for the roast, the two processes are identical.—*Australian Mining Standard.*

GEOLOGY OF THE COALINGA OIL FIELD

Abstracted for Mines and Minerals

Location and Topography. A Description of the Distinguishing Rocks and Fossils

The Coalinga, Cal., oil field is in the southern part of Fresno County, the western part of Kings County, and is bounded on the south by the Kern County line. To fully understand the conditions in Coalinga it is necessary to have a reference map showing the ranges, townships, and the developed oil field. In a general way, however, from the southern extremity of the developed oil field, the foot-hills run due north for about 7 miles, then turn east for about 3 miles, and then again bend north. From the curve thus formed a spur of hills extends in a southerly direction for several miles.

This topographical arrangement in the Coalinga oil field is that of a plain some 6 miles square, surrounded on three sides by the hills, and bounded on the south by the railroad. The town of Coalinga is on the railroad and close to the foot-hills, and therefore near the southwest corner of this plain, which is smooth, and slopes very gently northwest to southeast.

Oil has been found on the lowest portion of the foot-hills, and extending out for a varying distance from them, on the north and west sides of this plain, as well as on the outer or valley face of the foot-hills where they turn to the north, and on the same side of the spur range. The "East Field" is that portion lying on the valley slope of the north limb; the "Southeast Extension" on the east or valley slope of the spur. The "West Field" lies on the inner portion of the hills west and south of the plain. The "South Field" is the prospective territory south of the railroad, following the hills for an indefinite distance.

The length of the developed field, extending from the railroad north, then east, then north again, is some 13 miles. The

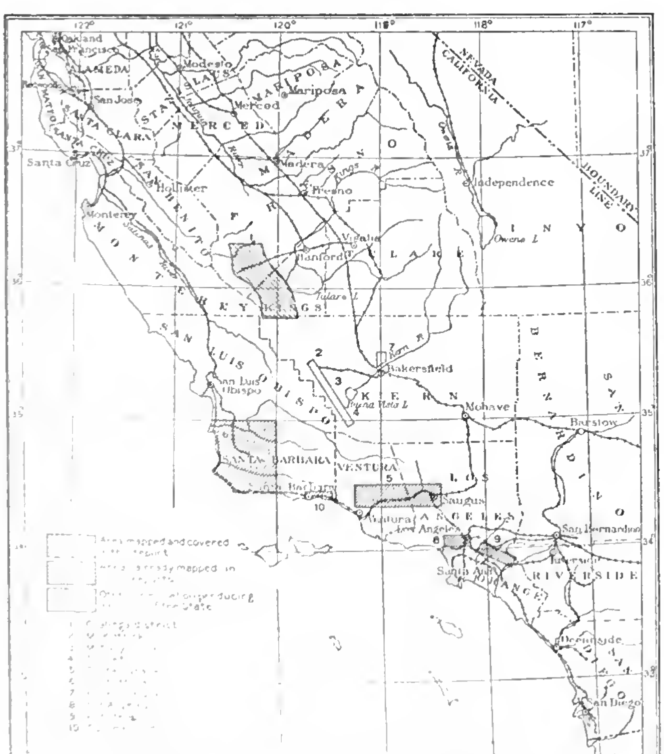


Fig. 1. Map Showing Location of Coalinga, Cal., Oil Field

extension to the ... has been proven up for a length of approximately a mile and a half. The width of the actually proven belt varies from ¾ of a mile at the lower extremity to about 2½ miles at the widest part, with an average of perhaps 2 miles.

Drilling is carried on in the foot-hills because the oil-bearing sand is there nearer the surface than it is in the plain, where it is over 4,000 feet deep.

The eastern slope of the mountains bordering the San Joaquin Valley is formed by a great thickness of strata dipping toward the valley.

The oldest rocks appear in the axis of the mountain and grade into younger rocks in the following order: Franciscan, Knoxville, Chico, Tejon, Vaqueros, Santa Margarita, Jacalitos, Etchegoin, Tulare, and late Quaternary alluvium and terrace deposits. With the exception of the magmatic rocks associated with the Franciscan, all formations are of sedimentary origin. The following table correlates the rocks of the coast range and the Coalinga oil district:

Era	Period	Epoch	Standard Coast Range Series Section	Coalinga District Series Section
Cenozoic	Quaternary	Recent	Alluvium	Alluvium
		Pleistocene	San Pedro —Unc.— —Unc.—	Stream deposits, valley fillings, and raised beaches —Unconformity—
	Tertiary	Pliocene	Merced	Tulare
			San Diego	—Unconformity—
		Miocene	San Pablo	Etchegoin —Unconformity—
			—Unconformity—	Jacalitos —Unconformity—
			Santa Margarita —Unconformity—	Santa Margarita (?) —Unconformity—
			Monterey	Lacking (with possible exception of a small part)
			—Unconformity— Vaqueros —Unconformity—	Vaqueros —Unconformity—
		Oligocene	San Lorenzo —Unconformity?—	Lacking
		Eocene	Tejon	Tejon —Unconformity—
			Martinez —Unconformity?—	
Mesozoic	Cretaceous		Chico —Unconformity— Horsetown —Unconformity—	Chico
			Knoxville —Unconformity—	Knoxville —Unconformity—
	Jurassic?		Franciscan —Unconformity— Granitic rocks, etc. —Unconformity—	Franciscan
	?		Schist and limestone	

The original rocks of the Franciscan series, which occupy the central portion of the Diablo Range are sandstone, shale, jasper, schists, and serpentine without fossils, being more or less metamorphosed. The Knoxville-Chico rocks comprise a thick succession of sandstone, shale, and conglomerate that may be recognized by the dark, thin-bedded, compact shale of the lower portion and the massive drab concretionary sandstone of the upper portion. From the Franciscan below to the Tejon above, the Knoxville-Chico formation is 12,800 feet thick. Owing to the insufficiency of evidence and the absence in the Middle Cretaceous epoch of the Horsetown series of the standard Coast Range series in the Coalinga district, the Knoxville and Chico series are not separated. Commencing with the lowest, and therefore earliest formation of the Knoxville-Chico rocks, the strata are about as follows:

1. Thinly bedded dark shale similar to that above, with some sandstone layers.
2. Massive iron-gray sandstone.
3. Thinly bedded dark shale and sandstone similar to that above, but without fossils.
4. Coarse massive conglomerate zone of locally variable thickness with large boulders of pre-Franciscan rocks. Probably basal conglomerate of the standard Chico.
5. Alternating thin, sharply defined beds of dark clay shale, sandy shale, iron-gray and brownish-gray sandstone and some beds of conglomerate and pebbly sandstone; marine

fossils of the Chico formation which is Upper Cretaceous, are found sparingly in the upper portion.

6. In the upper half of the upper division of the Knoxville-Chico series there are found purplish siliceous shale, dark clay shale, light-colored calcareous shale, white and yellow sandstone, and a minor zone of tawny concretionary sandstone. In the lower half mostly massive drab concretionary sandstone is found. The marine fossils shown in Fig. 2 are found sparingly throughout the upper half of the Knoxville-Chico.

The Tejon series of rocks above the Chico is named from Fort Tejon, in Kern County, Cal. It is of the Eocene epoch and attains a thickness of 1,850 feet in the Coalinga oil district. From the Chico up, the beds are:

1. Marine yellowish brown and gray fossiliferous sandstone and dark clay, with a local basal conglomerate.

2. Marine white and brown shale carrying diatoms and other minute shells of rhizopods called foraminifers. The distinctive feature of the formation is the increasing fine-grained texture from the sandy lower beds to the thin and tinted shales above. Fossils of the Tejon series are shown in Fig. 3. There are also a number of bivalves on the order of those found in the Vaqueros series of rocks.

The Vaqueros sandstone of the Lower Miocene epoch is said to attain a thickness of 900 feet in the Coalinga oil field. The upper member is a marine gray sandstone and sand with subordinate conglomerate, gravel, diatomaceous earth, and clay shale. The series is distinguished by the protruding tendency of the hard sandstone in its central portion known locally as the "reef beds." The lower number of the Vaqueros consist of marine white and brown diatomaceous earth and shale. The beds at the base of the Vaqueros series are said to be the chief oil-bearing sands of the Coalinga district. In many places they are saturated and discolored with petroleum. Wherever they overlap the Tejon stage and cover the Chico stage the lower beds lose their petroleum at a distance from the Tejon. This shows the oil horizon in the Coalinga district to belong to the Tejon and Vaqueros sands particularly, although upper sands may also contain oil. The fossils found in the Vaqueros formation, shown in Fig. 4, are not all bivalves, as an occasional gasteropod is found, as shown at *3*.

The Santa Margarita stage is said to be Middle Miocene and between 900 and 1,000 feet thick. On the eastern side of the Coalinga oil field there are a number of strata containing large fossil bivalves and a species of coral. Owing to this formation it is assumed that these beds are the Santa Margarita series, found in San Luis Obispo County, in the standard Coast Range. They are traceable as far south as the San Joaquin coal mine, and beyond that the beds are either lacking or are unfossiliferous. North of Waltham Creek the supposed Upper Middle Miocene is composed of marine fossiliferous sand, clay, gravel, and comminuted serpentine. South of Waltham Creek the Santa Margarita series are white, purple, and brown shales. The fossils belonging to this stage of the Miocene are shown in Fig. 5.

The Jacalitos stage of the early Upper Miocene epoch is said to be 3,800 feet thick in the Coalinga oil field. Jacalitos is named from a creek where the beds of this stage are exposed. The formation is composed of slightly consolidated marine fossiliferous beds of light-gray, greenish-gray, blue and brown sand, clay and fine gravel, interbedded with similar deposits indurated into sandstone, shale, and conglomerate, with some siliceous shale. The formation may be roughly distinguished as that portion of blue sand and sand rock above the brownish shale of the Santa Margarita and the large beds of blue sand of the Etchegoin formation above it.

A feature of the Jacalitos is the occurrence in it at intervals of hard rocks that project like saw teeth and which, by their resistence to weathering, protect the beds immediately above and below them, thus forming long parallel ridges. Another feature of the Jacalitos formation is the great number of sand and pebble beds full of fossil sea urchins, Fig. 6 *5*.

The Etchegoin formation is classed as the uppermost Miocene, and assigned a thickness of 3,800 feet in the Coalinga district. It derives its name from an abandoned sheep ranch about 20 miles northeast of Coalinga, and consists of slightly

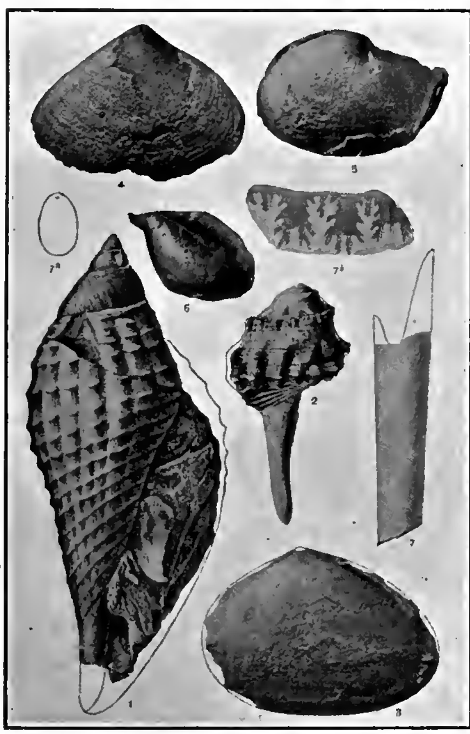


Fig. 2. Chico Fossils
1 and *2* are gasteropods; *3*, *4*, *5*, and *6* mollusks; *7*, *7a*, and *7b* cephalopods

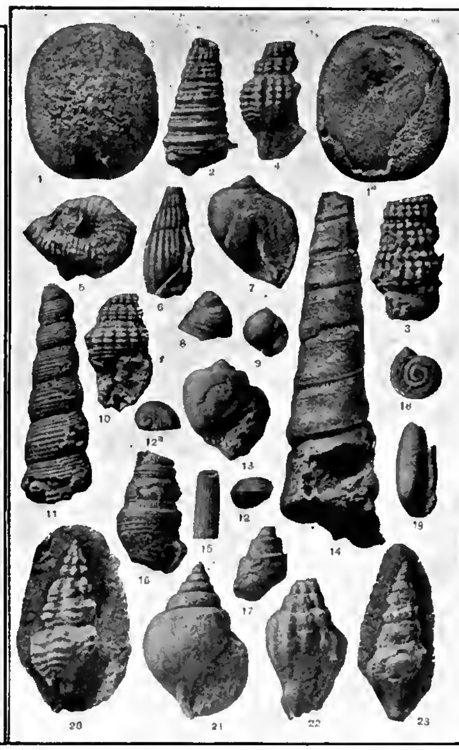


Fig. 3. Tejon Gasteropoda and Echinodermata
1 and *1a* are sea urchins or echinodermata; the remainder are gasteropods

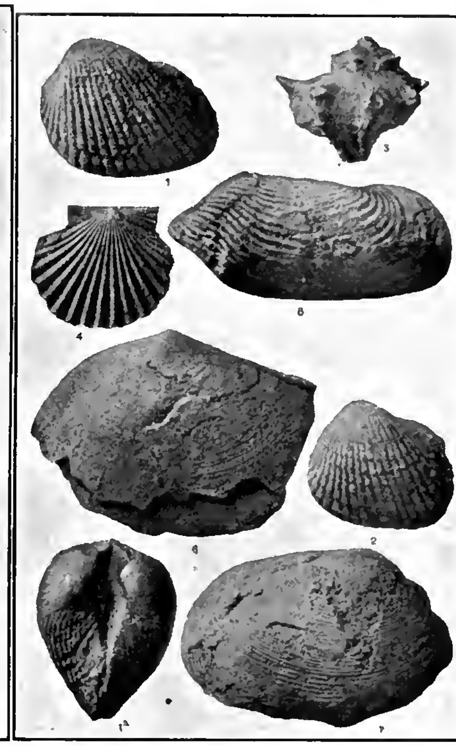


Fig. 4. Vaqueros Fossils
1, *1a*, *2* and *4*, scallops; *3*, periwinkle; *5*, *6* and *7*, clams

consolidated, chiefly marine, fossil beds of gray and blue sand, black clay, light sandy clay, pebbly sand and gravel, with locally hardened beds of sandstone and occasional layers of siliceous and calcareous shale. The upper third is largely dark clay, the lower portion blue sand. This stage is distinguished by an abundance of sea urchins, Fig. 6 *5*, and barnacles, Fig. 5 *1*, as well as those shown in Fig. 7, which are bivalves and gasteropods.

The Etchegoin formation in the Coalinga district is overlain along the border of the San Joaquin Valley by thick beds of gravel, sand, clay, sandstone, conglomerate, and some limestone, which form the Tulare formation of the Pleistocene The formation is said to be 3,000 feet thick and is in the Pliocene of the Lower Pleistocene epoch. The lithological condition of this formation is given as unconsolidated but locally hardened unfossiliferous, light gray and yellowish sand, light and dark clay, coarse and fine gravel, and thin layers of purplish sandstone; in part of fresh water and marine origin. At the base are fresh-

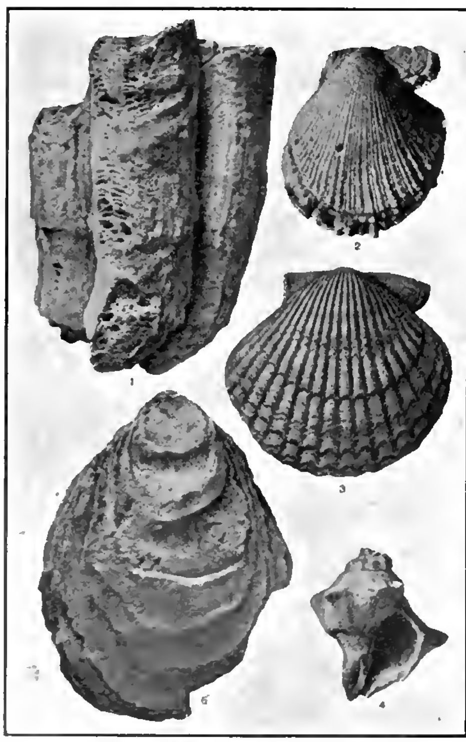


Fig. 5. Santa Margarita Fossils
1 is a kind of coral related to the barnacle; *2*, *3*, and *5* are scallops and oysters; *4* a periwinkle

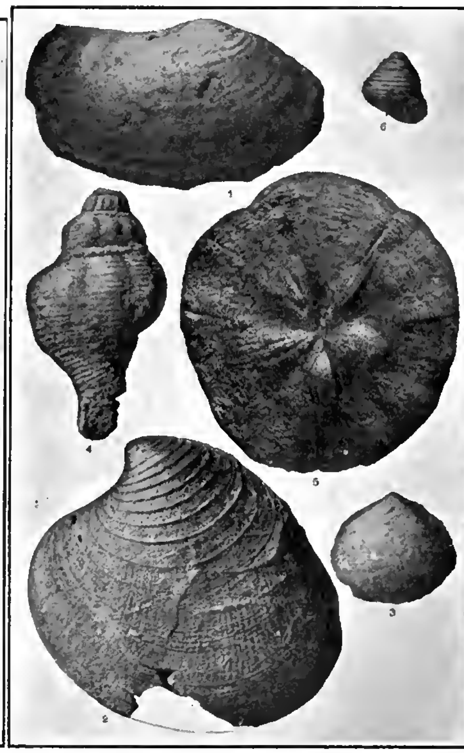


Fig. 6. Jacalitos Fossils
1, *2* and *3* bivalves; *4*, Periwinkle; *5*, Enchinoderm; *6*, Snail

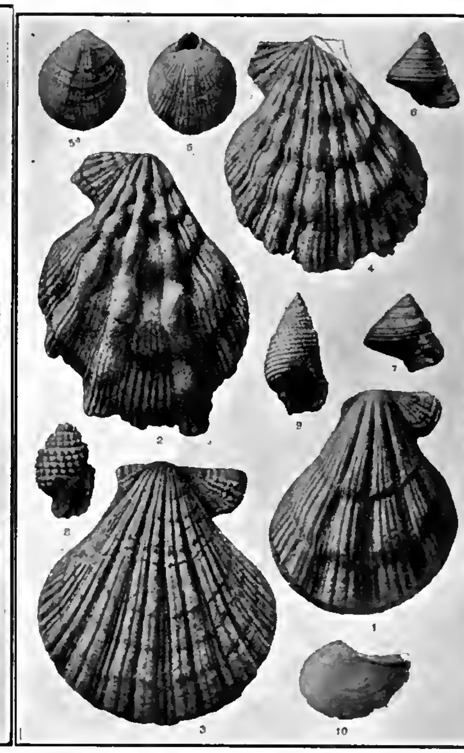


Fig. 7. Etchegoin Fossils
1, *2*, *3*, *4*, Scallops; *5*, and *5a*, *6*, *7*, *8*, *9*, Gasteropods; *10*, Mussel

water sand, sandstone, gravel, shell deposits, and limestone. In the Tulare formation are found shells shown in Fig. 8, which will be recognized as species of bivalves, periwinkles, and snails, approximately such as are to be found at this time. In places it is impossible to separate the Jacalitos, Etchegoin, and Tulare beds, and in the McKittrick district, southeast from Coalinga and west from Bakersfield, the three are combined and termed the McKittrick formation. The Tulare formation may be recognized most readily by the fresh water fossils and strange [illegible] its base, and by the boulder gravel, which is more [illegible] abundant than in any of the Tertiary formations.

[illegible] tion given in this abstract is obtained from Mr. P. W. [illegible]'s Report to the State Mineralogist, L. E. Aubury [illegible], and from Bulletin No. 398 of the United States G[illegible], by Ralph Arnold and Robert Anderson. M[illegible] field are being prepared by the State Mining Bu[illegible]nia, and very good geological and district maps [illegible] United States Geological Bulletin, No. 398. Fig. 9 [illegible] of the California oil well, one of the large[illegible]

RADIUM IN AUSTRIA

Thirteen grams of radium chloride have been produced at the Imperial Austrian radium factory located at St. Joachimsthal. It is estimated that this mineral has a value of $45,000 to $50,000 a gram, or more than half a million dollars for the entire amount. Pure metallic radium is never seen, but is always in combination. The bromide or chloride is extracted from pitchblende by repeated reductions. This latter mineral is taken from a worked-out silver mine near St. Joachimsthal, a small village in the Erzgebirge, about 12 miles northwest of Carlsbad.

The property belongs to the Austrian Government and the pitchblende has been mined during the last 50 years for the uranium, which is its principal mass, or about 50 to 90 per cent. of the whole.

Uranium is used in the manufacture of decorated glass and porcelain. Its yellow oxides are employed by glass manufacturers to give a delicate greenish-yellow tint to glass, and the black oxides are used as a pigment in porcelain painting. The colors are exported extensively and sell on the average for $12 a pound.

At present about 100 men are engaged in mining the pitchblende, which is found in small seams. They work in the mine 8 hours a day and the rest of the time cultivate small farms in the neighborhood.

The annual output of the mine is reported as about 400 hundredweight of pitchblende, equal to nearly 100 hundredweight of uranium ore. It requires approximately 10 tons of pitchblende to yield 1 gram of radium bromide, which is contained in the residue after the uranium is extracted. In the process 11,000 pounds of chemicals and 110,000 pounds of water are employed.

In order to obtain the radium bromide, more than a thousand crystalizations and reductions have been made at the factory. These require from a few hours to several days each, growing more difficult the nearer the approach to the radium chloride.—*United States Consular Report.*

BRAZIL'S IRON-ORE DEPOSITS

Consul-General George E. Anderson, of Rio de Janeiro, calls attention to certain advance figures on the known iron-ore deposits in Brazil.

Large Amount of High Grade Ores As Yet Undeveloped, Awaiting Means of Transportation

The report is furnished by Dr. Orville T. Herby, chief of the geological survey, of Brazil, as tentative conclusions on Brazil's available iron-ore supply. It is printed in Portugese for use of the International Geological Congress, which meets in Stockholm this year. Excerpts from the paper are taken.

Colonial records show that in 1590 iron was found south of Sao Paulo. Gold and silver being also reported in the same region, the Portugese Government sent in 1597, a colony from the mother country to promote the mining industry. One or two forges commenced to produce iron in about 1600, and this was the first to be manufactured in the Western Hemisphere; about 100 years later, the gold fields of Ouro Preto, the former capital of Minas Geraes, were discovered. This opened to exploration and permanent settlement the most extensive and important of the Brazilian iron fields, that of the Sierra do Espinhaco, or Backbone Range, forming the eastern rim of the basin of the Sao Francisco River. A large part of the gold mines of this and the neighboring district are actually in iron ore, though a century or more elapsed before any recorded efforts were made to turn it to use.

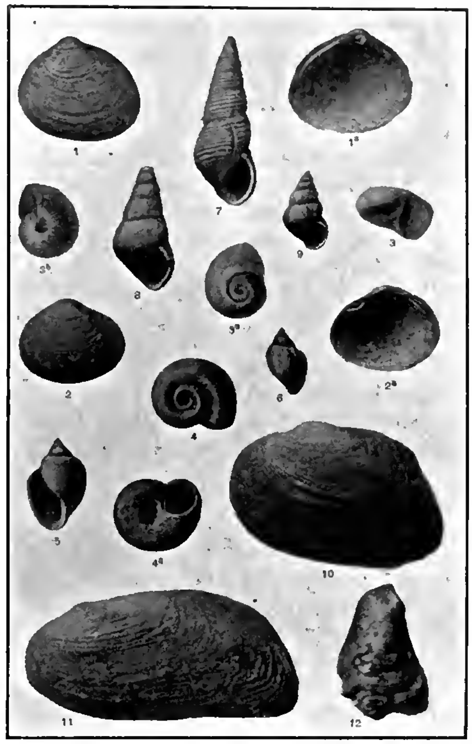


Fig. 8. Tulare Fossils

There is, however, a presumption that long before the definite establishment of the iron industry some metal may have been produced by the primitive African methods with which many of the slaves imported for the gold mines must have been familiar. Eschwege states that in 1811 most of the smithies produced their own iron, either directly in an ordinary blacksmith's forge, or in a primitive furnace constructed for the purpose.

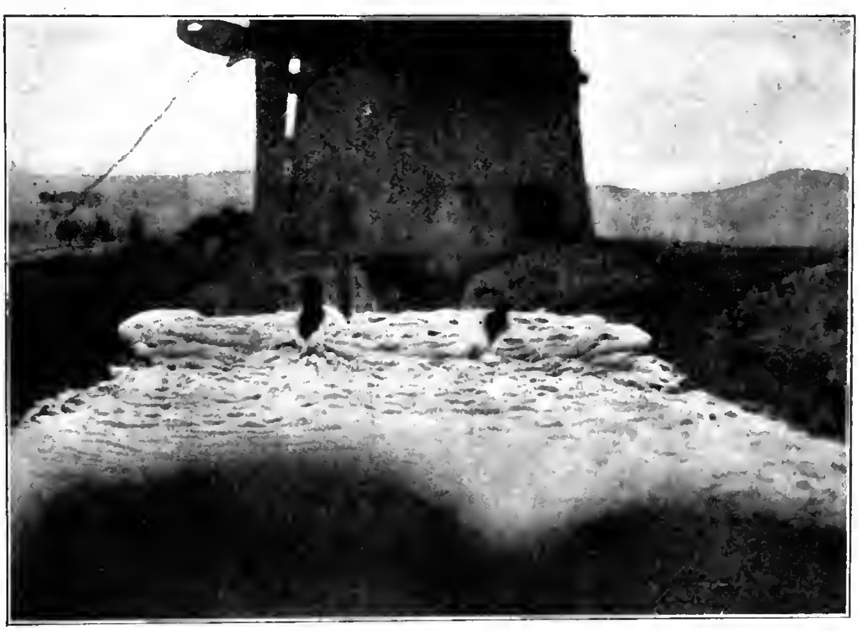


Fig. 9. Flowing Oil Well

Under the direction of Eschwege a company was formed to erect a direct-process plant near Ouro Preto which produced metal at the end of 1812 at the rate of about a hundredweight per day. The improvements here introduced were eagerly copied and in a short time the whole mining district of Minas Geraes was dotted with little furnaces. The number of these was estimated in 1864 as 120, many of which are still in operation.

In 1765 the production of iron was resumed at Ipanema in Sao Paulo, but again abandoned after a few years. An attempt made in 1800 to revive the industry, by means of a high furnace, was unsuccessful, and in 1810 a Swedish metallurgist under contract with the government constructed four direct-process furnaces which continued in operation until 1818, when two high furnaces constructed by the German engineer officer, Frederic von Vernhagen, then in the service of the Portuguese government, were put into operation. These continued in blast under government administration and with a daily production of 3 or 4 tons until 1895.

Between 1809 and 1814 an attempt was made to establish a high furnace at the Morro de Pilar in the neighborhood of Serro in Minas, but the enterprise was abandoned before reaching the productive stage, and until 1888 all the iron produced in Brazil, outside of Ipanema, was made by the direct process. In that year a high furnace with a daily capacity of four tons, afterwards raised to six, was put into blast at Esperanca near Itabira do Campo in the state of Minas Geraes, and this has continued to operate successfully, being now the only establishment of its kind in Brazil.

This inglorious history, extending over two centuries is suggestive of conditions unfavorable to the development of an iron industry, but it is sufficient to say that a deficiency of excellent ore is not one of them.

Iron ores are known in every state of Brazil and specimens of them figure in every national exposition, but for the most part definite information regarding the deposits is lacking. The greater part of the specimens exhibited are magnetites, highly titaniferous. With a single exception knowledge of the iron ore deposits of Brazil is lacking, and this exception does not furnish very definite information. A good number of specimens of hematites, representing numerous widely separated localities, have also appeared.

The districts regarding which definite information is at hand, belong to the states of Minas Geraes, Sao Paulo, Bahia, Goyaz, Parana, Santa Catharina, Rio Grande do Sul, and Matto Grosso The ores known in the first three and the last of these states are predominantly hematites; in the others,

predominantly magnetites. With a single exception, knowledge of these ore deposits is limited to the mere fact of their existence, their approximate geographical position, and the outward aspect of the ore picked up by unskilled observers.

The exception is the district situated in the eastern central part of the state of Minas Geraes, in the section of the Espinhaco Range that forms the divide between the Rio Doco and the Sao Francisco drainage systems extending over about two degrees of latitude with a width of about one degree of longitude. As already remarked, this is also the most productive of the gold fields of Brazil, and in consequence is one of the oldest and most densely populated of the interior regions. On this account and also on that of being on the road to the diamond fields of northern Minas Geraes, it has been more frequently visited and described by travelers than any other interior region. Notwithstanding this fact, it has been very imperfectly mapped and there is a singular lack of reliable and accurate information regarding its geological and economic features. A small party of the Servico Geologico e Mineralogico do Brazil, under the direction of Dr. Luiz Felippe Gonzaga do Campos, for the two past years has been occupied in mapping the district both topographically and geologically. The following brief account of the district is taken almost exclusively from a preliminary report now in course of preparation by Dr. Gonzaga do Campos.

The only all-rail means of access is by the Central Railroad of Brazil, which, starting from the port of Rio de Janeiro, enters the district at its southern border at kilometer 493, traverses it in a north and south direction for a distance of 90 kilometers to Sabara and extends a branch transversely across it which, when completed to Santa Barbara, will have an extension of 60 kilometers. The Leopoldina Railway, a narrow (1 meter) gage road belonging to an English company, starting from the same port, with a present extension of 630 kilometers will eventually tap the district at its northeast corner at Itabira do Matto Dentro with an extension of about 87 kilometers. The Victoria and Diamantina Railroad, also of meter gage and belonging to a French company, is now under construction from the port of Victoria, in the state of Espirito Santo, toward the same point (Itabira), which it can reach with an extension of about 630 kilometers. It is understood that the company proposes to complete this line as soon as possible and to equip it especially as an ore-carrying road. If this is done and if the port of Victoria, which is an excellent one, is properly equipped with handling facilities, one of the most important parts of the district will have a satisfactory outlet in the near future.

In its essential geological features the region is constituted by a basement complex of crystalline schists (gneiss, mica, amphibole, chlorite, and talcose schists), overlaid by a heavy series of partially metamorphosed sedimentary beds, profoundly folded and faulted. In this series, which consists principally of quartzites and clay slates with subordinate beds of limestone, a prominent member is the peculiar iron-bearing quartzite to which the name of "itabirite" has been applied. This name was originally proposed by Eschwege, in 1822, for the massive, pure iron ore of which the peak of Itabira do Campo, among others, is composed, and which is associated with a schistose rock composed of granular quartz and scaly hematite, which he discriminated as iron-mica schist. By common usage the name has come to be applied to the latter rock, and it is only in this sense that its retention can be justified as a convenient term that would otherwise have to be designated by an awkward and misleading descriptive name. Through variation in the proportions of the constituent elements, this type of rock grades off on one side to a purely quartzose, and on the other side to a purely hematitic phase. The phases sufficiently rich in iron to be commercially valuable may be conveniently designated as itabirite ores. This shows all possible gradations from an almost pure quartz rock with scattered flakes of hematite to a massive hematite free from quartz. The latter is in intercalated layers, varying from a few millimeters up to scores of meters in thickness, alternating with leaner quartzose portions. When limestones are with the series, they are also frequently associated with commercial ores of both iron and manganese.

The itabirite beds, become, when exposed to the weather, extremely friable, and as the region is one of heavy rainfall, they have been extensively denuded. In consequence of this, the massive portions, stand out as topographical features; the rain- and wind-swept slopes become covered with a rubble of iron ore due to the breaking up of the thinner intercalated layers, more or less completely freed from the associated siliceous elements by rain and wind action, and the bottom lands of the valleys become charged with deposits of iron sand separated by the natural sluicing of the streams. There are thus produced from the same series of beds three classes of ore; namely, (1) quarry ore, in the peaks and other natural exposures in situ of the massive portions of the rock; (2) rubble ore, on the denuded surfaces; and (3) sandy ore, in the valleys where sluicing action has taken place. To these must be added a fourth class, the so-called "canga" (contraction of Tapanhoacanga; i. e., "negro's head"), due to the cementation by limonite of the rubble ore into a hard stone conglomerate. It is probable also that still a fifth class might be recognized in the outcrops of quartz-hematite rock sufficiently friable to permit the separation of the metallic mineral by sluicing.

The crystalline areas of older rocks contain no iron ore, but in places, particularly in the southern part in the neighborhood of Queloz, there are important deposits of manganese that have been extensively mined. In the sedimentary area, on the contrary, iron ore of one kind or another is much more abundant and widespread. The outcrops on belts of the iron-bearing formation are so generally covered by superficial deposits of rubble ore and canga that by picking your route it is possible to traverse the area from one side to the other and along various lines without leaving, except for short intervals, one or another of the various kinds of ore deposits enumerated.

The deposit on the southwest extends to the mountain of Itatiaussu, which is known to be composed in large part of the iron-bearing formation and is reputed to contain large workable deposits of ore; that on the north includes the main ridge of the Espinhaco Range in the direction of Coniceicao and Serro, along which several iron mountains are known to occur, while the one on the northeast embraces the Candonga district, which is also reputed to be rich in iron ore.

From the above description it is evident that attempts to estimate the amount of ore in the district must be extremely fallacious. Practical geologists who have visited parts of the district hesitate to pronounce estimates that at first sight seem utterly preposterous.

The following estimates made by Dr. Gonzaga do Campos, of the Servico Geologico, will serve to give an approximate idea of what the ore may amount to when the district becomes better known. These estimates for nine of the deposits are as follows:

	Cubic Meters
Gava	72,000,000
Conceicao	80,000,000
Esmeril	19,000,000
Caue (Itabira peak)	33,000,000
Pitanguy	14,000,000
Sao Luiz	8,000,000
Peak of Itabira do Campo	8,000,000
Rio de Peixe	10,000,000
Cocaes	3,000,000
Total	247,000,000

Taking the specific gravity of these ores as 4, this volume represents 988,000,000 tons. In these estimates no account is taken of the presumed underground extension of the visible ore bodies.

No attempt has been made to estimate the volume of the rubble-ore deposits, which are both numerous and extensive throughout the district. So far only one such deposit has been actually measured by competent mining engineers, and this is

said to carry "20,800,000 tons of rubble ore, carrying 50 per cent. iron." From what is known of the district it seems quite safe to assume that there are scores of deposits of equal importance and that in the aggregate the volume of rubble ore is at least equal to that of the quarry ore.

As regards the canga, Dr. Gonzaga do Campos estimates that it covers about 10 per cent. of the area occupied by the iron-bearing formation, which is about 5,700 square kilometers. For the purposes of calculations, however, he takes 5 per cent. with a mean thickness of 2 meters which gives 570,000,000 cubic meters, which, calculated with a mean specific gravity of 3, gives 1,710,000,000 tons of ore, whose mean iron content will probably oscillate in the neighborhood of 50 per cent.

Most of the analyses are vague in that the phosphorus is given as "traces," without accurate determination, and no reference is made to titanium, leaving it doubtful whether this element had been sought. The oxide of iron is generally given as from 97 to 99.5 per cent., the remainder being almost exclusively silica. In two cases in which special tests were made for titanium none was found, and from this it seems safe to assume that as a class these ores are practically free from it.

The most reliable phosphorus determinations at hand, are from samples submitted to the Krupp works and to the United States Steel Corporation, both of which give the same result; namely, .0024 per cent.

From an industrial point of view, it seems tolerably safe to set the rubble ores down, as regards quantity and quality, as comparable with those of the quarry class.

The canga class of ore is naturally of lower grade than the others, owing to admixture of fragments of quartzose and argillaceous rocks, which it would be impracticable to separate in mining, and to the presence of water in the characteristic limonitic cement. Good observers estimate the mean iron contents of this class of ore at about 50 per cent. As already remarked, the apparent quantity of this class of ore is immense and, so far as can be judged by a simple ocular inspection, probably in excess of that of any other class.

Aside from the district above discussed and of its various prolongations that have not yet been examined, hematite ores are known to exist in various other districts of the state of Minas Geraes and in various other states of the republic.

The iron-ore formation seen many years ago by the writer at various points along the River Sao Francisco is believed to be identical with that of Minas Geraes, and some of the deposits may prove to be comparable with those above described as regards quantity and quality, but on this head nothing definite can be said, as the examination was of the most cursory character. That of Urucum in Matto Grosso occurs in association with manganese ores and with limestone, and in this respect offers a certain analogy with that of Minas Geraes, but it differs in the fact that the siliceous admixture, where it occurs, is in the form of jasper rather than that of granular quartz. The deposits are reputed, on good authority, to be extensive, but the quality judging from chance specimens that have come to hand, is not as high as in the Minas Geraes ores.

As already remarked, the magnetite is a very widespread mineral in Brazil, and many of the occurrences are reputed to be extensive. The most accessible and, so far as known, the most extensive, of these occurrences are situated in the coast region of Southern Sao Paulo, Parana, and Santa Catharina.

FAST TUNNEL DRIVING

Written for Mines and Minerals

In the April, 1909, issue of Mines and Minerals, was an article by R. L. Herrick in which he compared and tabulated the speed with which American and European tunnels were driven. In the April, 1910, issue he had an article on the Laramie-Poudre tunnel whose object is to divert the abundant waters of the Medicine Bow range of mountains to ditches to irrigate 200,000 acres of arid land in the vicinity of Fort Collins, Idaho. The length of the tunnel between portals will be 1,300 feet when completed.

During the month of May, 1910, the Laramie-Poudre tunnel, using No. 7 water Leyner drills, was driven from two headings a distance of 845 feet. The heading from the west end was advanced 332 feet, while the heading from the east portal was advanced a distance of 513 feet, thus breaking all Colorado records for a single heading in a calendar month.

In January, 1909, the Cripple Creek drainage tunnel was driven 435 feet, and in January, 1908, one of the headings in the Gunnison tunnel made 449 feet, and in October, 1908, the Elizabeth tunnel, near Los Angeles, was driven 466 feet. The rock in the Cripple Creek drainage tunnel and in the Laramie tunnel is very much the same, being very hard, firm granite in each; whereas in the Gunnison and the Elizabeth the rock was very much softer, and in the latter quite a large proportion of it was shale. The rock through which the Laramie tunnel is being run is so hard that 27 holes have to be drilled in the face of each heading and loaded with the most powerful modern explosives to break it successfully.

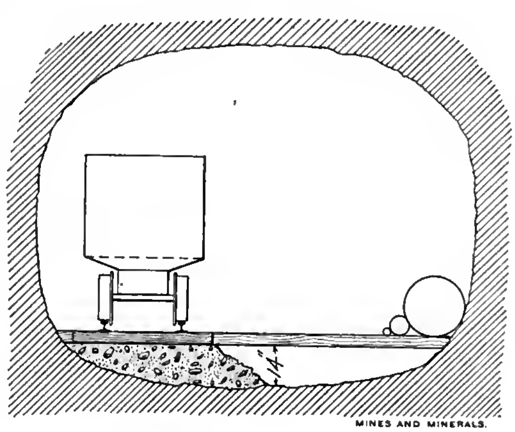


Fig. 1

The men work under the bonus system and receive an extra amount of pay for each foot driven over 250 in each calendar month and on such a record-breaking run as that of last month this bonus amounts to a very considerable sum, consequently both workmen and the contractor, Mr. James A. McIlwee, are happy.

The Laramie tunnel, Fig. 1, is driven 7 feet 6 inches high and 9 feet 6 inches wide and is nearly oval in form, thus obviating the necessity of taking out any corners and affording every possible facility for rapid driving. In fact, this was what the consulting engineer of the tunnel, Mr. D. W. Brunton, had in view in using this particular form, and in addition to this the undertaking was entirely new, no makeshift machinery had to be employed, so that in designing the big hydroelectric power plant there was nothing to be considered except actual conditions, to which every part of the plant was made to conform.

Five hundred and thirteen feet in a single heading in a calendar month is quite an achievement, but when the difficulties under which the work from the western end is carried on are considered, 332 feet is almost as good. The Laramie tunnel is driven on a grade of 1.7 per cent., sloping from the west toward the east, and consequently the material removed from the west end has to be hauled up grade, whereas on the east end the loaded cars run down grade and only empty cars have to be hauled up. In addition to this, the west end workings do not run directly out into the open air on grade, but are connected with the surface by a 200-foot incline. This was done to avoid about 1,400 feet of water-bearing rock and wash which, had it been opened up, would have drained directly down the tunnel into the heading. Even with this precaution a considerable amount of water has to be pumped, so that with the difficulties of pumping, uphill haulage, and hoisting through an incline, the 332 feet driven on the west end is a most creditable performance.

ZINC AND LEAD IN ARKANSAS

Written for Mines and Minerals, by Lucius L. Wittich

Development of New Region. Change From Tunnels to Shafts Admits Mining Deeper Deposits

The zinc and lead districts of Arkansas are attracting more outside attention than ever before and capitalists are taking a substantial interest in Arkansas properties, which should prove conclusively within the next year or two whether or not the mines of that region have been materially handicapped because of inadequate transportation facilities or whether they have failed merely because of the absence of mineral in paying quantities.

That transportation facilities have been bad cannot be denied. Many of the prospects have been miles removed from the nearest railroads; now, however, railroads are being constructed through the heart of the ore fields. Fuel in the form of pine timber has been plentiful and operators thus have been saved the trouble of shipping in coal. The roads as a rule, all through the northern part of Arkansas, are rocky and billy, and cannot be compared with the good level highways of the Joplin, Mo., district, and thus have made it difficult to haul concentrates to the railroads.

The counties of Boone, Marion, Baxter, and Searcy have seen more development possibly than any others. Newton County boasts of a number of promising prospects. Recently Washington County, farther to the west, is reported to have produced an exceptionally good zinc-blende mine. The outlook was sufficiently encouraging to warrant the operators coming to the Joplin district and purchasing a concentrating plant of 150 tons capacity, which has been removed to the new mine known as the Washington County Zinc Co. Mining at that point is conducted in an open pit, the blende occurring in brecciated chert almost identical to many of the shallow deposits of the Joplin field. And here it may be stated that the absence of breccias in Arkansas is given by many as the apparent reason for the non-existence of ore bodies of as great extent as those encountered in Missouri, Kansas, and Oklahoma. Where faults occur in Arkansas, slickensides in many instances have resulted, and such geological formation is not looked upon favorably as a place in which to find extensive ore bodies. Good mines, however, are promised with scientific development, and in the reports of the United States Geological Survey the Arkansas field is commented on favorably.

Unlike the ore deposits of the Missouri-Kansas-Oklahoma district, which are found largely in the Mississippian series of the Carboniferous system, the Arkansas ores occur chiefly in the Yellville formation of the Ordovician system, which is a much older sedimentation than the Mississippian. Above the Ordovician and below the Carboniferous is found the Devonian. Each system is divided into numerous formations, some being so thin that it is almost impossible for even an expert geologist to distinguish them.

The Yellville formation is composed of magnesian limestones, or dolomites. In the Northern Arkansas field these rocks are the oldest exposed, although the base of the system shows no outcropping and is estimated only by deep wells which indicate that it may extend down to the pre-Cambrian crystallines. Where the Yellville formation outcrops along the White River its width reaches 500 feet.

As the oolitic limestone is the guiding geological formation of the Joplin district, so is the Key sandstone a valuable guide, and is used as a datum by mining men throughout the Arkansas district. It is found in benches and often marks the cap rock overlying the dolomites in which the ore occurs. Fracturing, jointing, faulting, and slight brecciation characterize the areas where the mineral is most likely to be encountered. The ore, when a blende, is of a peculiar high grade which has distinguished the mines of that region.

In the Boone formation of the Mississippian series, some ore is found. In the Missouri-Kansas district this is the chief ore-bearing formation. The Key sandstone which forms the cap rock for the ore deposits of the Yellville formation is easily recognized through its resemblance to light brown sugar, and to its being easily pulverized.

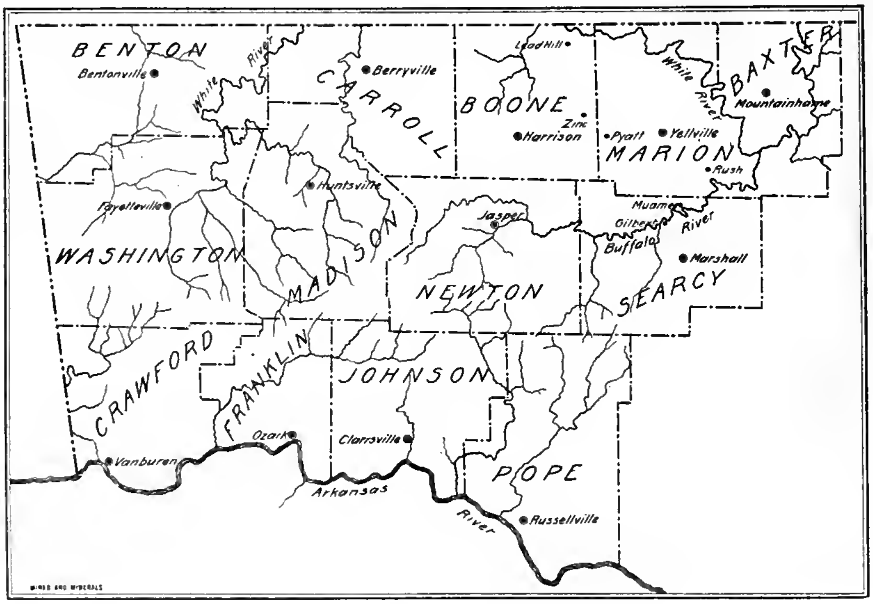


Fig. 1. Zinc Field in Arkansas

Much of the mining in the Arkansas district is done by tunneling into the mountain sides. In this way, many prospectors have sought the thinner deposits of the Boone formation in preference to the deeper ore bodies of the Yellville formation.

They would rather go up high on the mountains to mine than in the valleys, because there is less water to contend with; but, on the other hand, there is not so much ore.

One of the important steps which marks the gradual change of conditions in Arkansas was the recent purchase of the Red Cloud Mine, located in the Rush Creek district of Marion County, for $75,000. This mine has been under desultory development for 12 years, but did not produce ore until last year. Since the purchase of the mine, a modern concentrating plant of 150 tons daily capacity has been installed; a tunnel has been driven 400 feet in the mountain, and ore is being mined. Several hundred tons of concentrates have been produced within the past year. Farther down the creek this company is starting a second tunnel.

Across the Buffalo River from the Red Cloud Mine, the White Eagle Mine lies dormant, not because the ore body has been exhausted, but because the water proposition is one of the most troublesome in the entire Arkansas district. The White Eagle has a shaft 100 feet deep into a zinc ore body which has been followed for some distance, and which runs beneath the river.

The general tendency of Arkansas operators seems to be to break away from the custom of tunneling. Many companies that for years have experimented with tunnels are abandoning this system and are resorting to shaft sinking. A noteworthy example of this change is found at the Maumee Mine, 35 miles

southeast of Harrison. For years this company operated in a tunnel driven into the hillside. Now the company has sunk a shaft to a depth of 140 feet, from which levels are being driven into ore showing uniformly a height of 14 feet. This phase of the development is 300 feet deeper than the original tunneling operations. This company also has adopted prospect drilling, and out of 17 holes put down ore is reported to have been found in 15. The drifts are now out 230 feet from the shaft and considerable blende is in the crush pile. This mine is handicapped by being 8 miles removed from the nearest railroad, the Missouri & North Arkansas, which passes at Gilbert.

The concentrating mill of the Maumee Mine is shown in Fig. 2. It is so situated that pine is abundant and fuel does not have to be carted from the railroad station.

Among the new properties of the district is the Coker Hollow Mine, located near Zinc Station, on the Missouri Pacific Railroad. The discovery of ore was due to an outcropping in a clear, cold branch which bubbles through the valley. At a depth of 3 feet the mineral was found in big, free chunks, and continued to a depth of 75 feet. At the 35-foot level a drift proved the ore good for a distance of 45 feet. At the 65-foot level another drift was put out into ore. A 100-ton concentrating plant at this mine is just ready for operation. It is estimated that from 800 to 1,000 tons of blende-bearing rock is deposited in the stock pile.

That the epoch of prospecting is fast giving way to one of actual milling development is still further indicated by improvements at the Tar Kiln Mine, near Zinc Station, which is one of the pioneer 'diggings" of the district, but which was not equipped with a concentrating plant until recently. A 100-ton mill has just been completed. The mine is in Boone County. A number of drill holes show ore and a shaft is in mineral.

Fig. 2. An Arkansas "Wood Burner" Concentrator at Maumee Mine

One of the mines that actually has produced ore, both zinc and lead, is the Pilot Rock, 6 miles from Pyatt, the nearest station, which is on the Missouri Pacific Railroad. Ore was shipped from the Pilot Rock years ago, being barged down the White River. A company of Ohio capitalists has secured possession of the property and has sunk four shafts into ore, and a number of drill holes also have been put down.

Numerous other new developments are being made. At the Black Bear Mine the ore is carted to the Susquehanna Mill for treatment. The Madison Mine is a heavy producer of calamine, the ore being taken from a tunnel which has been driven 500 feet in the mountain. The Iola also has a tunnel 500 feet into the mountain, but is now working on a shaft which is to penetrate ore at a deeper level, according to drilling clippings. The North Star Mine, 3 miles from Harrison, in Boone County, is considered one of the best tunnel propositions in the district, but the owner, it is said, will not permit further development now that he knows what his property contains.

Long before the existence of zinc ore in Arkansas was realized, lead ore was mined extensively. Early records show lead ore was mined in 1818, the first settlers having used the product for rifle bullets. Until the early fifties little effort was made to produce the lead in commercial quantities. In the vicinity of what was known as Lead Hill, in the northeastern part of Boone County, the first Arkansas ore was smelted. In the seventies the smelting of lead was resumed after an idleness of many years. A few years later the works were abandoned.

The first production of zinc ore for commercial purposes occurred in 1857 when calamine was taken from a mine at Calamine, in Sharp County. Active prospecting for zinc ore began in 1886. In 1899 there was a light rush to the Arkansas fields but no extensive development was undertaken.

ORE MINING NOTES

Cobalt, Ontario.—Coniagas mine has 12 veins, only one of which is being worked at a depth of 170 feet from the surface. The Coniagas mill has been a success from the start, and produces three cars of concentrate per month which averages $1,000 per ton.

In the Cobalt Lake Mine about 15 feet from the McKinley-Darragh boundary, 13 stringers of ore were found in the face of the drift. Two of the stringers are 3 inches wide. Three assays of the smaltite in these stringers ran between 2,000 and 3,000 ounces.

Surface prospecting on the Nipissing has uncovered veins Nos. 34 and 35. They run parallel and are about 30 feet apart.

While the Cobalt Central is in liquidation and suits and cross-suits are being filed, the mill is running and mining carried on. The Bailey-Cobalt, a creditor of the Cobalt-Central, to the amount of $22,000, for ore mined, has asked that liquidation proceedings be set aside.

Those patriots who were feeling sore over the way the Canadians have been putting it over United States politicians can cheer up. The following shows how they are copying us:

"The Island Smelting and Refining Co., limited to $3,000,000, has been incorporated by Torontonians. The company controls the patents on the smelting process invented about 5 years ago by Doctor Island, formerly a dentist in Toronto. It is stated that the process will revolutionize the smelting industry, as it is available for treating low-grade and refractory ores with commercial success. It has been submitted to the leading chemists of the United States and pronounced a wonderful success. The demonstrating plant shows that metal in all classes of ore could be extracted with a loss of less than 1 per cent. and the residue made use of for commercial purposes."

This language is so familiar that all that is necessary is to read the line commencing "it is stated," then shut the eyes and declaim the rest adding, when finished, "fake."

The output of the Alaska-Treadwell mines will be $3,500,000 this year according to Superintendent R. A. Kinsie. Mr. Kinsie said that the explosion of the magazine in the 1,100-foot level some time ago has not affected the output and the company has experienced no difficulty in keeping its men. The cause of the explosion, in which seven lives were lost, has never been determined.

There has been a marked cessation of diamond drilling operations in all the Lake Superior iron-ore fields. Many drilling outfits have been transferred from the iron to the lake copper district where exploration for new lodes is actively in progress.

"Mr. Kelly who desires to be Governor of Michigan, wants the mines of iron ore and copper in his state to pay a much larger tax than now. Anything that adds to the cost of producing iron and copper must be paid out of the earnings of the mines. The farmer cries for free steel and iron manufactures. Then give us free farm produce. One family in Ishpeming pays more for farm produce in a year than the farmer pays for product of the mines in five."—*Iron Ore.*

The Oliver Iron Mining Co. has secured options on 313,000 acres of mineral and timber lands in the upper peninsula in Michigan, also 111,640 acres of mineral rights. According to the contract the Oliver company is to do exploration work on the property, and for all ore discovered and mined, is to pay a stated royalty to the Michigan Iron and Land Co.

The anticipated movement to secure a radical change in the management of the Lake Copper Co. has crystallized. The plea is a more aggressive policy. The present management has gone slowly, believing that to be the safer policy at least until the strike and course of the big lode in the company's lands becomes known.

The Quincy Copper Co. has done remarkably well when the percentage of copper to the ton of rock is considered. It cannot be expected to pay big dividends in the face of a copper market universally acknowledged to be too low to pay a fair return for the money invested and the labor required for production.

To reduce the cost of making metal at the Greene Cananea Copper Co.'s operations in Mexico, electricity is being introduced in the mines, mills, and smelters. A new reverberatory furnace is being added to the smelting plant and the old concentrator is being overhauled and made more effective. Greene Cananea is in possession of rich mineral ground and should be a money maker even under prevailing conditions with low copper prices. The Utah porphyries are not in so favorable a position as regards ore, and they are said to be making money.

The first annual report of the Lake Copper Co. shows that from 8,109 tons of rock there was a yield of 21,064 pounds of refined copper.

The Clark mines in Butte, Mont., have been taken over by the Anaconda company. From this time on the ores will be shipped to the Washoe smelter, at Anaconda, for treatment. The plan is to clean up all the ores at the mines, then to close them until such time as they can be worked in conjunction with the old Anaconda properties, the plan being to put things into condition so a saving in cost of operation may be secured. Anaconda is working at full pressure, producing about 25,000,000 pounds monthly at the two smelters of the company. The Clark mines have been yielding about 1,500,000 pounds at the Clark smelter in Butte.

The reduction in the quarterly dividend of the Calumet & Hecla Copper Co. from \$8 to \$7 is a direct result of the poor copper market, and poor outlook. This places the stock on a \$28 per annum basis. The Calumet & Hecla dividend is becoming sensitive to the price of copper. In 1900 the conglomerate below the 57th level yielded about 60 pounds of refined copper to the ton of rock; for the fiscal year the percentage has dropped to about 35 pounds per ton of rock.

There is considerable talk concerning the merging of the Superior and Pittsburg with the Calumet and Arizona companies in Arizona. When overtures were made by the Superior and Pittsburg to bring about consolidation with the Calumet and Arizona, the latter would have none of it, because Superior and Pittsburg was pumping water. Calumet and Arizona has a smelter and a good mine but a rather restricted area. Superior and Pittsburg has developed a large area of producing ground and a territory that will last a long time; it is also producing at the rate of \$110,000 per month above all expenses, even in this time of low copper prices.

It is interesting to note the record which the Calumet and Hecla, of Michigan, has made as a copper mining proposition. Figuring the past year's production at between 70,000,000 and 80,000,000 pounds the total output of the Calumet proper since organization has been approximately 2,360,000,000 pounds of refined copper, or about 45 per cent. of the total production of Lake copper. From this output the company has paid, including the latest payment, \$112,050,000, which represents slightly more than five cents for every pound of the metal which the company has produced. This compares with an average payment per pound of all lake companies of less than 3.70 cents per pound.

The directors of the Davis-Daly Copper Co., Butte, Mont. have levied an assessment of \$1 per share on the stock payabl in two equal instalments July 15 and September 20, 1910.

The Snowstorm Mining Co. has produced 500,000 tons of copper-silver ore since 1904 and has paid \$959,000 in dividends to date. The Snowstorm Mine occupies the central position in the copper belt east and north of Mullan, Idaho. Other properties in the belt are Lucky Calumet, Missoula, Independent, Pandora, Snowshoe, Snowstorm Extension, and East Snowstorm.

Years ago the old Planet Mine near Parker, Ariz., was a famous shipper of high-grade copper ore to Swansea, Wales. In the early work no attempt was made to prospect the ground below the 100-foot level. The Lewissohns, of New York, who took over the old Planet company, about one year ago, have been prospecting and have a shaft down 600 feet with large deposits of copper ore blocked out.

The Precious Metals Mining Co.'s work on the Atlanta lease in Goldfields, Nev., has developed some mineralized quartz on the St. Ives vein. Drifting from two different points on the 750-foot level is now going on.

The Atlas Leasing Co., which has taken over the Jack Pot ground for 25 years, has postponed the new work in order to give the outside stockholders of the Jack Pot Co. an opportunity to trade in their stock for shares in the Leasing company. The plans of the company include the installing of the big hoist, the sinking of the new shaft, and the building of a mill, provided the Jack Pot stockholders will come across.

The Utah Copper Co. is now established in the 100,000,000 pounds per annum class of producers. The merged Anaconda comes first, the Phelps-Dodge Co. second, and the Utah Copper Co. third.

What may prove another gold strike is reported about 8 miles south of Gold Dyke, Nev. Since the discovery, a month or two ago, there has been a rush of Manhattan people to the place and a number of San Francisco people have gone in on telegraphic communication from their Manhattan correspondents. Probably everything is "sewed up" by this time and stock companies are in order.

In cleaning out a prospect hole on the Golden Crown claim, Twin Peaks, N. Mex., a vein of ore was uncovered which panned well and showed free gold. A shaft when sunk 40 feet showed a vein 7 feet wide that had more free gold than that at the top. The ore is telluride.

Prospecting for zinc and lead ores is being extended to the remote camps of the Joplin, Mo., district. Drill rigs are working in localities never before mined, and occasionally reports are being received of encouraging finds.

In the Sarcoxie district a number of holes have been put down on the Boyd land, and according to reports the extent of the conglomerate ore mass found in a number of shafts in that district has been proven great. At the Lone Pilgrim Mine operations have been extended to the stage requiring a mill, and until a plant has been secured work is temporarily at a standstill.

The open-ground propositions as a rule do not require extensive milling equipment; in fact, the largest profits frequently are made from mines where the milling capacity is small. The soft ground mine, as a rule, will not furnish sufficient dirt to keep a large mill running continuously. In order to keep a large concentrating plant busy it is necessary to produce steadily from three or four shafts; while with a smaller plant, built at much

ess expense, an equal tonnage of cleaned ore will be turned out. This feature accounts for the erection of so many plants that appear to be inadequate for the work required of them.

Among the Tintic, Utah, mining companies to declare dividends were the Colorado and the Iron Blossom. The former paid 8 cents per share, the latter 6 cents per share. The Grand Central and Victoria, both of this district, passed their dividends.

The rich body of ore recently uncovered on the property of the Big Elk Mining Co., near Wallace, Idaho, located a half mile west of the Chicago, Milwaukee & Puget Sound tunnel, has been developed to a considerable extent, and the shaft which is being sunk on the ore is now down about 10 feet and the floor is all in ore. The ore is of a high-grade copper which will assay from 20 to 35 per cent. The property has been in the limelight recently on account of ore being uncovered by a crew of men working on the Milwaukee railroad, who were filling in trestles in that vicinity with hydraulic pressure.

Reports from the Mudhole mine of the Penn-Arizona Mining Co. state, that at a depth of 750 feet, a rich body of ore was discovered by a cross-cut. The gold runs from $50 to $80 per ton and three Huntington mills are working on the output.

Lessees on a block of ground on the Ibex mining property, Leadville, Colo., have been working with indifferent success for some time past. They took out a 13-pound nugget a few weeks ago that encouraged them, and keeping on they encountered a piece of ore that looked different from the ordinary run. It seemed to be one piece and when broken and placed on the company scales weighed 66 pounds. According to the graduated scale of royalties maintained at the Ibex the owners of the mine receive 65 per cent. of the proceeds from this kind of ore. The lessees' share of this amount—allowing them 35 per cent.—is therefore $1,435.20. The ore was found in the porphyry about 1,000 feet from the place where the nugget was found and in a stope that was thought to have been worked out 15 years ago.

A new daylight mine is being opened in the Galena camp, Mo., by H. B. Savage, of Galena; H. F. Smeltzer, of Kansas City; and John H. Estes, of Richmond, Mo., who have a lease on 12 acres of the Windsor Mining Co.'s land. The property is known as the Mess Mining Co.

The company has broken in the ground from the surface and opened one of the old drifts leading from the mill shaft of the property. The shaft is 75 feet deep, but at the 55-foot level a platform has been built over the opening. A short distance to the east the ground is opened to the surface, the ore at this point running to within 10 feet of the top.

TESTING FOR PROTECTIVE ALKALINITY

*By Bede Collingridge**

The results of experiments recently made by me show that potassium iodide influences the testing for "protective alkalinity" in cyanide solutions. As far as I can ascertain, this fact is not mentioned in standard textbooks now in use.

The experiments which I carried out show the influence exerted by potassium iodide to be important, especially with regard to cyanide solutions which contain no protective alkali, for in testing these in the presence of potassium iodide they show protective alkalinity.

Cyanide decomposes much faster in solutions which are deficient in protective alkalinity than it does when protected by an alkali; therefore, the method I give of testing without potassium iodide will mean a considerable saving of cyanide on large cyanide plants, as the method generally used is to test the cyanide and protective alkalinity in the one measured portion.

In Clennell's "Chemistry of Cyanide Solutions" (page 63), he says: "* * * so that the same measured portion of the liquid to be tested will serve for the determination of both cyanide and protective alkalinity."

The method which I have used for these tests is as follows:

Into 10 cubic centimeters of the test solution, using potassium iodide as indicator, run standardized silver nitrate until a deep brown coloration of iodide of silver shows clearly, and note number of cubic centimeters of silver nitrate taken.

Now using another 10 cubic centimeters of the test solution, without adding potassium iodide, run in twice the quantity of silver nitrate that was necessary for the above test, to make sure of getting rid of the alkalinity due to cyanide.

Add phenol phthalein and titrate with standardized acid. The result will be protective alkalinity.

A second method I have used, which I consider gives clearer results than the foregoing, and which also does away with the first part of the previous test, is as follows:

Into the measured portion of test solution, to which has been added the indicator, phenol phthalein, run in silver nitrate, until the indicator color remains constant, then titrate with standardized acid in the ordinary way. With a little practice this gives very good results.

These methods (potassium iodide being absent) will immediately show any deficiency of protective akalinity, and this deficiency may be easily remedied, and so stop rapid decomposition of cyanide by the addition of a suitable alkali.

The chemicals I used for these experiments were: Commercial sodium cyanide, 126 per cent.; pure potassium iodide, 10-per-cent. solution.

The standardized solutions, using 10 cubic centimeters of test solution, were: Silver nitrate, 1 cubic centimeter = .01 per cent. KCN; hydrochloric acid, 1 cubic centimeter = .001 per cent. alkalinity; caustic soda, 1 cubic centimeter = .001 per cent. acid.

Indicator, phenol phthalein.

For the tests a solution of the above-mentioned commercial cyanide was made up showing .117 per cent. cyanide, 10 cubic centimeters of which were used for each test.

In the following table, 10 cubic centimeters of the test solution were taken for each test, and tested for protective alkalinity by the first of the foregoing methods. The results were:

I. Without adding KI, protective alkalinity, per cent .0033, clear end reaction
II. With ½ cubic centimeter KI, protective alkalinity, per cent. .0046, indistinct reaction.
III. With 1 cubic centimeter KI, protective alkalinity, per cent. .006, indistinct reaction.
IV. With 5 cubic centimeters KI, protective alkalinity, per cent. .0095, indistinct reaction.

It will be noticed by the above table that the error in testing protective alkalinity increases with the increase of potassium iodide.

In the next table the same test solution was used as in that previous, but each 10 cubic centimeters used was made slightly acid as regards protective alkalinity by adding 3.5 cubic centimeters standard hydrochloric acid; 10 cubic centimeters of the acidulated test solution were taken for each experiment and tested for protective alkalinity by the method given, when the following results were obtained:

I. Without adding KI, protective alkalinity, per cent. nil, clear end reaction.
II. With ½ cubic centimeter KI, protective alkalinity, per cent. .002, indistinct reaction.
III. With 1 cubic centimeter KI, protective alkalinity, per cent. .0037, indistinct reaction.
IV. With 5 cubic centimeters KI, protective alkalinity, per cent. .0068, indistinct reaction.

The results of the foregoing table prove that solutions which contain no protective alkalinity give tests for protective akalinity after the addition of potassium iodide.

From these experiments it will be seen that potassium iodide must be absent when testing cyanide solutions for protective alkalinity.

I have obtained similar results from ordinary working cyanide solutions.

*A paper discussed at a meeting of the Institution of Mining and Metallurgy January 20, 1910.

THE YAMPA SMELTER AT BINGHAM

Written for Mines and Minerals, by Leroy A. Palmer

A Description of the Process by Which the Low Grade Copper Ore Is Smelted

The Yampa smelter is concentrating the lowest grade copper ore in the country and the resultant matte is the lowest in copper content of any of the smelters now in operation. An average of the ore during a recent month shows copper *Cu* 1.92 per cent., iron *Fe* 27.9 per cent., silica SiO_2 29 per cent., lime *CaO* 3.1 per cent., alumina Al_2O_3 6 per cent. Some gold and silver occur but their value is so small it is not counted on to bear any great portion of the expense of mining and concentrating. It will be seen that the per cent. of copper in the furnace charge is but little more than the Bingham Cañon copper porphyries which are only profitable when concentrated, and are out of the question as a direct smelting proposition on account of the silica they contain.

The Yampa smelter is at Bingham, Utah, about a mile and a half from the entrance to Bingham Cañon and slightly over 2 miles from the Yampa Mine which supplies the greater portion of the ore, although some custom ore is treated. The smelter is placed on the side of the cañon where it is rather steep for such a plant. This site was chosen when the plant was first built and was not expected to assume its present proportions. It has the advantage of being connected with the mine by an aerial tramway operated by gravity and of being so situated as to minimize the danger of injuring the ranches in the valley by fume. The site was all right for the small original smelter, but since the plant expanded it has presented disadvantages relative to the situation of the buildings to each other. These disadvantages, however, do not seem to be sufficient to overcome the advantage of proximity to the mine and freedom from adverse litigation.

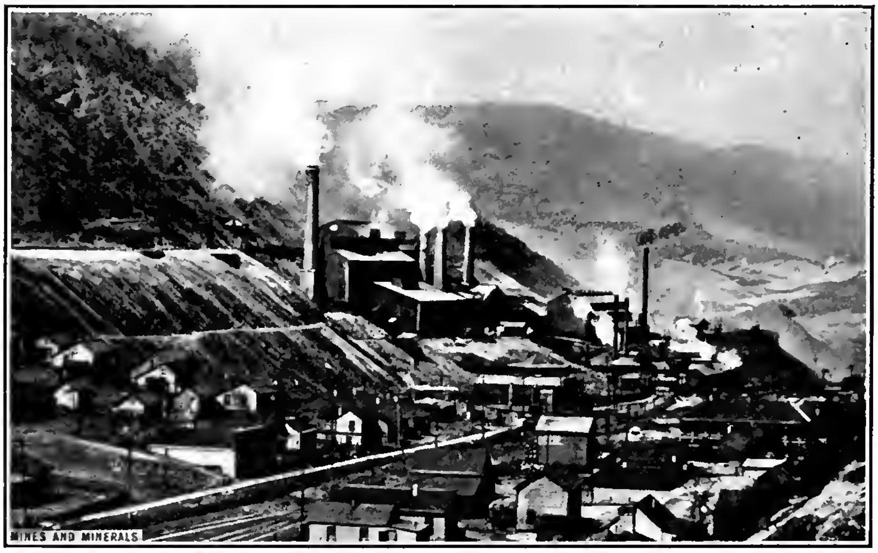


Fig. 1. Yampa Smelter

Sampling Mill.—The ore comes from the mine over a Bleichert aerial tramway in 1,500-pound buckets, which are dumped by hand to two slope-bottomed ore bins, having a capacity of 500 tons. The Yampa ore is no longer sampled as a guide to making up the furnace charge, and only a hand sample is taken from the charge cars. The custom ore is received over a standard-gage railroad spur, and dumped to a bin between the two stock bins. The bins are all lined with iron, 12-pound rails being used where the ore falls.

The custom-ore bin is provided with two hand-operated rack-and-pinion gates, and discharges direct to a 10″×20″ Blake crusher set to break to 2 inches. From the crusher the ore falls to the No. 2 elevator which is belt driven and has a lift of 75 feet and a speed of 350 feet per minute. The elevator discharges the ore to a set of 14″×28″ Cornish rolls having a speed of 95 revolutions per minute, and set to crush to ¾-inch ring. The ore from the rolls falls to the No. 3 elevator, which is also belt driven and makes a lift of 30 feet with a speed of 350 feet per minute. At the elevator dump is a Vezin sampler, which makes a one-tenth cut, the sample reject going to the boot of the No. 1 elevator. The sample is coned and quartered until about a wheelbarrowful is left when it is recrushed in a laboratory crusher and further cut down to 4 or 5 pounds.

The Yampa ore goes from bin 1 direct to trommel 1, and from bin 2 to the trommel, by a 22-inch horizontal rubber belt conveyer having a speed of 160 feet per minute. This trommel, which has a speed of 18 revolutions per minute, is 3 ft.×8 ft. with a frame of four 6-inch angle irons over which is a plate of half-inch steel in which are punched holes. Different screen plates are used so that the revolving screen has a mesh varying from one-half to 1½ inches, according to the kind of the ore received from the mine. If there is a plentiful supply of fine ore, the half-inch screen is used and as much as possible thrown into the oversize. If the ore is largely coarse and not sufficiently fine to supply two reverberatory furnaces, the coarser screen is put on and the undersize crushed to pass a three-quarter-inch mesh, and is then sent to the reverberatories.

The oversize of the first screen goes to a 30-inch rubber belt conveyer with a speed of 140 feet per minute, running at an angle of 9 degrees and dumping to the blast-furnace bins. The undersize goes to the No. 1 elevator which is gear-driven with a 14-inch belt and makes a lift of 90 feet with a speed of 285 feet per minute. This elevator discharges to trommel No. 2 which is 3 ft.×8 ft. with ¾-inch punched holes, making 22 revolutions per minute. The undersize goes to the roaster bins, and the oversize is chuted to the discharge of No. 2 elevator from which it goes through the rolls thence by No. 3 elevator to the boot of No. 1, by which it is again elevated to trommel 2.

A 40-horsepower induction motor drives the sampling mill.

Roasting Furnaces.—The fine ore is loaded from sector gates to cars of 20-cubic-feet capacity and trammed a distance of 50 feet to the top floor of the roaster building, which is on the same level as the loading floor of the sampling mill. Here it is dumped to the roasters each of which is provided with a 1-inch grizzly to keep out any coarse ore that might become mixed if the No. 2 trommel was broken or clogged.

In the roaster building are nine 18-foot McDougall roasting furnaces, in two rows. Each furnace has 6 hearths with water-cooled rabbling arms, except the two upper hearths where the heat is low and the water is not needed. The rabbling arms make a revolution every 58 seconds, and 1 hour and 15 minutes is required to work the charge through the furnace. Each roaster has a daily capacity of 47 tons and eliminates about 83 per cent. of the sulphur in the ore. In one row four of the roasters are provided with 24-inch steel elbows, two to each furnace, for the discharge of the fume. Two of the furnaces discharge through brick stacks 40 inches square, to a chamber 6 ft.×6 ft.×66 ft., connected with the main dust chamber. The other three roasters discharge direct to an **L** shaped chamber 4 ft. ×7 ft.×37 ft., connected with the main chamber 20 ft.×32 ft. 8 in.×60 ft., provided with two brick baffle walls. The main dust chamber discharges to a brick stack 120 feet high with an inside diameter of 10 feet at the top. Beneath the dust

chamber are two steel hoppers 10 ft. × 10 ft. with 12 in. × 12 in. lever-operated slide gates, for loading cars which are run beneath. The McDougall dust assays on an average: Copper 2.36 per cent., iron 30.5 per cent., silica 25.8 per cent., sulphur 21.2 per cent., lime 2.6 per cent., alumina 7.4 per cent.

The accretions which form on the sides of the roasters are barred off and trammed to a bin from which they are mixed with the charges going into the blast furnace.

A 20-horsepower direct-current motor drives the McDougall roasters.

Reverberatory Furnaces.—The roasted ore is discharged from the McDougall roasters into cars, having a capacity of 7,300 pounds each, and hauled by a motor to a building adjacent, where there are three reverberatory furnaces, two of which are in continuous operation. The furnaces (a drawing of which is shown in Fig. 2) are 17 feet wide, 47, 53, and 59 feet long, each with a rated smelting capacity of from 150 to 175 tons of raw ore per day, but they actually treat much more. The fireboxes *a* are of uniform size, 9 ft. × 12 ft., divided into two compartments by a brick wall *b*, so that each compartment is 5 ft. 6 in. × 9 ft. Each side of the firebox is connected to a 15-inch air line, while the fuel is fed from the roof through three 15-inch openings *c*. Draft for the fireboxes is supplied by a 16-inch centrifugal blower at a pressure of considerably less than 1 inch of water. A 20-horsepower induction motor drives the blower.

The furnaces are of brick, stayed with 4-inch I-beams tied across the top and spaced on 16-inch centers, except where two doors *d* 15 in. × 24 in. have been left in each side to allow the furnace men to feel the charge. A 12″ × 15″ door *e* is in the back of each furnace for skimming.

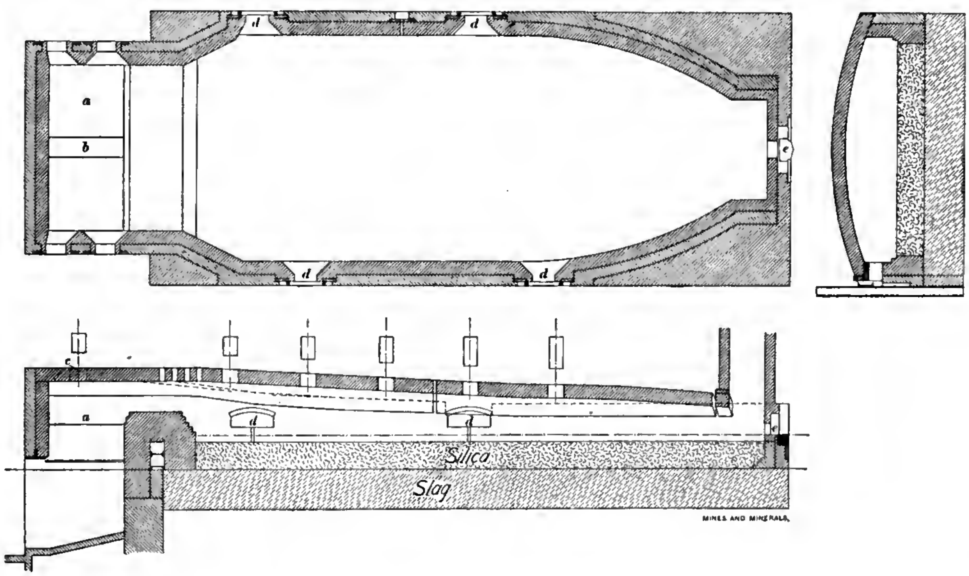


Fig. 2. Reverberatory Furnace

Between each furnace and the stack is a 300-horsepower water-tube boiler utilizing the waste heat from the furnace to generate steam at 110 pounds pressure. These boilers deliver an average of 180-horsepower, effecting thereby an economy equal to 150 tons of coal for each furnace per month. This 180 horsepower represents an efficiency of only 60 per cent., which is small. One reason for this is that in planning the installation it was deemed advisable to put in larger boilers than there would be heat to operate at full capacity, because there was room and the increased radiating surfaces presented made it possible to get a greater horsepower than would be possible with boilers with nominal ratings corresponding to the quantity of heat thrown off by the furnace. The last boiler installed is of a later model than the others and has shown a higher efficiency, due largely to the fact that the tubes are staggered while in the older models they are in regular rows. The later models of Stirling boilers seem to be most favored by smelter men for this work and in one plant this style of boiler is used in a furnace which generates heat proportional to the rating of the boiler, which in turn delivers practically the full rated boiler horsepower. The consumption of coal at this plant is somewhat higher than at the Yampa and it may be that some of this goes toward making steam. The Yampa waste-heat boilers are fed by a 7″ × 4½″ × 8″ duplex Snow piston pump, and a 5″ × 5½″ × 8″ Smith-Vaile plunger pump is held in reserve. An 8″ × 5″ × 12″ Fairbanks, Morse & Co. piston pump furnishes water at high pressure for fire protection and for operating the turbo boiler washers. Each furnace discharges through a brick stack 100 feet high with an inside diameter of 7 feet at the top.

No dust chambers are provided for the reverberatory furnaces, the fumes passing from the boilers directly to the stacks. The boilers are cleaned about every 50 days and as only a ton of dust accumulates in this time it is taken as sufficient indication that the loss from the stack is so small as to be negligible.

The question of the use of forced or open draft for reverberatory furnaces is one that has been much discussed, therefore a comparison of two plants is of interest. The plant used for comparison is of late design and in this respect has many points the advantage of the Yampa. It uses about one-half ton of good lump coal per ton of ore treated; the ashes from the furnace are extinguished and sent to the power house for reuse under the boilers, as they contain about 60 per cent. of the dry weight in good coal. The wet ash makes it necessary to use forced draft on a hand-fired boiler, and so as far as the question of draft is concerned it seems to be only whether it will be used at the furnace or in the power house. The Yampa reverberatory uses one-fourth ton of run-of-mine coal per ton of ore treated and the coal is so thoroughly consumed that an attempt to burn the ash resulted in smothering the fire. Thus the rehandling of the ash is eliminated. Once a shift, the grate bars each side of the furnace firebox are cleaned, during which time the draft is shut off from that side. This cleaning requires about half an hour for each side, and while it is going on about half of the pressure is lost from the boiler connected with the reverberatory, making it necessary to fire more heavily at the power house. When the cleaning is finished the steam rises at the power house and blows off for some time, which of course represents a certain amount of coal from which no benefit is derived. The actual amount of coal consumed at the Yampa is .83 ton per horsepower month, while at the newer plant it is only .78 ton, but the superiority of equipment of the latter should make this difference greater as it is using a late model of Babcock & Wilcox boiler in the power house, while the Yampa boilers are of the return tubular type and quite old.

The forced draft in the reverberatory furnishes almost complete combustion of the coal, so that the charge is rapidly smelted and the capacity of the furnace is increased from 150 to 175 tons of raw ore per day, to from 217 to 220 tons per day. This is accomplished with a thorough fusing of the charge so that the quality of the matte and slag is not affected.

The matte is tapped once a shift and sometimes not so often, the low grade of the ore making it slow to accumulate. The slag is skimmed two or three times a shift through the door *e*

to an iron launder which discharges to a slag car with a capacity of 60 cubic feet. In these the slag is hauled to the slag dump, by a Jeffrey locomotive. In the slag launder is a section 6 feet long made 8 inches deeper than the remainder of the launder. Should the skimmings contain matte, or an unfused piece of silica come out bringing some matte with it, the deeper portion of the launder acts as a settler to catch the escaping matte, the principle being the same as the settling or "monkey pot" used at some blast furnaces.

The matte is tapped from the reverberatory to a series of iron molds 9 in.×20 in.×48 in. in the floor. An anchor with a ring is set in the mold, as shown in Fig. 3, before casting. When cool the matte is raised by a chain block, carried to a slag car and unloaded by striking it a blow with a sledge that breaks the matte from the anchor. The cars are dumped to an iron-lined bin from which the matte is loaded into the railway cars.

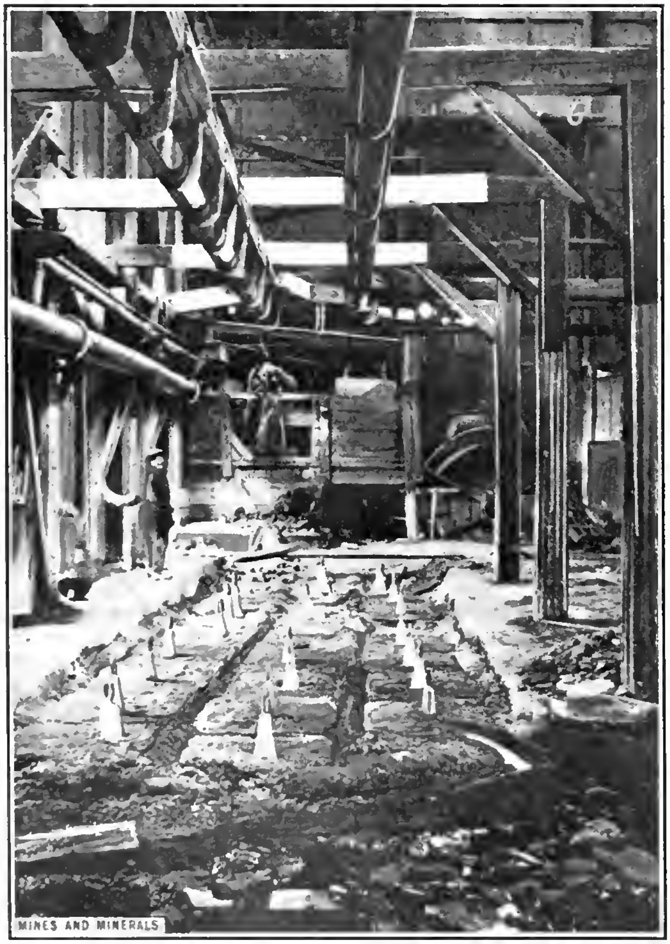


Fig. 3. Matte Molds Before Casting

A month's average of reverberatory matte assayed: *Cu* 30 per cent., *Fe* 36 per cent., *S* 26 per cent.; while the slag assayed *FeO* 39 per cent., SiO_2 40 per cent., *Ag* 1 ounce, *CaO* 3 per cent., Al_2O_3 15.5 per cent.

Blast Furnaces.—As will be seen from the analysis, the Yampa ore needs an addition of lime to make it a favorable smelting mixture. The process of making up a charge of Yampa ore for the blast furnaces is therefore a very simple one. The charge cars are drawn in front of the coarse-ore bins and loaded with 4,000 pounds of ore and 1,000 pounds of lime rock. To this charge is added an average of 525 pounds of coke so that the fuel is about 10.5 per cent. of the charge, as a rule. An electric locomotive hauls the cars to the blast-furnace building about 100 feet distant, where the charge is dumped through upward sliding doors, as shown raised in Fig. 4. The charge floor is 16 feet above the tuyeres. The slag that is barred out from the launders at the reverberatories is mixed into the charges dumped into the blast furnace. This amount which is only about 4 tons per shift serves to keep the furnace open and also affords an easy way of recovering the small quantities of matte that it contains.

The plant has three blast furnaces of the following sizes: No. 1 is 44 in.×180 in. at the tuyeres and 66 in.×180 in. at the shaft; the ratio of shaft to tuyere area is therefore $1\frac{1}{2}$ to 1. It has three water-jackets on each side and one at each end; eighteen 3-inch tuyeres 10 inches apart on each side with Siamesed tuyere pipes, individual valves and all metal connections to an 18-inch bustle pipe. The forehearth is circular, 2 ft.×11 ft. and the fumes are discharged through a brick downcomer 48 in.×102 in. in size.

No. 2 is 42 in.×168 in. at the tuyeres and 72 in.×168 in. at the shaft; thus the ratio of shaft to tuyere area is $1\frac{3}{4}$ to 1. It has three water-jackets on each side, and one at each end; also eighteen 3-inch tuyeres 9 inches apart with individual valves and all metal connections to a low 18-inch bustle pipe. The forehearth is circular, 3 ft.×12 ft., and the downcast is of brick 72 in.×92 in.

No. 3 is 44 in.×186 in. at the tuyeres and 66 in.×186 in. at the shaft, giving ratio of shaft to tuyere area of $1\frac{1}{2}$ to 1. It has six water-jackets on each side, and one at each end; and is supplied with sixteen 5-inch tuyeres 11 inches apart, controlled in groups, and with metal and canvas connections to an 18-inch bustle pipe. The forehearth is circular 3 ft.×16 ft. and the downcomer 72 in.×118 in. area.

Only one of these furnaces is operated at present, but it is handling about 300 tons of material daily. Blast is received at 32 ounces pressure through the main air line which has a diameter of 40 inches.

The blast furnace is tapped continuously, the slag being handled in 30-cubic-foot slag cars which are hauled to the dump by motors. The matte is tapped to molds in the floor casting into slabs 28 in.×44 in.×4 in., which are picked up by a chain block carried to a 6-inch grizzly and broken. Passing the grizzly they fall to an 18-cubic-foot car and are trammed to the railway bins.

The gases from the furnace discharge to a square dust chamber 20 ft.×20 ft.×400 ft. and thence to a steel stack 7 ft.×200 ft. This stack follows the slope of the hillside for 150 feet, only the last 50 feet being vertical. The dust is withdrawn from the dust chambers through iron doors 15 in.×18 in. spaced on 14-foot centers, to hooded cars having a capacity of 15 cubic feet. These cars are hauled up an incline by a small electric hoist to the top of the reverberatory building, and the contents charged into the hoppers. The dust analyzes 2.34 per cent. copper, 33.5 per cent. iron, 25.6 per cent. silica, 3.1 per cent. sulphur; 4.6 per cent. lime; 9.8 per cent. alumina. An average analysis of the blast-furnace slags for a month gave ferrous oxide *FeO* 22 per cent., silica SiO_2 44 per cent., sulphur *S* .7 per cent., lime *CaO* 4.6 per cent., alumina Al_2O_3 9.8 per cent. This slag is more silicious than would be practicable to make at many plants, but is giving good results and has the advantage of requiring less lime, with a corresponding increase in the amount of ore treated.

An analysis of the matte shows copper *Cu* 14 per cent., iron *Fe* 50 per cent., sulphur *S* 27 per cent. This is exceedingly low in copper. The company has a converter plant but it is closed down, and the matte is shipped to the Garfield smelter for converting. Before the United States plant at Midvale was forced to close its copper plant, the matte was treated there, but when this plant was put out of commission by a court order, the Yampa was forced to make other arrangements. Attempt to obtain a satisfactory arrangement with the Garfield plant failed, so the company built its own converter plant and for some time made blister copper. This was not satisfactory, as the returns

from a ton of matte were too low and the cost of converting as much as for a better grade. The Yampa matte made a desirable product for the Garfield plant where it could be mixed with the higher grade so an agreement was finally reached and the Yampa renewed shipping. No long-time contract has been made and the Yampa plant is in a position to resume converting at any time.

The question as to the advisability of following the Highland Boy practice, which will also be employed at the International smelter, at Tooele; namely, of crushing ore and treating it in the reverberatory furnaces, thus obtaining a matte of 30 per cent. copper, and then converting, was carefully considered in all phases, but rejected as the cost of crushing the matte fine enough for the reverberatories would more than offset the advantage of the company operating its own converters. There would be no saving in labor nor in treatment costs, while the slag losses from a reverberatory furnace are invariably more than from a blast furnace. A sufficient amount of ore is crushed fine to supply two reverberatories to one blast furnace, so the average matte is about 27 per cent. copper.

Power Plant.—The boiler house of the Yampa smelter contains four 85-horsepower return tubular boilers which generate steam at 120 pounds pressure. They are operated in conjunction with the waste-heat boilers and consequently are under steam only part of the time, the average for 5 months being one and one-half boilers per month. They are hand fired with slack coal, and during the 5 months mentioned 728 tons were consumed in generating 877 horsepower, an average of .83 ton per horsepower month. Considering the fact that the boilers are old and not of the most approved type this is a very creditable showing. A 7″×4½″×8″ Snow duplex piston pump is used for the boiler feed and a duplicate of this pump raises the water for the waste-heat boilers from the feedwater heater.

The steam from the waste heat boilers is carried to the engine room in a 10-inch line which is 300 feet long. The boilers are provided with dry pipes and there is a trap and separator where the main steam-pipe line enters the power house. The pipe is insulated with asbestos covering so there is very little radiation and although it is down hill no trouble is experienced from wet steam.

The engine room has the following equipment.

One 12″×30″ 110-horsepower Corliss engine direct-connected to a Connersville blower making 130 revolutions per minute, with a capacity of 130 cubic feet per revolution.

One 150-horsepower Bullock induction motor belted to a Connersville blower making 145 revolutions per minute with a capacity of 100 cubic feet per revolution.

One 10″×10″ 70-horsepower Ideal slide-valve engine belted to a Connersville blower making 165 revolutions per minute with a capacity of 55 cubic feet per revolution.

One 11″×12″ 75-horsepower Ideal slide-valve engine making 300 revolutions per minute, direct-connected to a 440-volt direct-current generator.

One 16″×16″ 175-horsepower Skinner slide-valve engine making 250 revolutions per minute direct-connected to a 440-volt direct-current generator.

One 30-horsepower 14″×14″×20″ Rand straight-line air compressor with a capacity of 75 cubic feet of free air per minute, compressing to 80 pounds for use in the shops, etc.

One 16″×34″×48″ 250-horsepower cross-compound Corliss engine, tandem connected to a 34-inch duplex air compressor with a capacity of 7,800 cubic feet of free air per minute. This was installed as a blowing engine for the converters, compressing to 15 pounds.

All except the Skinner engine are run condensing, being connected to a 16-inch Worthington barometric jet condenser with a 5″×12″×8″ dry vacuum pump which gives a vacuum of 20 inches, the mercury column at this altitude registering 22.64 inches. The exhaust from the Skinner engine is led to an open feedwater heater which raises the temperature from 54° F to 194° F. so that the feed enters the boilers at almost boiling point, which is 198° F. at this altitude. This increase in feedwater temperature is equal to 4½ pounds of steam pressure.

Three hundred additional horsepower are derived from the Telluride Power Co. This is received in the camp at 40,000 volts and stepped down at the power company's substation to 5,000 volts, at which tension it is distributed. At the Yampa it is further stepped down to 440 volts at the transformer house. Here are six 100-kilowatt transformers, three oil cooled and three water cooled, 300 horsepower being provided for an emergency. The transformer house is also equipped with three General Electric Co. 40,000-volt lightning arresters. The 300 horsepower regularly drawn from this source goes to a 300-horsepower 440-volt induction motor direct-connected to a 220-volt direct-current generator. This Telluride power is used only for such purposes as will not be affected by an interruption as the current comes into camp over very long transmission lines and is consequently liable to interruption in case of an electrical storm. The fires are kept banked under one of the boilers so that if the power from outside fails, there may be little interruption before starting one of the steam-operated generators.

Fig. 4. Furnace Charging Doors

Water Supply.—The water is derived from a shaft sunk on the premises which gives a minimum flow of 150 gallons per minute. By the use of various economies only about 50 gallons are used regularly. The shaft is provided with three pumps, only one of which is used at a time and that at only part capacity. These are two 200-gallon Cameron sinking pumps, each 14 in.×7 in.×13 in., and a 260-gallon Goulds single-acting triplex pump, 8 in.×10 in., belted to a 15-horsepower induction motor. The shaft pump discharges to a tank which also receives the water from the main cooling tower. The water is pumped from this tank to a 30,000-gallon tank on the hill back of the plant, from which it is distributed by gravity. This work is done by a 13″×12″ Aldrich single-acting triplex pump with a capacity of 900 gallons per minute, belted to a 75-horsepower induction motor. An 8″×10″ Goulds single-acting vertical triplex pump with a capacity of 260 gallons per minute, belted to a 25-horsepower motor, runs about 4 hours each day on this work. Three auxiliary pumps are provided that the flow of water to the distributing tank may not be interrupted. These are a 200-gallon 12″×7″×12″ Knowles; a 100-gallon 10″×5″×12″ Knowles, and a 200-gallon 14″×7″×13″ Cameron sinking pump.

The water runs by gravity from the roasters, water-jackets, etc. to the main cooling tower, where it is passed over a quarter-inch screen to clean it before spraying. The cooling tower reduces the temperature of the water about 30 degrees.

The condenser water flows by gravity from the hot well to a second cooling tower from which a two-stage Wheeler high-lift pump with a capacity of 1,000 gallons per minute belted to a 50-horsepower induction motor raises it 70 feet to a tank from which it flows by gravity to the condensers. This second cooling tower provides a reserve supply of 30,000 gallons which can be run to the Cameron sinker at the shaft tank and pumped to the distributing tank on the hill. The boiler blow-off is discharged to a small tank with a vent discharging to a launder to the main cooling tower. The waste water around the blast furnaces flows through a launder to a 10,000-gallon tank in which it is settled, the overflow going to the main cooling tower. This water contains some flue dust, and when the tank is cleaned, every 2 or 3 months, the sediment collected is returned to the furnace and smelted. These settlings amount to about 1½ tons per month and carry an average of 3 per cent. copper.

The furnaces use 900 gallons of water per minute; 300 gallons going to the roasters and 600 gallons to the blast furnaces. The Aldrich pump is sufficient to keep up this supply, and for boiler feed and all other purposes the 4 hours work per day by the Goulds pumps is sufficient. The total quantity of water averages 945 gallons per minute, and owing to the various devices used for conservation draws on the shaft supply only to the amount of 50 gallons per minute. It will be seen from this that the loss of water is 5 per cent.

The coke is from Sunnyside, Utah, about 175 miles distant. The Sunnyside coal cokes well and is used at all of the Salt Lake Valley smelters. The limestone is shipped from Parley's cañon across the valley, about 35 miles distant. It contains 51 per cent. lime CaO and about 5 per cent SiO_2.

The writer's acknowledgement is due the courtesy of Mr. C. A. Pringle, General Manager, Mr. Frank Murphy, Smelter Superintendent, and Mr. Thomas Maslin, Master Mechanic.

THE HISTORY OF THE ROCK DRILL

By W. L. Saunders

The rock drill is an American invention, conceived and developed in the United States. J. J. Couch, of Philadelphia, took out the first practical patents in 1849. In his experiments he was assisted by Joseph W. Fowle. The Couch drill was a crank-and-flywheel machine.

Couch and Fowle separated in 1848, the latter filing a caveat in 1849 covering a drill of his own invention and describing the successful power rock drill substantially as it is today. The most important feature of Fowle's drill is that the cutting tool was attached directly to the piston. In other words, the steel leading into the hole was an extenison of the drill piston rod.

Fowle described this invention in his testimony before the Massachusetts Legislative Committee in his contest with Burleigh in 1874, as follows:

"My first idea of ever driving a rock drill by direct action came about in this way: I was sitting in my office one day after my business had failed and happening to take up an old steam cylinder, I unconsciously put it to my mouth and blew the rod in and out, using it to drive in some tacks with which a few circulars were fastened to the walls."

Abroad, the nearest approach to rock-drill invention was the work of the German, Schumann, carried on in 1854. Fowle being without means to develop his ideas, they remained in obscurity until Charles Burleigh purchased his patents and produced the Burleigh drill, about 1866. This drill was used in driving the Hoosac Tunnel, in Massachusetts, in 1867.

Following Couch, Fowle, and Burleigh, came Haupt, Wood, Ingersoll, Sergeant, Waring, and Githens. Githens was the inventor of the Rand drill.

The Ingersoll drill was invented in 1871. Simon Ingersoll, a modest, ingenious, and honest mechanic, came to New York from Connecticut, bringing with him the models of several inventions. He was riding in a New York horse car one day and was describing one of his inventions to a fellow passenger. Another passenger in the car was John D. Miner, who overheard Ingersoll's conversation. Miner was a contractor, engaged with a gang of men on some rock excavation in New York City.

Miner broke into the conversation to ask Ingersoll why he didn't invent a rock drill, telling him that he had a gang of men at work striking a steel with a hammer to make a hole for blasting; that they could put in only about 10 feet of hole per day; and that he did not see why a machine could not be built that would do the work.

Ingersoll said he could make such a machine and would go at it at once if he had the money. Miner gave him $50, and his card, saying that, though he had never seen Ingersoll before, he had an honest face and he would trust him to spend that $50 in building a rock drill. "When you want any more," said Miner, "come to me and I'll give you another fifty."

Ingersoll's first rock drill was built in a shop at Second Avenue and Twenty-Second Street, New York City, owned by J. F. de Navarro, and was managed by Sergeant & Cullingworth.

One day Henry C. Sergeant saw the patterns for Ingersoll's drill. He noticed that the front head was attached to and was a part of the cylinder. He told the workmen that they should be in two pieces, and proceeded to saw off the pattern. At this point Ingersoll came into the shop. "What are you doing?" he asked. "I'm making this thing practical," said Sergeant, as he finished cutting off the pattern before Ingersoll could stop him. The result was the first row between Ingersoll and Sergeant, and it led later to Mr. Navarro purchasing, on Sergeant's advice, all rights and patents held by Ingersoll. The Ingersoll drill was made with the separate front head as used today.

Mr. Navarro organized the Ingersoll Rock Drill Co., investing $10,000 in the concern. Litigation arose with Burleigh, of Massachusetts, who owned the rights of Fowle and others. However, Mr. Navarro's plentiful supply of funds and his liberal nature, brought about a settlement of the suits, and all the patents became the property of the Ingersoll Rock Drill Co.

The business quickly paid back to Mr. Navarro the $10,000 he had put into it, and in later years he sold his interests to Mr. R. W. Chapin for $525,000. Sergeant sold out because of friction with the management, went West, engaged in mining, returned to New York about 1885, and organized the Sergeant Drill Co.

Early in rock-drill development the Rand brothers, Addison C. and Jasper R., had become interested through their connection with the Laflin & Rand Powder Co. Addison C. Rand formed the Rand & Waring Drill and Compressor Co., later controlled exclusively by Rand and merged with the Rand Drill Co., established in 1871, and incorporated in 1879.

J. C. Githens, superintendent of the Rand Drill Co., invented the "Little Giant" rock drill. He was the originator also of many improvements, notably the double-screw column with column arm, which made practical the application of the rock drill to mining and tunneling.

The Sergeant & Cullingworth Co., manufacturing the Ingersoll drill, the Sergeant Drill Co., and the Ingersoll Rock Drill Co., were merged into the Ingersoll-Sergeant Drill Co. Later on, the Rand Drill Co. and the Ingersoll-Sergeant Drill Co. were consolidated in the Ingersoll-Rand Co., today carrying on the business of all these pioneer concerns. The Rand drill from the beginning had been the most formidable competitor of the Ingersoll and Sergeant types. The conjunction of the Ingersoll-Sergeant and Rand companies, therefore, was a combination

of valuable patents in rock drills, compressors, and general machinery for mining, tunneling, and quarrying. Each shop received the benefit of the experience of all the others and the best features of the Ingersoll, Sergeant, and Rand types were taken to make an improved product.

POWER FOR CONCENTRATING MILL

*Compiled for Mines and Minerals, by F. C. Bowman**

No.		Horsepower Required	Horsepower of Motor
1	2 6-foot Chilian mills 1 Elevator, capacity 200 tons, 25-feet high 1 Sand pump	113.00	150.0
2	1 6-foot Chilian mill 1 Set rolls 16″ × 36″ 2 Elevators, capacity 200 tons each, 25 feet high 1 Sand pump 4 Wild shaking screens	107.20	150.0
3	1 6-foot Chilian mill 1 Elevator, capacity 200 tons, 25 feet high 1 Sand pump	53.10	150.0
4	1 Crusher 14″ × 25″ 1 Roll 16″ × 36″ 1 Elevator, capacity 200 tons, 45 feet high 1 Belt conveyer, 14-inch belt	30.00	75.0
5	18 Card tables 3 Sand pumps	19.00	50.0
6	15 Card cables	15.00	50.0
7	1 8″ × 10″ crusher 2 Sets rolls 14″ × 27″ 4 Revolving screens 5 Three-compartment jigs 1 Card table	32.16	50.0
8	8 Three-compartment jigs 1 Centrifugal pump, 3-inch discharge, head 30 feet, $3\frac{1}{2}$-inch Suction, No. 3 Byron & Jackson	24.80	30.0
9	3 Revolving Screens 1 Elevator, capacity 35 tons, 25 feet high	4.00	10.0
10	2 sets rolls 14″ × 27″ 1 Crusher 7″ × 10″	10.95	20.0
11	1 Samson crusher 7″ × 11″ 2 Sets rolls 14″ × 27″ 3 Revolving screens, 36″ × 72″ 1 Elevator, capacity 50 tons, 35 feet 1 Elevator, capacity 30 tons, 15 feet 4 Single three-compartment jigs 2 Card tables 1 King sizing screen 1 Belt conveyer, 18-inch belt, 60 feet long	33.36	50.0
12	1 Crusher 7″ × 11″ 2 Sets rolls 14″ × 27″ 1 Card table 7 Jigs, four-compartment, 18″ × 24″ 1 Elevator, capacity 50 tons, 40 feet 1 Elevator, capacity 20 tons, 20 feet 3 Revolving screens 36″ × 72″	32.97	50.0
13	8 Jigs, four compartment, 16″ × 30″	16.08	20.0
14	1 Crusher 7″ × 10″ 2 Sets rolls 14″ × 27″ 4 Revolving screens 36″ × 72″ 1 No. 3 Taylor centrifugal pump, 4-inch suction, 3-inch discharge, head 30 feet	18.76	35.0
15	1 5-foot Huntington mill 3 Sand pumps 1 Triplex pump 9″ × 10″, 60 revolutions per minute, against head 30 feet	29.88	60.0
16	7 Wilfley tables	4.02	7.5
17	1 Bucket conveyer	13.72	25.0
18	1 Crane washer 2 Conveyer belts	6.18	15.0
19	1 Crusher 7″ × 10″	6.53	25.0
20	1 Belt conveyer	2.07	5.0
21	1 Belt conveyer (small)	1.34	2.0
22	8 Jigs, four-compartment (This is not in same mill as No. 13)	16.08	30.0
23	2 Sets rolls 16″ × 37″ 1 Elevator, capacity 200 tons, 30 feet 4 Revolving screens 1 Triplex pump 5″ × 8″	40.46	50.0
24	1 $2\frac{1}{2}$-inch Morris centrifugal pump, 940 revolutions per minute, head 42 feet	13.80	10.0
25	1 Line shaft $2\frac{7}{16}$″ × 40′ long, 280 R. P. M., 4 bearings $7\frac{1}{2}$″ long 1 Wood pulley 4″ face × 8″ diameter 1 Wood pulley 6″ face × 36″ diameter 1 Wood pulley 8″ face × 11″ diameter 1 Wood pulley 8″ face × 14″ diameter 1 Wood pulley 9″ face × 15″ diameter 4 Card tables, $\frac{5}{8}$″ to $\frac{13}{16}$″ stroke, 255 to 265 R. P. M. 1 Deister slime table, $\frac{7}{16}$″ stroke, 285 R. P. M. 1 3″ × 6″ compartment Richards classifier, 175 R. P. M., 350 pulsations, average head 35 feet	1.50	15.0
	1 10″ × 54″ Frenier sand pump, 24 R. P. M., head 14.5 feet	1.50	
26	1 Main line shaft $3\frac{7}{8}$″ diameter × 11′ long, $3\frac{7}{16}$″ diameter × 27′ 6″ long, 200 R. P. M 6 bearings 12″ long 1 Iron pulley 16″ face × 60″ diameter 2 Wood pulleys 12″ face × 16″ diameter 1 Wood pulley 10″ face × 48″ diameter 1 Cam shaft { $4\frac{3}{16}$″ diameter × 13′ long; $4\frac{3}{16}$″ diameter × 18′ 6″ long } 45 R. P. M., 6 bearings 13″ long 2 Wood pulleys, 12″ face × 72″ diameter	8.20	50.0
	25 700-pound stamps, 6″-7″ drop, 90 drops per minute, woven wire screen 20 mesh, No. 24 wire, stamp duty $2\frac{1}{4}$ tons per 24 hours	25.20	
26	1 Crusher jack-shaft $3\frac{7}{16}$″ × 24′ long, 265 R. P. M., 3 bearings 8″ long 1 Wood pulley 13″ face × 36″ diameter 1 Wood pulley 7″ face × 30″ diameter 1 9″ × 15″ Blake crusher (Peter McFarlane & Sons); 250 to 265 R. P. M., crusher set to $2\frac{1}{2}$″, feed 2″ to 10″; duty 7 to 10 tons per hour	6.00 with crusher empty; 9.1 crushing at full capacity	
27	1 24″ Link-Belt apron conveyer, 37′ − 0″ centers, 8′ − 8″ rise, speed 17 and 40 feet per minute, capacity 10 to 20 tons per hour	1.90	7.5
28	1 24″ Link-Belt apron conveyer, 55′ − 7″ centers, 3 foot rise in first 17′ 2″, balance level, speed 40 feet per minute, capacity 20 tons per hour	1.60	7.5
29	8 Dunlap clay screens, speed driving pulleys 7 R. P. M., 1 line shaft and belting to screens	5.00	10.0
30	1 Dorr classifier, run full capacity	$\frac{1}{4}$	
31	1 Dorr continuous thickener	$\frac{1}{4}$	

*Consulting Mining Engineer, Denver, Colo.

METAL-MINE VENTILATION

Written for Mines and Minerals, by Evan W. Buskett

In the Joplin zinc district a large number of drill holes have been put down for prospecting purposes. The workings underground run into these holes, and at one place in the southwest portion of Joplin advantage is taken of the hole to supply fresh air to the mines.

In an open field a steam engine is seen running at full speed without any apparent connection with any industrial operation. On closer inspection it will be seen that the engine is belted to a blower and that the end of the blower is connected, as shown in Fig. 1, to a drill hole and this forces fresh air to the

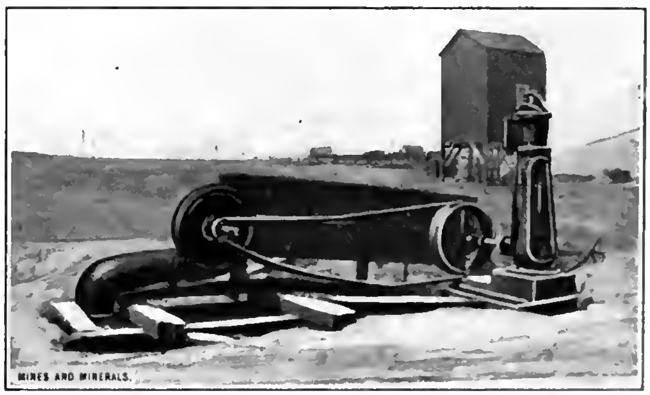


FIG. 1. BLOWING AIR DOWN A DRILL HOLE

miners several hundred feet below. The blower engine is about 8 horsepower and it obtains its steam from a boiler at the main shaft of the mine several hundred feet distant. The blower has a horizontal bottom discharge about 8 inches in diameter. This is connected with the casing of the drill hole by means of oiled-canvas air pipe.

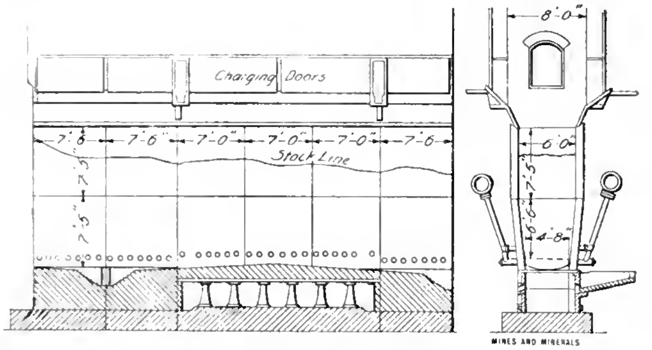
The drill hole taps one of the drifts in the mine below, and by means of this blower this drift is furnished with an ample supply of fresh air. The whole apparatus, while crude, is a step in the right direction, as better and more work will be done by men underground if furnished with pure air.

CIRCULAR COPPER BLAST FURNACES

Written for Mines and Minerals, by Thomas E. Lambert

There has been phenomenal progress made during the last few years in the treament of copper ores. In the process known as pyritic copper smelting, the constituents of sulphide ores provide almost sufficient fuel for their reduction, thereby reducing the cost of smelting and also eliminating costs of concentration, roasting, and reverberatory operations. In view of these facts, without in any manner criticising the present methods of treatment, but merely in the nature of suggestion, attention is called to the different designs of furnaces which generally are employed in copper blast furnace smelting, and to the product

Comparison of Forms of Furnaces Used for Smelting Iron and Copper Ores



(a) (b)

Fig. 1. Water-Jacketed Copper Blast Furnace, Anaconda, Mont., 1909

produced per square foot of hearth area relative to their fuel consumption, amount of air required, utilization of the air furnished, and to the by-products of their operation.

When a comparative investigation is made between the various furnaces employed in the production of copper and iron from ores, it will become evident that a similarity exists. In the early stages of the manufacture of pig iron, the operations were practically identical with those for the production of copper, but as the demand for iron increased, scientific men devised means whereby it could be more extensively and economically produced. Through their investigations and experiments the crude appliances and methods were abandoned, and today the iron and steel industry is a splendid example of what may be achieved by science.

Copper appears to have been the first metal used by man both in war and the peaceful arts, and it will eventually take the place of iron in many articles of manufacture for which it is a superior metal, and the demand will increase to enormous proportions.

Copper can be extracted from its ores at a comparatively low temperature, and in its production profits may be derived from the utilization of the by-products from the various operations.

At present the smelting of copper is in about the state of iron between the years 1850 and 1870 at which latter date improvements were made in blast furnaces, some of which were built of proportions that have not since been exceeded. Through constant study and experimentation, furnace practice has improved, until the output of a furnace has increased tenfold, and, by mechanical means the cost of operation has been reduced to a minimum.

The Washoe plant of the Anaconda Copper Mining Co., Anaconda, Mont., is the largest copper smelting plant in the world, and adopts the most modern scientific methods and appliances for the economical treatment of ores.

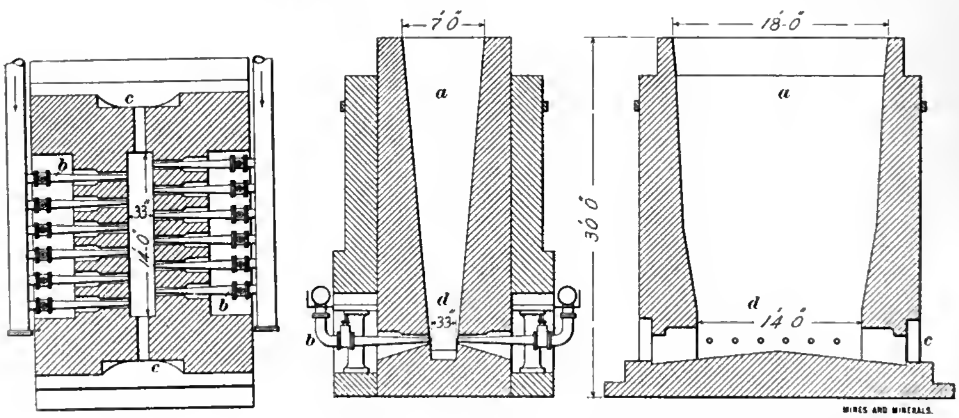
The blast-furnace building contains three furnaces, two 51 feet long by 56 inches at the tuyeres, and one 87 feet long, also 56 inches at the tuyeres. The two smaller furnaces have 88 tuyeres each, 14-inch centers by 4-inch diameter, and the larger furnace has 150 tuyeres of the same size and centers. The charges for the furnaces per 24 hours, are 1,600 tons of ore, flux, and coke for each of the smaller, and 3,000 tons for the larger.

For these furnaces, one power house contains six Connersville and four Root blowers, direct-connected to Corliss engines, each blower having a capacity of 25,000 cubic feet of free air per minute, equaling 250,000 cubic feet. Another power house contains four blowers, each with a capacity of 36,000 cubic feet per minute, equaling 144,000 cubic feet, making altogether 394,000 cubic feet of free air per minute against a pressure of 42 ounces per square inch.

Fig. 1 (*a*) shows a longitudinal, and Fig. 1 (*b*) a transverse section of a portion of the 87-foot furnace. The bottom of the crucible is formed of silica brick, laid on water-cooled plates, and has a gradual slope from the center to the discharge spouts. The 87-foot furnace has three discharge spouts and settlers, but is otherwise built similar to the smaller furnaces.

A daily report of the Anaconda furnaces gives the following information: 1,396.4 tons of material was charged in one of the 51-foot furnaces and 1,400.4 tons into the other. The hearth area being 238 square feet, the smelting capacities of the two furnaces are 5.867 and 5.88 tons per square foot of hearth area. The 87-foot furnace was charged with 2,502.9 tons of material, and as its hearth area is 406 square feet its smelting capacity is 6.165 tons per square foot. The total hearth area divided into the total charges give the capacity of these furnaces as 6 tons per square foot of hearth area. If 394,000 cubic feet of air is used per minute to reduce 5,300 tons of charged material, each ton will require 107,050 cubic feet.

The available capacity of the 87-foot furnace from the top of the bosh to the top of the jackets in the upper part of the shaft is (7 ft. 5 in. × 6 ft. × 87 ft.) = 3,871 cubic feet and below the bosh to the tuyeres (6.5 ft. × 5.33 ft. × 87 ft.) = 3,019 cubic feet or a total of 6,890 cubic feet. This is equivalent to carrying $\frac{6890}{406} = 17$ cubic feet of charge per square foot of hearth area,



Plan *Cross-Section* *Longitudinal Section*

Fig. 2. Rachette's Furnace

with a smelting column of 14 feet from the tuyeres to the top of the jackets. With a smelting column of 9 feet from the tuyeres, then the capacity will be 6 ft. 6 in. × 5 ft. 4 in. × 87 ft. = 3,019 cubic feet in the lower portion, and 2 ft. 6 in. × 6 ft. × 87 ft. = 1,305 cubic feet in the upper portion + 3,019 = 4,324 cubic feet total capacity, which divided by 406 = 10.64 cubic feet of charge to the square-foot hearth area.

A modern iron blast furnace with an available height of 81 feet from hearth to stock line, a diameter of hearth 14 feet and an area 154 square feet, has a capacity of 21,400 cubic feet, and receives over 2,000 tons of material per day; also 60,000 cubic feet of free air per minute (=43,200 cubic feet per ton), at from 15 to 30 pounds pressure. Such a furnace makes 500 tons of pig iron; carries 138 cubic feet of charge per square foot of hearth area, or nearly 13 tons per square foot of hearth area. The blast forced into the furnace having such a large amount of material to operate upon, will in traveling up the narrowing circular shaft of the furnace, thoroughly penetrate the charge, in the portions of the furnace where most required to effect the various reactions necessary to the reduction.

The Washoe furnaces are of abnormal length, and of modern design, yet it does not appear that length increases quantity of ore smelted, or is an advantage over the types in general use, and, should the three furnaces (two 51 feet and one 87 feet) be connected, making a furnace 231 feet long by 56 inches at the tuyeres, the results would be practically the same, though with an increased loss of heat and waste of blast.

To present one of the best illustrations of rectangular blast furnace, Rachette's has been selected, as with few exceptions all the modern copper blast furnaces are modifications of it, and it is the best form of furnace, next to the oval and circular types.

The Rachette furnace, Fig. 2, is employed in smelting iron ores in remote districts, and almost universally in copper-ore smelting in modified forms. It differs in shape from the modern iron blast furnace. The shaft *a* is an inverted four-sided pyramid, the throat being the widest part, 7 feet wide and 18 feet long, toward the hearth the width diminishes to 33 inches. The total height is 30 feet. At each end of the hearth is a slag hole and tapping hole *c*, *c*. The tuyeres *b*, six in number, on each side are staggered so that the nozzles will not be opposite. The advantages claimed for this shape of furnace, are, that its small height and comparatively slight construction, render it less costly to build than more massive furnaces; that the gases slacken in speed as they ascend, and must consequently be more efficacious in reducing the ore, than in furnaces, the shafts of which taper toward the top, so that the material smelted relatively to the cubic capacity is larger.

In respect to the design and claims in favor of low rectangular furnaces, it will be evident from Fig. 3 that, where the charge is distributed over so large an area, it will be impossible to maintain the charge in an even and porous condition throughout the length of the furnace. This is necessary in order to obtain an equal distribution of the blast and effect a perfect oxidation and reduction of the charge. At some place in the furnace extreme heat will be developed through concentration of the blast and there excessive reduction occurs, until finally the blast forces its way through the charge with the consequent overfire and violent agitation of the material. When this takes place the feeders put cold charge into the hollow space formed, thereby cooling and chilling that part, which then crusts over and diverts the blast to another part of the furnace where there is less resistance to its passage. With an overfire, the force of the blast and the smelting zone are being continually changed from one place in the furnace to another. This causes irregularity in the tonnage smelted and in the concentration of matte; and also is productive of large volumes of sulphur gases of a high temperature.

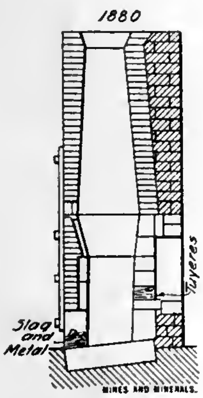


Fig. 4. Rectangular Furnace, Mansfield

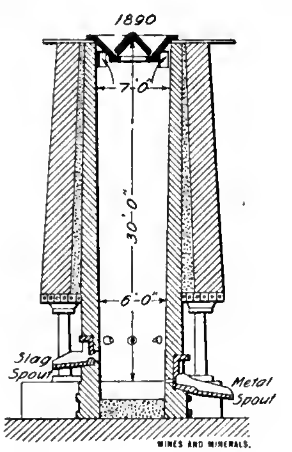


Fig. 5. Six Tuyere Furnace, Mansfield

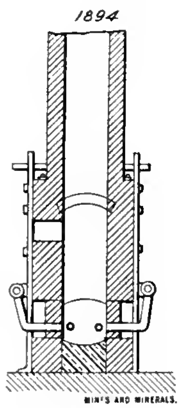
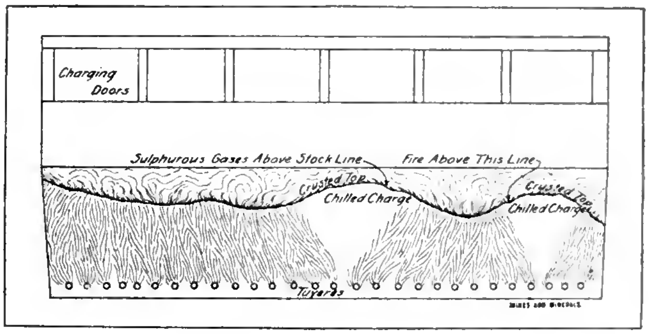


Fig. 6. Orford Brick Furnace

It may be inferred that, so soon as a circular or square shape of furnace is departed from, and with a low smelting column, there cannot be the same equality in the condition of the materials composing the charge, nor can there be such an even distribution and utilization of the blast to secure the necessary temperatures to effect the various reactions.



87 feet long, 56 inches at tuyeres

Fig. 3. Diagram Showing Production of Sulphurous Gases and Waste of Blast

When a long rectangular furnace is supplied with an unlimited quantity of blast, it will necessarily smelt more ore than one of half its length, still this may not prove the most economical practice, as much depends on the method of preparing the charge, charging the furnace, the amount of the blast, and the result, which is the amount of charge smelted per square foot of hearth area.

It will be evident from the following examples of different furnaces that better results are attained by the employment of shorter furnaces:

In a series of experiments made by the Tennessee Copper Co., 1903, to determine the effect produced by different volumes of blast, the following results were obtained with a blast furnace of the following dimensions: 180 in. × 72 in. at the top; 180 in. × 56 in. at the tuyeres; hearth area 70 square feet; number of 4-inch diameter tuyeres, 28; water-jackets 15 feet high; height of charge above tuyeres, 11 feet. Total amount of charge smelted in 24 hours is 581 tons, consisting of heap-roasted ore with quartz, converter slag, some custom matte and 56 tons of coke; the amount of blast used is 16,800 cubic feet per minute of free air. Tonnage 8.5 per square foot of hearth area. Free air per ton smelted 41,638 cubic feet.

The same furnace running on 507 tons of raw sulphides with quartz and 11.7 tons of coke, carried a charge 6.5 feet above the tuyeres, smelted 7.4 tons of material with a blast of 22,080 cubic feet of air per minute, or 61,262 cubic feet of air per ton.

The blast furnaces of the American Smelting and Refining Co., Garfield, Utah, 20 feet long by 4 feet at the tuyeres, smelt from 550 to 600 tons a day equaling 7.5 tons per square foot of hearth area.

At the Mt. Lyell copper works in Tasmania, a blast furnace

210 in. × 42 in. at the tuyeres with a column of ore 9.5 feet, made a phenomenal run by smelting 724 tons of mixed matte, silica, limestone, and slag, in 24 hours. There were forty 3-inch diameter tuyeres to this furnace, which with 40 ounces of blast

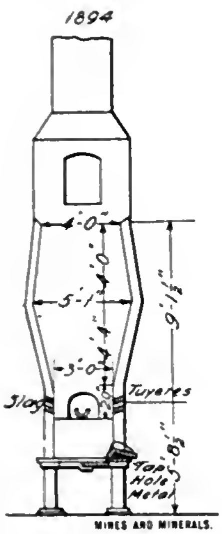


Fig. 7. Rectangular Copper Furnace

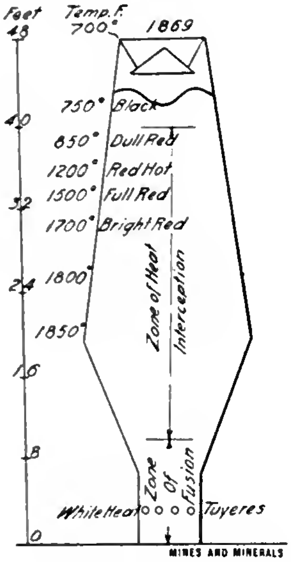


Fig. 8. Iron Blast Furnace

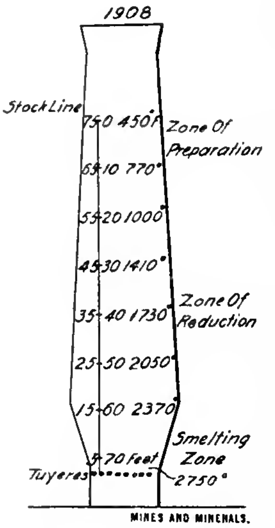


Fig. 9. Iron Blast Furnace

pressure smelted 11.83 tons of material to the square foot of hearth area.

The operations at the Mansfield copper works, Germany, are, according to some writers an excellent example of modern economical smelting practice. Fig. 4 shows a vertical section of the rectangular blast furnace formerly used at the Mansfield works, which was replaced by a circular furnace of much larger dimensions between the years 1885 and 1890. The old furnaces were from 15 to 20 feet in height.

Fig. 5 is a vertical section through the center of the circular furnaces, which as shown is 5 feet 4 inches in diameter at the tuyeres, 7 feet diameter at the top, with a height of 30 feet from the crucible to top of the furnace, and widening evenly from the crucible to the top. The blast is heated to 576° F. and enters the furnace by six water-cooled tuyeres at a pressure of 2 pounds per square inch. The top which is closed by a bell permits the collection of the escaping gases.

A furnace of this description, with the blast heated to 576° F. will smelt from 120 to 135 tons of burnt schist in 24 hours, with from 26 to 29 tons of coke. Total tonnage 164, hearth area 23 square feet, equal to 7 tons per square foot.

The escaping gases are collected, purified, and used in gas engines from which is developed a mean of 2,700 horsepower and at times as high as 4,500 horsepower.

Fig. 6 shows an altered form of Rachette's furnace, as employed by the Orford Copper Co. The interior dimensions are 11 ft 8 in. long × 3 ft. 5 in. wide, with 14 tuyeres—6 behind, 4 in front, and 2 at each end, with a diameter of from 5 to 6 inches. The height from the tuyeres to the charging doors is 8 feet.

Fig. 7 illustrates a rectangular, copper-lined, water-jacketed furnace of the United Verde Copper Co., Arizona. The cross-section is similar to iron blast furnaces except in height from the bosh to the throat, which is only 4 feet, and 9 feet 1½ inches from the tuyeres to top of jackets.

Figs. 8, 9, and 10 show different sections of iron blast furnaces, and are given because a consideration of the shape, dimensions, capacities, temperatures at different heights, color of the materials at different temperatures and heights, and the various temperatures in the zones of preparation, reduction, and fusion, will make the adaptation of similar furnaces for smelting copper ores appear more feasible.

To discuss all the matter presented by these illustrations, which practically cover the process of smelting iron ores, is beyond the scope of this article, nor is it necessary, because various solutions of these problems are given in several text-books on the metallurgy of iron; however, as the blast is one of the most important agents in successful smelting, in conjunction with the shape of the furnace a few remarks will be made as to the use of hot blast in preference to cold.

Whether hot or cold blast is used, it is produced from the same air at the same atmospheric temperature and in order that it may effect a perfect oxidation and reduction of the charge it must be of sufficient volume to contain the requisite amount of oxygen.

A modern iron blast furnace requires approximately 60,000 cubic feet of free air per minute at a pressure of from 15 to 30 pounds per square inch, with an available charge column of 80 feet height to smelt 2,000 tons of material per 24 hours. When the air is blown into the furnace direct at a temperature of 70° F., it absorbs heat from the furnace until it reaches the temperature required for fusion, 2,750° F. The air also expands in volume and raises the zone of fusion to a height governed by the time required to raise its temperature from 70° F., say, to the temperature of fusion. As the smelting zone is raised in the furnace, so also are the zones of reduction and preparation with increase in temperature toward the top and a consequent loss of heat in the hot escaping gases.

The same furnace supplied with the same volume and pressure of free air per minute at 70° F., but heated after leaving the blowing engines to a temperature of 1,000° F., will expand by the increase in temperature to a volume of approximately 230,000 cubic feet per minute, yet only contain the same

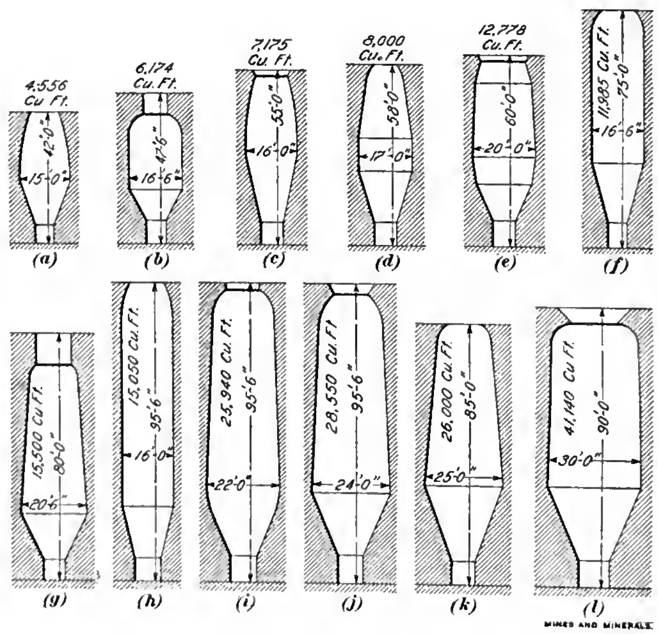


Fig. 10. Different Forms of Blast Furnaces

weight of oxygen. This hot air forced into the furnace absorbs the heat to bring it to the temperature of fusion at 2,750° F. (see Fig. 9) in the vicinity of the tuyeres, and in passing upwards the temperature is gradually reduced until it descends to 450° F. at the stock line, with a reduction in volume to about

90,000 cubic feet of escaping gases, and with a practically cold top.

It will appear from the shape of the furnace that, being so much wider at the bosh, this will provide a larger area for the expanded air to exert its force in sustaining the weight of the high smelting column, and, as it becomes diffused with the charge in passing up the narrowing shaft of the furnace its heat and other values are absorbed where required, to produce the reactions necessary to the perfect oxidation and reduction of the charge.

The use of hot blast has proved so economical in smelting iron ores in furnaces, with high smelting columns and a high pressure blast, it seems as if equally good results should be attained with smaller furnaces of the same shape with a proportionate reduction in volume and blast pressure.

The dimensions and shape of blast furnaces cannot be definitely stated as will be understood by examining the different shapes shown in Fig. 10; therefore, in order to obtain the best design, those that have given the best results should be taken as they represent the latest achievements of scientific and practical men after years of study and experiments.

Blast furnaces similar to those employed in smelting iron ores can be employed in copper smelting to advantage especially with sulphide ores containing gold, silver, lead, zinc, etc.

In regard to the use of hot blast in the modern copper furnace, there are differences of opinion. It has been experimented with and abandoned; however, this should not condemn its use with different furnaces. The trouble heretofore consisted in the use of rectangular furnaces with low smelting columns, and as shown, such furnaces are not suitable for effective smelting, nor as used do they furnish the best results from either hot or cold blast. With either kind of furnace the amount of free air required, when expanded by the temperature of 2,000° F. in the smelting zone, not having sufficient height of material to act upon, passes through the charge in its impoverished state, that is, with a smaller percentage of oxygen, with the resultant hot top, volatilization of material, loss of heat and waste of blast, whereas, should the furnaces be of the same shape and dimensions as those employed in smelting iron ores, it is manifest that similar heat reactions would take place as when smelting iron ores.

A DURABLE TRIANGULATION STATION

Written for Mines and Minerals

Mining engineers and deputy surveyors will doubtless appreciate the following details concerning a durable station which has come into extended use in the Globe, Ariz., district. Fig. 1 will serve to illustrate its construction. At the point selected for the station a block of concrete 15 inches square is set with three ¾″ × 12″ bolts, protruding from it as shown. The concrete is allowed to set for several days before attempting to attach to it the rest of the station, whose parts are of steel. These consist of a triangular base plate, 15 inches on each side, of ⅜-inch sheet metal bored in the center to receive the threaded end of a 1¼-inch pipe. It is likewise bored at the corners with

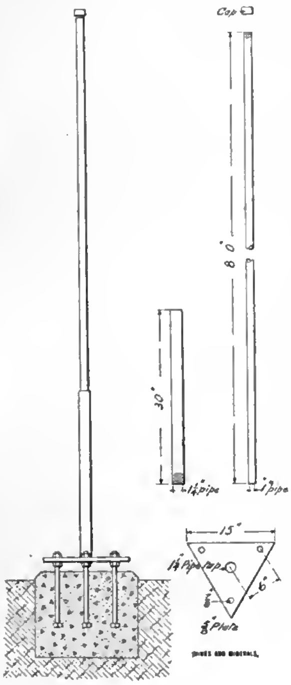


Fig. 1

⅞-inch holes to receive the holding bolts set in concrete. This base plate is first set on its bolts approximately level by eye but the locknuts not tightened. The 30-inch length of 1¼-inch pipe is then screwed in and the tight-fitting 8-foot length of 1-inch diameter pipe, capped, is set inside it. It then remains to accurately level up the base plate. To do this the transit is set up about 25 feet off at *A*, Fig. 2.

Sighting on the station rod, the base plate is leveled by adjusting the locknuts till O. K. in the direction *B C*, when the bolt nuts *1* and *2* are jammed down tight. The transit is

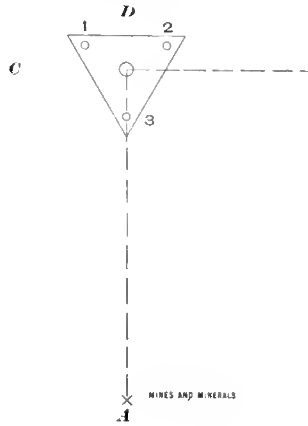


Fig. 2

then set up again at *B* and the locknut at *3* adjusted till the plate is level in direction *A D*, when the bolt nut *3* is jammed tight and the leveling is complete. The station is best painted with a black base, but with broad alternating black and white stripes on the flag, thus making it easily discernible for miles.

In case the station is located near enough to town to make it a probability that some one will fancy the striped flag for a walking stick or fish pole, it is easily kept in place by putting a steel pin through both pipes and locking with a padlock.

There are several varieties of this station in use in the Globe district, the form shown being that used by the Superior and Boston Co., and devised by Superintendent John D. Wanviz.

TARRIFF ON ALLOY OF IRON AND CERIUM

The protest of American Express Co. was overruled. The merchandise consisted of a metal alloy of iron and cerium, imported in slabs as a raw material. The merchandise in this case was a piece of steel 15 feet long, 4 feet 2 inches wide, 6½ inches thick, and weighing over 6 tons, with a geometrical design engraved on one side. It was a completed article ready for use in the manufacture of glass. The circuit court affirmed the decision of the Board of General Appraisers assessing it under paragraph 193 of the tariff act of 1897, which reads:

"Articles or wares not specially provided for in this act, composed wholly or in part of iron, steel, lead, copper, nickel, pewter, zinc, gold, silver, plantinum, aluminum, or other metal, and whether partly or wholly manufactured, 45 per centum ad valorem."

The importers, on the other hand, claimed that it should have been assessed under paragraph 135, the relevant parts of which read as follows:

"Steel ingots, cogged ingots, blooms, and slabs, by whatever process made; die blocks or blanks; billets and bars and tapered or beveled bars; mill shafting; pressed, sheared, or stamped shapes; saw plates, wholly or partially manufactured; hammer molds or swaged steel; gun-barrel molds not in bars; alloys used as substitutes for steel in the manufacture of tools; all descriptions and shapes of dry sand, loam, or iron-molded steel castings; sheets and plates and steel in all forms and shapes not specially provided for in this act * * *."

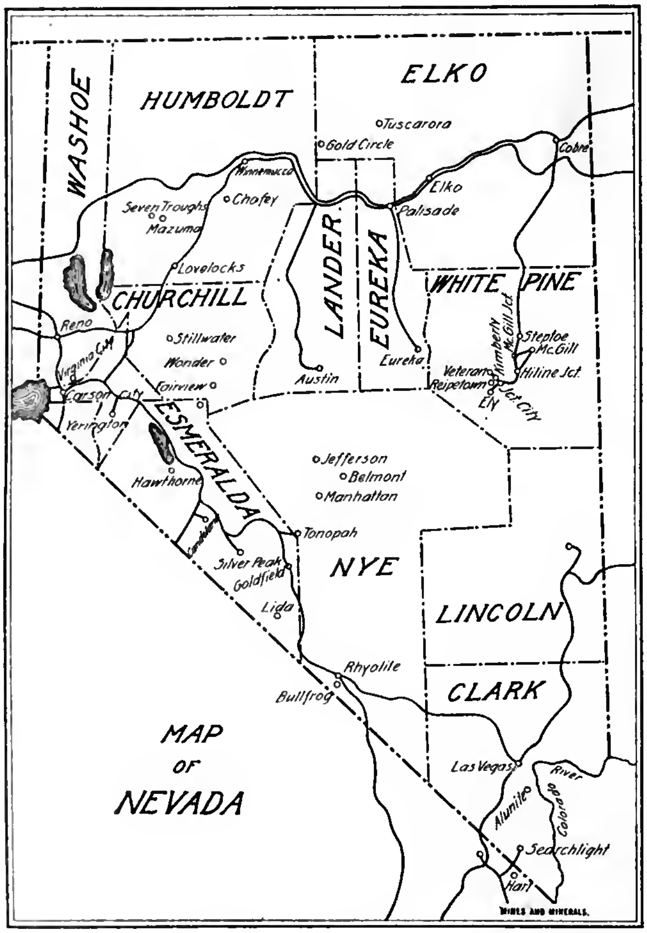
THE YERINGTON DISTRICT

*Written for Mines and Minerals, by C. S. Durand**

Reopening of Old Mines Made Possible by Improved Transportation and Cheaper Power

The Yerington mining district is in Lyon County, Nev., 20 miles southeast of the Comstock lode. As a gold camp it was abandoned when the Pine Grove Mine, credited with shipping \$6,000,000, shut down owing to an inrush of water which took place about the time that the Comstock was drowned out in 1882. That the mine was not closed from lack of ore, may be understood from the fact that it is now being opened by a tunnel. The conditions which caused the Pine Grove owners to stop operations were quite different from those of today. At that time labor, supplies, and freightage were costly, and when to this is added the cost of wood at \$55 per cord, it is not improbable that \$100 gold ore would be too low grade to pay. Comparing the past with the present, labor, supplies, and power are about one-tenth of the early days, a condition which makes \$20 ore a good mining venture. Yerington is in Mason Valley Mountains, a small range about 25 miles long, 5 miles wide, and of an extreme height of 6,500 feet. The district about 3 years ago received a new impetus through Boston capitalists acquiring copper prospects and developing them into mines, although copper has been removed in a desultory way for 30 years. The Nevada-Douglas Copper Co. is 20 miles from the nearest railroad station and hopes to have railroad connections this year. According to E. P. Jennings, the property carries 10,000,000 tons of ore averaging 3.5 per cent. copper and by expending \$1,500,000 for railroad, concentrating mill, and smelter, it should produce 24,000,000 pounds of copper yearly for 8.5 cents per pound at Atlantic seaports. The Mason Valley Copper Co., organized in December, 1906, has so promising a property and so large a tonnage of ore blocked out that it is building a smelter. This company uses electric power which it obtains from the Truckee River Power Co. It owns approximately 150 acres of copper properties. The Spragg Mine of this company, lying south of the Bluestone, was apparently opened on a contact between porphyry and limestone. This deposit was said to be 70 feet wide, with gossan top, and carbonate ore from the gossan to a depth of 100 feet, where it changed to copper sulphide.

About one year ago another gold property was discovered, which bids fair to help rehabilitate the district. This was the Effie May, now owned by the Yerington Mines and Exploration Co. In the search for copper this well-defined vein, 4 feet wide and now traced 2,000 feet on the surface, was overlooked. The ore from this vein averages about \$30 per ton, although one assay went as high as \$588. There has been no excitement over the find, as the Yerington company increased its holdings to 450 acres to include what is known as the "Mohawk group," located at the north end of the copper dike on which the other copper companies are located. Where the other companies have a carbonate outcrop, the Mohawk claims have a sulphide outcrop that contains gold. The basement complex rock of the Yerington district is given in "The Copper Handbook"† as intrusive granite, exposed by erosion in the higher peaks and deeper cañons, and covered by metamorphosed sedimentary strata on both slopes, with rhyolite flows along the eastern base. Copper ores occur in fractures and shear zones, and as bedded veins in dolomitic limestone, also as impregnations in limestone of the Carboniferous period. The ores found in the dolomitic limestone are mainly oxidized, with quartz and calcite gangue, while the Carboniferous limestone shows mainly chalcopyrite, with a little chrysocolla and covellite in the oxidized zone, but the secondary sulphides, chalcocite, and bornite are found rarely. Ore charges of two-thirds sulphides and one-third oxides and carbonates are self-fluxing, making concentration by smelting a rather easy proposition so soon as the railroad is completed this year.



The Mason Valley property which has gossan cap, with copper carbonates underneath and chalcopyrite below that carries 4 per cent. copper, merely follows the law of copper ore deposits formed by secondary concentration. The Nevada-Douglas Co. shows granite porphyry and wollastonite, carrying veins and contact deposits. The principal ore deposits are in contacts between porphyry and limestone with quartz veins carrying oxidized ores and copper pyrite mainly in wollastonite‡ rock which is also a contact deposit. At the surface, and to a depth of 10 feet, there are oxidized ores and chrysocolla, succeeded by a secondary zone carrying oxidized ores and secondary sulphides. The zone of oxidation and secondary enrichment apparently extends to a depth of 500 feet, giving an ore zone of 100 feet to 400 feet thickness in wollastonite with claimed superficial area of approximately 42 acres. The principal ore body is chalcopyrite associated with pyrite in granules, in wollastonite. The three principal veins of about 20 feet average width are estimated by the company officials to carry an average of 6.5 per cent. copper. The Mohawk group of the Yerington company covers a particularly interesting piece of ground geologically and mineralogically. Located at the north end of the copper dike it has a sulphide outcrop 100 feet wide, with eight veins carrying gold, some paralleling the copper deposit and some intersecting it at various angles. These veins carry little or no copper and are evidently from a different source and were formed at a different time. In addition to the copper dike and gold veins there are several vein-like deposits of fine turquoise which is suitable for jewelry.

Probably no other mining district in Nevada is so favorably situated for food and water as the Yerington, and this will

*[illegible], Peoria, Ill.

†Vol. IX, page 195.

‡Calcium silicate $CaSiO_3$.

materially add to its importance. Water is available from the Walker River and from artesian wells. The Singatse Mountain range, on which most of these mines are situated, overlooks the Walker River Valley, which is probably the best agricultural part of the state. Power for the mines is brought by wire from the Truckee River power plant, near Reno, Nev. With the advent of the railroad, the construction of concentrating mills and smelting furnaces, Yerington will become a noted copper camp, and with the development of its gold mines it will be no mean factor in restoring the pristine vigor of this part of Nevada made famous by the Comstock lode

PORTLAND CEMENT CALCULATIONS

Written for Mines and Minerals

Portland cement is a finely ground slag containing definite proportions of silica SiO_2, alumina Al_2O_3, iron oxide Fe_2O_3, lime CaO, magnesia MgO, potash K_2O, soda Na_2O, and small quantities of other ingredients as impurities.

Various proportions of limestone and clay are finely ground and fused to a clinker, but the mixture must be such that the lime formed shall have a definite ratio to the silica, alumina, and ferric oxide in the slag, or if magnesia be present in small proportions the same conditions must apply as for lime. The slag should have the following ratios for lime:

$$\frac{3CaO}{SiO_2}=\frac{3\times 40+3\times 16}{28+32}=2.8$$

$$\frac{2CaO}{Al_2O_3}=\frac{2\times 40+2\times 16}{2\times 27+3\times 16}=1.1$$

$$\frac{2CaO}{Fe_2O_3}=\frac{2\times 40+16}{2\times 56+3\times 16}=.7$$

The results show that 2.8 times as much lime as silica, 1.1 times as much lime as alumina, and .7 as much lime as ferric oxide must be used to form a proper cement slag. The figures used are the atomic weights of the various elements.

Magnesia should not be present in the mixture beyond 5 per cent., and what is present should be proportioned to form the following combinations:

$$\frac{3MgO}{SiO_2}=\frac{3\times 24+3\times 16}{28+32}=2$$

$$\frac{2MgO}{Al_2O_3}=\frac{2\times 24+2\times 16}{2\times 27+3\times 16}=.784$$

$$\frac{2MgO}{Fe_2O_3}=\frac{2\times 24+2\times 16}{2\times 56+3\times 16}=.5$$

The results show that there must be 2 parts of magnesia to 1 of silica; .784 part to 1 of alumina; and .5 part to 1 of ferric oxide. It is usual, however, to reckon magnesia in terms of lime, thus, $\frac{CaO}{MgO}=\frac{56}{40}=1.4$ parts of lime are equivalent to 1 part of magnesia.

The lime is obtained from the limestone used and contains some impurities, but these are ascertained by analysis. The clay is also analyzed so that the product to be derived from the fused mixture can be accurately foretold. Special rotary furnaces are used at present in preference to the old stack furnaces, and these are fired by gas or pulverized coal. The clayey material which is practically aluminum silicate, with small percentages of impurities is said to commence to fuse at 2,912° F. Pure limestone does not fuse but is converted into lime and then uniting with the materials in the clay forms the cement clinker. The lumps of clinker are separated from the unfused material and pulverized to form Portland cement. It has been ascertained that the finer the clinker is pulverized the better will be the cement, since, in extremely fine condition the cement forms a silicate with the sand used in practical construction work. The majority of limestones contain some clay and clays often contain limestone, and in making a mixture of raw materials previous to slagging these must be balanced. Magnesia may replace the lime up to 5 per cent. without bad results, and ferric oxide may replace alumina. A certain amount of iron acts as a flux, lowers the fusion point of the mixture, and promotes the combination of lime and silica. The replacement of practically all the alumina by iron has been advocated for cement that is to be used in sea water. Limestone containing approximately 75 per cent. calcium carbonate $CaCO_3$ and 20 per cent. of silica, alumina, and iron oxide is termed natural cement rock because it contains as impurities the quantity of clayey material that must be added to pure limestone in making cement. It is seldom that the right proportions are found as impurities in limestone to make a good natural cement, and to produce the right proportions clay or limestone must be added.

Other things being equal, raw material approaching natural cement rock can be a little more cheaply prepared than a mixture of pure limestone and shale.

Assuming a clay and a limestone to contain the following ingredients, it is desired to calculate a cement mixture:

	Clay	Limestone
Silica SiO_2	60.17	.38
Alumina Al_2O_3	22.65	.18
Ferric oxide Fe_2O_3	4.66	
Calcium oxide CaO	.31	55.62
Magnesia MgO	1.85	
Alkalies K_2O and Na_2O		
Loss on ignition	6.35	43.50
Water H_2O below 220° F.		.17

Clay.—

60.17×2.8 = 168.47 parts CaO for 100 parts SiO_2
22.65×1.1 = 24.92 parts CaO for 100 parts Al_2O_3
4.66×0.7 = 3.26 parts CaO for 100 parts Fe_2O_3
196.65 parts CaO for 100 parts clay
.31 + (1.85×1.4) = 2.90 parts CaO in 100 parts clay
193.75 parts CaO to every 100 parts clay

Limestone.—

$\left.\begin{array}{l}.38\times 2.8=1.064\\ .18\times 1.1=\ .198\end{array}\right\}$ 1.262 parts CaO in 100 parts limestone which is not available, hence 55.62 − 1.26 = 54.36 parts calcium oxide available in 100 parts of limestone.

Cement Mixture.—

$\frac{193.75}{54.36}=3.54$ parts limestone required for 1 part clay, or 354 pounds of limestone for every 100 pounds clay. The quantity of volatile matter in the limestone is 354×(43.5+.17) = 154.59; the volatile matter in the clay 6.35; hence the cement when burned will weigh

$$454-154.59=299.41\text{ lb.}$$

PARTS OF MATERIALS BY WEIGHT

	Clay	Limestone	Burned
Total	100.00	354.00	299.41
Silica	60.17	1.27	20.50
Alumina	22.65	.60	7.76
Ferric oxide	4.66		1.55
Lime	.31	196.89	65.90
Magnesia	1.85		.61
Volatile	6.35	153.99	

The percentage of lime should be from 60 to 63.5 per cent. in good Portland cement, while in the above mixture it is 65.9. The ratio for the silica-alumina must be within 3 : 1 to 4 : 1 or 2.5 : 1 to 3.5 : 1. A richer clay will have a lower burning temperature and carry less lime, which makes a poorer cement

GEOCHEMISTRY

*By George T. Holloway**

(Continued from July)

Probably the most promising field for ore bodies exists in the oldest plutonic rocks and particularly in such pegmatites and other extremely old granitic and other rocks as have been subjected at great depth and pressure and at high temperatures, to the action of intruded flows of fused mineral matter from still deeper-seated sources, or of vaporized mineral matter of similar origin. Such rocks exist in many parts of the world, but the pegmatites of Norway, the old granites of Greenland, and many of the old but less highly crystalline tin-bearing deposits of Cornwall may be instanced as likely to throw light on the origin of many metals, and especially of those at the "heavy" and "light" ends of the periodic table.

The Relation Between the Mineral and the Chemical Industries

It is extremely probable that some of the many missing heavy elements near the uranium end of the table will be found in such rocks and that certain light elements for which room may have to be made in the table, may also be unearthed.

My experience of the rare minerals found in the older rocks induces me to particularly refer to the vast field which they furnish for research for those who regard the rare metals from the commercial point of view, for those who wish to improve our present defective methods of both qualitative and quantitative analysis, and for those who are searching for new elements. I would specially draw attention to the "black tin" concentrates obtained from the tailings and slimes run to waste from the "dressing floors" of the Cornish tin mines. This waste passes down what is known as the "Red River" from Camborne to the sea at Gwithian, and is collected and retreated over a length of about 8 miles, by 16 independent firms or individuals known as "streamers," each of whom extracts as much as possible from what passes by his holding. Finally, on Gwithian beach, where the waste passes into the sea, the waves perform the last dressing operation, and, at low tide, little patches of the heavy tin ore which have thus been collected, are scraped up by the last of the tin dressers, to complete the tally. It is an interesting proof of the necessity for efficient dressing appliances for minerals, to note that the Red River produce amounts to over 800 tons annually of "black tin"—which by the way is red—containing an average of about 50 per cent. of metal, notwithstanding that the ore, before its waste is rejected by the mines, has passed through more "dressing" processes than any other mineral with which I am acquainted.

My excuse for this digression, is that the black tin from the Red River contains not only the tin, the only substance which is extracted from it, but there have become concentrated in it minerals of tungsten (representing according to my analyses, an annual loss of over 100 tons of wolfram), uranium, vanadium, tantalum, niobium, molybdenum, and other known heavy rare metals as well as strong indications of rare elements, which have somewhat similar properties, but which have not yet been isolated.

The occurrence together of practically all the heavy rare elements and the almost invariable occurrence with them or at any rate in the same or similar rocks, of many of the rare alkali, alkaline earth and allied elements such as lithium, caesium, rubidium, cerium, thorium, didymium, beryllium, zirconium, etc., is of extreme interest. All occur in pegmatites or others of the oldest and particularly the acid rocks, and practically none are found elsewhere except under conditions directly traceable to such an origin. Where uranium abounds, the other heavy metals are usually present in comparatively small quantities and the other groups of rare elements are practically non-existent while, under conditions where uranium is absent or only found in comparatively small quantity, the others may be found associated together either as components of minerals of complicated constitution or each in its own characteristic mineral form.

The occurrence of fluorides with these rare metals or in the rocks where the same occur, is also of importance. Both fluorspar and cryolite are commonly, although by no means always, associates of them, and it is considered by many that these fluorides, which often occur in fissure veins in granites, diorites, gneisses, etc., of great age, have been produced, in many cases, by the action of metallic fluorides whose decomposition has resulted in the production of metalliferous deposits and of such fluorides as fluorspar, cryolite, etc., and minerals containing fluorine such as topaz, tourmaline, etc.

The peculiar action which the heavy rare metals have when added to steel shows how curiously they stand as a class distinct from others. Uranium, tungsten, molybdenum, tantalum, niobium, vanadium, and even titanium, have extraordinary powers in producing special effects when added to steel, even in small quantity. These powers might be considered only as of passing interest from the point of view of the chemist, were they not practically confined to one group of metals and possessed more or less by each member of that group. It is true that valuable properties are imparted by other metals such as aluminum, manganese, nickel, and chromium, but their effects are less marked and are produced by the addition of a comparatively large quantity of the metal, excepting in the case of aluminum, whose action is, however, of an entirely different nature.

The recent demand for these rare metals has resulted in a large increase in the production of the minerals containing them. They were commonly passed over by the prospector or ignored by the analyst, and the present increased supply is mainly due to increased education, and is much on a par with the production of monazite and other thorium-containing minerals used for incandescent lamp mantles, and now obtainable from alluvial and other deposits in many parts of the world, in addition to its early and only source in Brazil. Over 200,000,000 of such gas mantles are now produced annually. The best concentrated monazite sand averages only about 5 per cent. of thoria and other earths actually required by the mantle industry, so that a large field is open for research to utilize the ceria, zirconia, didymia, etc., which are obtained as by-products.

Tantalite, at one time almost unobtainable, is now found in larger quantities than are required, and is even produced as a by-product in washing cryolite, with which it occurs, as might have been expected from a study of the natural associations of the rare elements mentioned above.

Reliable statistics as to the output of minerals of the rare metals are not obtainable, but it may be mentioned that the world's output of tungsten ores was over 5,500 tons in 1908, as against about half as much in 1905, although the price obtainable per ton had nearly doubled by 1907-8.

Indications of a simultaneous primary origin exist in the case of a large number of minerals and mineral aggregates. No lead mineral has, I believe, ever been found free from silver or from gold nor has any metallic lead or salt of lead free from silver or gold been prepared except by the most costly processes and in extremely small quantity. Practically all commercially treated copper minerals also contain gold and silver and modern methods of treatment now permit the buyer to pay for both, even when present in little more than traces which, 20 years ago, would have been totally lost. The attempts at obtaining the gold and silver from antimony ores in which both metals also invariably occur, have been largely responsible for the highly successful volatilization methods of extracting antimony which have resulted in so great a fall in the price of that metal.

The invariable association of selenium and tellurium with sulphur, of cadmium with zinc, and of nickel with cobalt, and

* An abstract of part of paper printed in the Journal of the Society of Chemical Industry, January 31, 1910, Vol. XXIX, pages 53-65.

the common association of potassium and sodium, of calcium, barium, and strontium and of the halogens, excluding fluorine, is too well known to need comment. It may be explained by the similarity of their properties, but there are many other mineral associations of elements which, like that of the precious metals as compared with lead, are not in any way similar, and a knowledge of them is of great assistance to the mining man and mineralogist in searching for minerals and in his anticipation of what may be expected to follow such indications as are found in prospecting, as well as to the metallurgist and chemist as a guide in searching for valuable or objectionable constituents in such minerals as he has to treat.

One of the most common associations is that of zinc blende with galena, copper and iron pyrites being also usually present. Such an association is objectionable in every way commercially, and although these minerals may be separated mechanically, hydraulically, or magnetically, when the minerals are not so fine grained or so intimately interlocked that a preliminary crushing will not break them apart, millions of tons of ore of high "assay value" exist which have not yet been treated at a profit and which are known as "refractory zinc-lead ores." When it is remembered that ores containing metal values of zinc, lead, silver, and gold to the amount of perhaps 10 pounds or more can be obtained for little more than the cost of mining, of almost uniform composition and in unlimited quantity, one cannot wonder at the numberless attempts which have been made to solve the problem and which have raised more hopes and emptied more pockets than any other metallurgical problem during the last century.

A knowledge of the common associations of minerals and especially of the metalliferous minerals, may often save the miner and the metallurgist from serious error and loss, and experience as to the condition in which they are required for the market is of equal importance. In the earlier days of mining, zinc blende, the most important ore of zinc, was regarded as a worthless mineral and was picked out from the galena with which it occurred and thrown away. Nickel ores are still commonly mistaken for copper ores and the writer, in one of the outports of Newfoundland, has seen molybdenite picked out from granitic rocks where it occurs under conditions which, to the mineralogist, would at once identify it, and used as black lead for blacking grates.

Even within the last few years, amblygonite, the hydrated phosphate of lithium and aluminum—now the most important source of lithia—was sold from a tin mine abroad as inferior phosphate for making fertilizers. The purchaser sold it to the lithia manufacturers at a large profit until the mine owners had their eyes opened by a friendly chemist.

Concerning metalliferous minerals generally, the Cornishman's adage, "Where they are, there they are," is a safe one for a non-geological paper.

As the most important group, and as typical of others, the sulphides are deserving of especial consideration by the chemist. In conjunction with the selenides, tellurides, arsenides, etc., with small quantities of which they are so commonly associated and which they so strongly resemble mineralogically, the sulphides directly or indirectly furnish a large proportion of the ores from which the world's metal supplies are derived.

It is true that pyrites as such, does not constitute a commercial ore of iron, but the oxide left by the roasting of pyrites is regularly smelted for the production of pig iron and it must not be forgotten that a considerable proportion of our iron ore deposits are themselves the result of the decomposition of sulphide of iron in one or other of its various mineralogical forms.

Where exposed to atmospheric action, or roughly speaking, above "water level" (that is, above the line below which pumping is necessary in the working of a mine) most of these sulphides oxidize more or less, often completely, so that metallic sulphates, carbonates, and oxides, commonly occur at and near the surface, the proportion decreasing as depth increases and finally giving place entirely to the sulphides from which they were derived.

The "gossans," or oxidized outcrops, which, in so many instances, have been the surface indications leading to the discovery of important mining fields, are of this class and have furnished more of the rare mineral specimens which grace our museums than all other sources combined.

Since iron pyrites is of such commercial importance, being the chief source of the world's supply of sulphuric acid upon which so many of the chemical industries depend, it may be well to discuss the changes which occur during the weathering of this mineral, and their important and far reaching results.

Briefly stated, the result of the weathering of iron pyrites is the production of ferrous sulphate and free sulphuric acid, followed by the oxidation of the ferrous—to normal, and ultimately, to basic, ferric sulphate. With certain kinds of iron pyrites—notably marcasite—the decomposition proceeds with great rapidity, a clean specimen soon becoming coated with minute crystals of ferrous sulphate even when enclosed in a mineral cabinet.

The sulphates so produced are mainly carried away in solution or suspension, but a small quantity usually remains and, by final conversion into oxide, reddens the mineral mass from which the pyrites has been leached out.

Should gold be present in the solutions acting upon the pyrites, the reducing action of the ferrous sulphate precipitates it as metal, producing oxidized ferruginous gold ore. The presence of extremely minute traces of gold in many saline and other natural waters is well known and, although this store of precious metal will most certainly never be profitably tapped by mankind, nature is continually drawing from it and depositing the gold in concentrated and therefore available forms. Such gold ores are often of great beauty and richness, and probably most of those in which the metal occurs in the crystalline state have been produced by the above or some similar reducing action.

Enormous quantities of pyritic ores containing the precious metal exist under conditions which altogether preclude such an explanation of the deposition, but the fact that the rich surface indications of a gold mine are by no means a sign of continuance in depth (as would be the case were the gold merely left behind by the weathering of the pyrites already containing it) is a significant proof of the frequent deposition of gold from surface and other waters.

In addition to their far reaching effects upon the igneous and other rocks already referred to, the soluble products from the weathering of pyrites also react upon such carbonates and other compounds of lime, magnesia, alumina, etc., as they come into contact with, producing the corresponding sulphates.

The carbonate of iron produced from such interaction may be directly deposited as chalybite or spathic iron ore, but is more commonly converted, by loss of carbonic acid and subsequent oxidation, into various forms of ferric hydrate or oxide, such as limonite, hematite, etc.

The familiar rusty iron stain which settles from chalybeate waters, affords an indication of the manner in which such ores have been produced, and the thick brown mud so frequently found in disused mines and on the beds of streams, are further examples.

The natural or induced weathering of iron pyrites is taken advantage of in the case of low-grade cupriferous iron pyrites at the Rio Tinto Mine, both metals being dissolved out as sulphate and the copper finally precipitated as metal by treatment with pig or scrap iron.

It is interesting to note that the Rio Tinto ore is one of the very few which lend themselves to such weathering. Many large deposits exist, which, although very similar in appearance, are almost incapable of such weathering and, being too poor for smelting, have had to be abandoned.

Although not a tenth of the total sulphur mined yearly in the form of pyrites, and practically none of that existing in other minerals, is utilized, the remainder passing into the atmosphere or being otherwise directly lost, nearly 2,000,000 tons of iron pyrites averaging about 45 per cent. of sulphur, is annually mined, as a source of sulphur, mainly for utilization in the manufacture of sulphuric acid. This huge supply is supplemented by a comparatively small return of sulphur recovered from alkali waste by the Claus-Chance process, and particularly by the large amounts of native sulphur mined in Italy and probably attributable to natural processes acting on sulphurous gases somewhat as in the Claus-Chance process, and by the rapidly increasing output of Louisiana. The two latter sources account between them for about three quarters of a million tons of crude native sulphur annually.

The consideration of the pyrites and sulphur industry must be left for the writers of special papers, but a few words may be devoted to the many by-products which are obtained during roasting, smelting, or other treatment of pyritic ores.

Selenium occurs, although only in minute quantities, in native sulphur and in all pyritic ores, and becomes concentrated throughout various stages of their treatment. Its production as a by-product of sulphuric acid manufacture is well known, but its latest and principal source is the anode mud or slime from the electrolytic purification of crude copper. Although its percentage in the cupriferous pyritic ores from which the bulk of the world's copper is obtained is so small that it cannot usually be determined, it becomes to a large extent concentrated with the copper through the various stages of treatment, and when one remembers that the bulk of the world's copper is purified by that process and that most of the selenium becomes accumulated in the mud left by solution of the anodes, the possibilities as to supply of selenium, should the demand increase, will be readily realized. Not only do these anode residues contain selenium; they contain often as much as 50 per cent. of their weight of gold and silver, together with much bismuth, all similarly concentrated by the selective power of metallic copper from the otherwise non-recoverable traces which occur in the original ore and all obtained as true by-products by a treatment which would be necessary in most cases even were they not present. As regards bismuth, it may be noted that, in addition to the amount recoverable from the copper, over 800 pounds is stated to escape daily from the smoke stack of a single copper smelter, the Washoe smelter, of Montana, and that bismuth is now obtained from the anode mud from the electrolytic refining of lead bullion.

In the roasting of tin ores and of many others where pyrites is present, large quantities of arsenious oxide are produced from the arsenical pyrites which almost invariably accompanies even ordinary iron pyrites and which is still more common with mixed ores. Although in mining districts unlimited amounts of sulphurous acid are often permitted to pass into the air, the escape of arsenic fumes is usually rigidly forbidden. A large proportion of the white arsenic of commerce and, through it, of practically all other arsenical compounds, is thus obtained as a by-product. The residues from the roasting of pyrites for sulphuric-acid manufacture are by no means exhausted of their values. The extraction of their copper contents and the recovery of their silver and gold by the Claudet process are well known, and the use of the burnt pyrites as pigments and polishing material and for smelting down for production of pig iron may be mentioned as the principal among its other applications.

In conclusion, the writer expresses his indebtedness to Mr. W. E. F. Powney and Mr. F. Rowley, whose help has been invaluable both in the preparation of the notes and as critics and correctors of the manuscript.

THE HECLA MINE HOIST

Written for Mines and Minerals, by J. C. McQuiston

An Electric Mine Hoist, Capacity 16,000 lb. From a Depth of 2,500 Feet at Speed of 2,400 Feet per Minute

The operation of large electric hoists, with mutual satisfaction to the mining company and the power company, has, during the past few years, received very careful consideration and today there is no question but that this problem has been satisfactorily solved.

With the introduction of the polyphase alternating-current generators and high-tension transmission lines, electric power has become available in most of the great mining districts. The simplest method of utilizing this power for hoist work is through the direct application of an alternating-current induction motor. If the capacity of the hoist is comparatively small, this is quite satisfactory, but where heavy loads must be handled at high speed, the peak due to acceleration of the moving parts is often far in excess of the average requirements of the hoist, in which case, if the current is purchased from a power company which also carries a lighting load, the voltage fluctuation each recurrence of the hoist cycle peak will prove most objectionable. If the hoist is located at the end of a long transmission line, an excessive amount of copper must be installed to prevent an undue drop in the voltage during the maximum demand. In any case, the power company must install all its equipment with reference to peak instead of the average load, and must charge accordingly. To meet these conditions the flywheel motor-generator type of hoist has been developed. The essential parts of such a hoist system are:

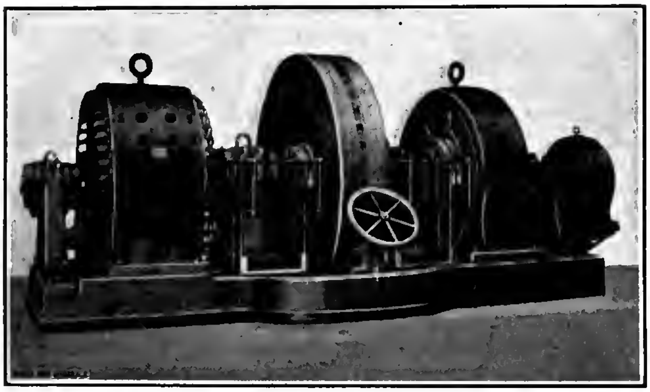


Fig. 1. Motor Generator

(*A*) A shunt-wound direct-current motor geared or coupled to the hoisting drums.

(*B*) A flywheel motor-generator set consisting of: (*a*) An alternating-current induction motor with variable external secondary resistance; (*b*) a direct-current shunt-wound generator with auxiliary commutating poles; (*c*) a heavy flywheel mounted on the motor-generator shaft; (*d*) a direct-current exciter for separately exciting both generator and hoist motor.

(*C*) A slip regulator which, by varying the secondary resistance of the alternating-current motor causes it to slow down under heavy loads.

(*D*) A suitable controller for varying and reversing the direct-current generator field current.

The armatures of the direct-current generator and hoist motor are connected in series, and the field of each machine is separately excited, constant full field being maintained on the motor. The hoist is started by applying a gradually increasing field to the generator, thus causing a proportionate voltage to be impressed upon the armature of the hoist motor. When the load on the alternating-current motor reaches a predetermined limit, resistance is automatically cut into its secondary circuit by means of the slip regulator. The speed of the motor

generator is thus reduced and a portion of the stored energy in the flywheel utilized in overcoming the peak of the hoisting cycle, with a minimum demand on the power system. In stopping the hoist at the end of a cycle the generator field is gradually weakened so that any excess energy stored in the descending cage, rotating hoist motor, and drums, is returned to the flywheel through the direct-current generator which momentarily acts as a motor, receiving electrical energy from the hoist motor, the functions of which are also momentarily reversed. To reverse the hoist, the field of the generator is reversed, causing the motor to rotate in the opposite direction. The heavy peak loads are thus eliminated from the line and the wear on the mechanical brakes is very much reduced. Since the current handled by the controller is very small, a great many steps can be provided, giving the operator a very elastic and sensitive control under all conditions.

Perhaps no electric-hoisting plant has attracted more attention from the mining men of the entire country than that of the Hecla Mining Co., at Burke, Idaho; the American Institute of Mining Engineers during their 1909 tour, made the trip to this mine for the express purpose of inspecting the installation.

The power available for this plant is three-phase, 60-cycle current, ranging from 2,080 to 2,300 volts. The motor generator, Fig. 1, is self-contained, having a cast-iron base, four bearings, and shaft. A 450-horsepower, three-phase, 60-cycle motor drives the direct-current generator, which is equipped with commutating poles to enable it to handle full-load current at any voltage between 0 and 600 volts. The flywheel, weighing 30,000 pounds, is mounted on the shaft between the motor and generator, while the direct-connected exciter is carried upon the shaft extension at the end of the set. The direct-current 550-horsepower, 600-volt hoist motor, operating at 60 revolutions per minute, Fig. 2, is direct-connected to the main reel shaft by a flange coupling.

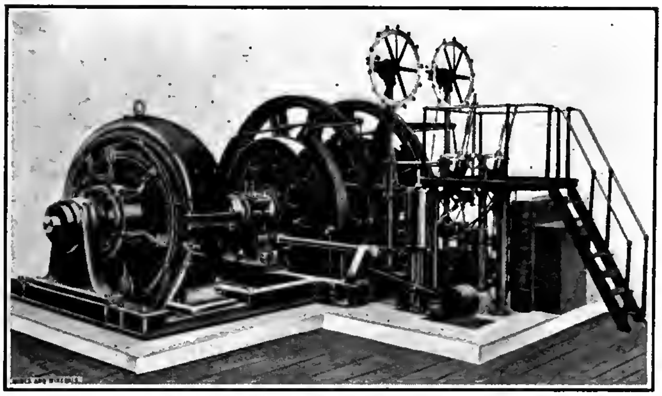


Fig. 2. Electric Hoist, at Hecla Mine

The control of the direct-current circuit for operating the hoist is obtained by means of a lever, the forward movement of which starts the hoist in one direction, and the backward movement in the opposite direction. This lever is direct-connected to a reversing field rheostat with a large number of points; since it is used to vary the generator voltage instead of cutting resistance into and out of the hoist-motor armature circuit, economical operation is secured at all points even when conditions necessitate running at lower than normal speed. All the operating levers have the same direction of throw with a movement of less than 30 inches. They are conveniently grouped upon a raised platform consisting of an iron frame and hardwood floor, with a suitable stairway leading from the engine-room floor to the platform, and a brass hand rail enclosing the stairway and platform. The entire electrical equipment described above was supplied by the Westinghouse Electric and Mfg. Co.

The hoist, Fig. 2, which was built by the Wellman-Seaver-Morgan Co., consists of two reels, each capable of holding 2,500 feet of $\frac{3}{8}'' \times 4''$ flat rope. Under normal conditions the hoist operates in balance, that is, the empty cage descends as the loaded cage is raised. If desired, however, either reel may be operated independently of the other. The winding diameter of the reel varies from a minimum of 5 feet when empty, to a maximum of 13 feet when the entire 2,500 feet of rope is wound up. Each of these reels is fitted with a Webster, Camp & Lane band friction clutch, post brake, and indicator.

The brakes, which are operated by means of combined air and oil cylinders and heavy counterweights, are so designed that they are set by the weights and released by means of air cylinders.

This hoist has a maximum hoisting speed of 2,400 feet per minute, and will handle an unbalanced load of 16,000 pounds, including rope, from a depth of 2,500 feet; yet its operation is the acme of simplicity.

The complete equipment, including the motor-generator set, hoist motor, slip regulator, switchboard, and hoist, weighs over 300,000 pounds.

STOPE TIMBERING

Written for Mines and Minerals

In many cases where a stope is not more than 8 feet wide stulls are set across with plank lagging for floor, and this is all the timbering required when the stulls are formed. In the case of crumbling walls, which require support, square sets and lagging are necessary. In stopes of a width only requiring one length of timber to reach across, but also requiring posts, the sill stulls are set first in an overhead stope, and the cap stull set first in an underhand. In all cases the stulls are cut of such length as to fit tightly against both walls, and a wedge is driven between the wall and the end of the stull to make it very tight. Each stull is set so that the end against the hanging wall makes an angle which is slightly greater than a right angle, in order that the least settling of the hanging wall may tighten the stull. The side posts are set after the stulls are in place in cases where the lower stull is set after the upper in underhand stopes.

In overhead stopes the posts can be set on the ends of the lower stulls and the upper stull driven down upon the top of the posts. When the side posts have to be set, after both top and bottom stulls, it is well to have the top of the posts framed for a gain in the end of the stull, then drive the bottom of the posts to place and secure it by drive bolts. The lower end of the posts and the upper side of the stull are not cut at all in this case, provided the timber is sawed.

Another way to put in square sets is to have them properly framed at the ends and of such a width as to approximately fit the width of the stope, and drive lagging between the square set and the wall to tighten the frame and support the walls. If the timber is to be removed and used again, this is much the better way, but in cases where there is no intention of removing the timber the framed method is to be preferred, as being both quicker and giving stronger support to the walls, although not making so neat a job. These methods have both been described, presupposing that each square set stands by itself, the sets being only connected loosely with one another by the lagging.

It is sometimes necessary in working in wide stopes to frame two or more wide sides so that they fit together to span the distance between the walls. In this case ties are necessary besides the caps, sills, and posts, the ties being similar timbers to the others, and framed so as to join the square sets together lengthwise of the stope and form a good joint. In case there is a squeeze on the square set timbers it is customary to use reinforcing sets which fit snugly with the square sets. If the pressure comes from the top, posts are employed to take it up, if from the sides, braces.

GASOLINE MINE LOCOMOTIVES

Written for Mines and Minerals

For a considerable time German and Austrian mining companies have been using gasoline locomotives for mine haulage, and there are now about 300 of these engines in various parts of the globe.

Application of Gasoline Locomotives for Mine Haulage. Successful Use in Germany

This shows that this system of haulage has long passed the experimental stage, and judging from the advantages such motors have over other kinds of locomotives, it seems reasonable to suppose that there will be a demand for them in this country. There is a large field for internal combustion locomotives in the West, where coal for fuel is scarce, or electric power is lacking; where water is lacking, or too bad for use in boilers; where great expense for fuel or the installation of electric power makes air, electric, or steam power prohibitive.

Compared with steam locomotives for mine haulage in general, these machines have the following advantages which may appeal to some, even though equipped with steam locomotives: They consume no fuel during stoppages; they require no water to replace that evaporated in the boiler; they possess no boiler to require repairs or inspection; they are of less weight; and when at work or idle do not fill an excavation with foul gases. They may be used also in mines where naked lights are prohibited and the purity of the ventilation is essential.

FIG. 1. GASOLINE MINE LOCOMOTIVE

The various advantages offered by gasoline locomotives of the explosive type suggest that they might be employed to distribute and gather cars, from mine rooms, where small wheel-base is required to round curves of small radius, and where mine tracks are not made sufficiently strong and solid for weightier motors. The makers of this locomotive claim that their "Century" model shown in Fig. 1 can, by special design, be constructed to replace mules for gathering purposes. This machine is made absolutely explosive-proof by the total elimination of any form of carbureter (or vessel for vaporizing fuel) outside the cylinder and by the ignition of the charge in the cylinder, which is effected by means of a low-tension magneto. For underground haulage in fiery mines, both terminals are insulated to comply with the Austrian Coal Mining Regulations, and the electric current conveyed to and from the sparking points in special metallic-woven cables so that all chance of fraying is obviated.

Compared with electric locomotives, internal combustion locomotives have the following advantages in general: The danger from accidents due to over head trolley wires of high voltage is eliminated. The danger of fire arising from defective insulation is also removed. The danger arising from explosion due to sparks setting fire to mine gases is obviated. They require no central or other power plant and are independent machines, which if out of order can be speedily repaired or be replaced. They represent considerable economy in first cost over an electric installation, in the comparative cost of the two different kinds of motors, and in track construction. Another advantage is found in the haulage system being capable of large expansion by the employment of more locomotives as the ouput demands, where a new electric installation and its accompanying expense is avoided.

Compared with compressed-air locomotives, they have several advantages, the chief of which when enumerated are: The first cost of installation; the decreased cost of operation; the increased flexibility of the system; and the smaller size of the motors.

Compared with rope-haulage systems they have all advantages which other kinds of motors have over the systems, and those mentioned, which others do not possess. To be sure they can not haul up as steep grades, nevertheless they are able to negotiate reasonable grades and uneven roads; however, they are not dependent on a central power station and admit of more ready extension as the mine expands. The cost of maintenance and of operation is less, the latter varying for underground haulage with inferior tracks and sharp curves from 2.4 cents to 2.6 cents per ton-mile. Actual working costs taken by owners over a sufficiently long time show figures for surface haulage as low as 1.2 cents per ton-mile, after deducting 20 per cent. for amortization.

This type of locomotive is regarded as perfectly safe by the Austrian, French, and German Government Mine Inspectors, for they permit the use of this engine in gassy mines in which firedamp is continually present. The locomotive machinery is housed over to prevent its being injured by falls of slate or rock from the roof; and is housed in on the sides to protect it from dust. The sides are readily removed as shown in Fig. 2, so that examination and repairs can be quickly made. The machinery in the rear end of the locomotive is easily reached by swinging doors, as shown in Fig. 3.

The gasoline engine employed in this locomotive is the most important part of the machine and upon this the manufacturers lay particular stress. The engine has a single cylinder, that makes a comparatively small number of strokes per minute, and operates in cycles common to all internal combustion engines, by taking in a quantity of air, next gas, then compressing the mixture and finally, by the explosion of the mixture, generating the power.

To control the speed of the engine, there is a regulator which is operated from the cab, by the engineer who runs the locomotive. The motor which is horizontal has the advantage over vertical motors of a low center of gravity, a point of considerable importance when one considers the narrow track gauges, sharp curves, and the limited head room in most mines. The fuel may be alcohol, gasoline, naphtha, or petroleum, although for underground work the volatile liquids, naphtha and gasoline, are recommended on account of their combustion being better than petroleum and less costly than alcohol. By means of a specially constructed inlet valve it is impossible for any explosive mixture to be formed anywhere except in the cylinder, although as an additional precaution the air intake pipe is provided with

gauze baffles as an additional precaution. The exhaust gases are sprayed with a small quantity of water as they enter the exhaust box from the cylinder, with the result that they are cooled, condensed, rendered harmless, and cause no inconvenience to the mine workers. It is said that mine air is vitiated by gasoline locomotives to a considerable less degree than by animals and miner's lamps, on account of the treatment given

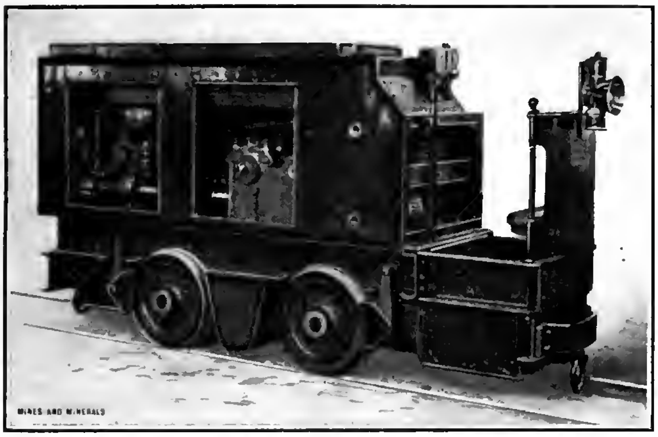


Fig. 2. Locomotive With Sides Removed

the exhaust, and further the spray is an absolute safeguard against any flame entering the exhaust box. The motor cylinder is cooled by a thermosyphon of such generous proportions that no circulating pumps or other mechanical devices are required. The water tank is of large capacity, but it should be refilled after each shift in order to replace the water lost by cooling the gases in the exhaust box.

The fuel tank has a capacity of 10 gallons, although the consumption is rather less than $\frac{1}{10}$ gallon per horsepower hour at full load. As, however, the locomotive is never run at full load except when starting and traveling up grade, it is found that $\frac{1}{20}$ gallon of liquid is rarely exceeded, so that a 12-horsepower gasoline locomotive consumes 6 gallons per shift of 10 hours. The fuel tank, which is perfectly air-tight, is arranged beside the water tank and is refilled at the end of each shift. The motor is mounted on a strong iron or steel frame, which rests on the boxes of four driving wheels having steel tires. Power is transmitted from the motor to all four wheels by means of a system of gearing, and in order to balance the power a heavy flywheel is fitted to the crank-shaft.

The gears which are always in mesh are engaged by means of clutches so arranged that their operation is quite "fool-proof." Powerful brakes are applied by means of a screw and lever fixed in the cab where the engineer can reach it readily. All the necessary appurtenances common to mine locomotives are attached to this, such for instance as headlights, warning bell, sandbox, etc., etc.

Small mining companies having an expensive haulage account can lessen it by the introduction of these locomotives, as they can be had in sizes that vary from 6 to 50 horsepower, and that will haul from 15 to 150 tons at speeds from 4 to 14 miles an hour. The engine for propelling the locomotive illustrated develops 16 horsepower, and a speed of 8 miles per hour.

The drawbar pull of such locomotives depends on their weight, condition of the tracks and road bed, and the grade; based on a drawbar pull of 1,500 pounds on the level the locomotive should have a hauling capacity of 50 tons on a good track, any grade of course will proportionally reduce the amount that can be hauled.

The drive is as stated on all four wheels and the locomotive moves forward or backward according to the motion given to a wheel in the driver's cab.

JOPLIN DISTRICT ZINC AND LEAD ORES

Written for Mines and Minerals, by L. L. Wittich

No proposed mining development of recent years has meant so much to the Joplin district as the announcement of the Granby Mining and Smelting Co. that it will mine its own properties. Heretofore the Granby company has done only a leasing business. Being the largest land owning company in the district, the decision to conduct mining operations means an era of development more extensive than any undertaken heretofore on the Granby land. As a starter the Granby company has drill rigs at work testing its holdings in the West Joplin camp and in the Oronogo camp, north of Webb City. At the former place open ground is found to contain good ore at shallow depths, while a deeper sheet formation exists. In the Oronogo camp efforts are being made to determine the extent of the deeper ore bodies which are just being discovered below the 400-foot level.

The Oronogo Circle Mining Co., operating on the Granby land, at Oronogo, has inaugurated a new system of keeping accurate accounts of all prospecting done in the mine. The machine drill men have received instructions to drill holes into the roofs, a distance of 8 to 10 feet apart, and also into the floor when stoping. The cuttings of all holes are saved and assays made. These cuttings are carefully bottled and labeled and placed away for future reference. A complete system of mapping enables the manager to tell at a glance how much ore has been left in reserve in roofs, floors, and stopes. The new system avoids the necessity of prospecting with winzes and rises.

At the 175-foot level of the Jack Harvard Mine on the Joplin tract of the Continental Zinc Co's. ground, a hoister has been

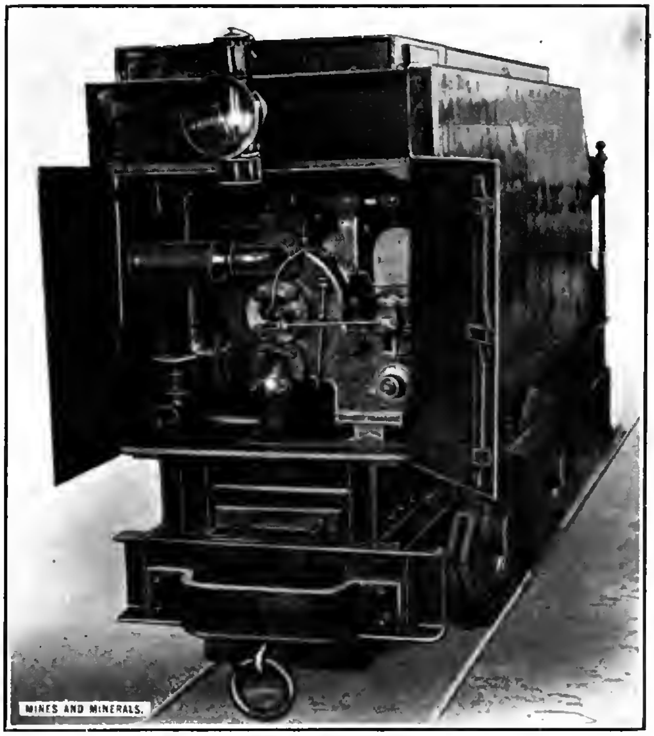


Fig. 3. End View of Gasoline Locomotive

installed and an inclined tramway leading into a winze has been put in operation. The winze extends at a gentle angle to the 190-foot level. A 15-foot stope in open ground is being taken up. Cars containing 1,000 pounds of ore are operated on the incline and are trammed to the shaft and hoisted. The company has drilled to 230 feet, encountering sheet ground at 210 feet.

On the Gregg land and on the Mattes Brothers' land, southwest of Joplin, in a region little mined since the early boom days

of the district, several drill rigs are at work and a number of new shafts are being sunk. Zinc blende has been found in a number of places near the surface while open ground at greater depths has been encountered. A number of concentrating plants have been erected recently and more are contemplated.

Allen Hardy and associates have taken a lease on the Weissbrod land, northwest of Joplin, and the 75-acre tract is to be thoroughly developed. A big daylight mine on this land is to be reopened.

The first turn in of ore from the Holy Moses Mine on a 40-acre lease of the Leckie land, northwest of Joplin, has been made. Hand jigs are used temporarily at the property which is operated by H. C. Moses.

The Tiwah Mining Co., operating on the McElroy land at Galena, Kans., has purchased a 250-ton concentrating plant from the Nowata Mining Co. The mill recently was purchased by the Nowata company and was to have been stationed on a tract west of Joplin. The plant was operated by the Kenwood Mining Co., at the Miami, Okla., camp. The Tiwah company has open ground ore in limestone and dolomite at a depth of from 250 to 300 feet.

The Nowata Mining Co. has purchased from J. C. Barr the Ave Maria concentrating plant, stationed south of Webb City. The company had planned to remove the mill to one of its Joplin leases, but an experimental try-out of the Ave Maria ground is being made. Former operations on this tract were unsuccessful but the present company is getting into exceptionally good zinc blende in ground heretofore overlooked.

The Peacock Valley Mining Co. has purchased an 18-acre tract from George Boughton, at Galena, Kans. Operations both with drill and with prospect shafts have shown exceptionally good zinc and lead ores, both near the surface and at depths below the 100-foot level.

The Victoria Mining Co., operated by George Lurwick in Cottonwood Hollow, northwest of Joplin, is getting into a rich pocket of zinc and lead ore at a depth of 60 feet.

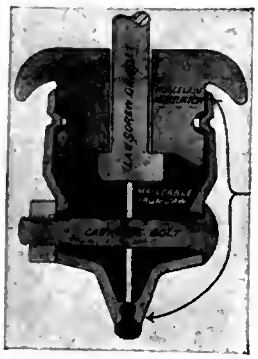
TROLLEY-WIRE SUPPORT FOR MINES

The conditions existing in mines demand trolley-wire supports of different design than those commonly used on the surface, and several hangers of excellent design have been in use. Recently, however, a new hanger has been devised which embodies some radical departures from other types, and contains features of a very desirable nature This hanger, known as the "Vise Grip" has been but recently placed on the market, and has met with success

The following description of the device, which is being manufactured by the Trolley Wire Support Co., of Syracuse, N. Y., is furnished through the courtesy of Mr. John L. Wagner, of the Pneumelectric Machine Co., of Syracuse, N. Y.

One of the features of this hanger, which departs most radically from prior practice, is the material used for insulation. Heretofore the insulation has been a composition, molded, under pressure and heat, into an outer casing of iron or other metal. In this composition was embodied a stud bolt to which the hanger was to be attached. This manner of constructing insulators has been exclusively used, and while it is efficient, if the composition is of uniform and proper quality, there has always been much difficulty in maintaining uniformity of composition and absence of imperfection in molding; so that absolute protection from electrical leakage and assurance of proper mechanical strength could not be obtained.

The insulator used with the Vise-Grip hanger is porcelain, a material noted for its high insulating qualities and great mechanical strength under tension or compression. In all the high-tension transmission work of 30,000 to 40,000, and even in some cases 100,000 volts, porcelain is used and is giving more satisfactory results than any known insulator commercially available. In addition to the use of porcelain in the "Vise Grip," there is no imbedding in an outer metallic cap, but a plain, specially formed, highly glazed insulator is provided, to which the jaws for supporting the trolley wire are attached without the use of an intermediate stud bolt or other device imbedded in the insulation.



Malleable iron jaw grips at these two points only. A "Vise-Like" grip to insulator and trolley wire.

Vise-Grip Trolley-Wire Hanger

The insulator being cylindrical in form, provided with a drip hood made integral with the body, permits the attaching of the jaws for supporting the wire in a very simple and effective way. The jaws are made semi-cylindrical in section at one end and straight at the other. The cylindrical part of the jaw is made to conform to the insulator and a bead is provided which fits in a corresponding groove in the insulator. The straight end of the jaw is formed to correspond to the shape of trolley wire to be suspended, and in the middle of the casting a hole is provided suitable for an ordinary carriage bolt. Two of these jaws, both alike, with a porcelain insulator and a carriage bolt, constitute the complete hanger. Its simplicity can accordingly be appreciated.

A hole through the insulator provides for suspending it by means of lagscrew, bolt, or other device. When this is in place the jaws are put in position, the carriage bolt passed through the holes provided for that purpose, and the nut screwed on with the fingers; the jaws will then be suspended from the insulator by means of the beads on the jaws resting in the groove of the insulator. The jaws are loose on the insulator, until finally clamped tight, and can be revolved at will to line up with the trolley wire when it is ready to be strung. In attaching the jaws to the wire it is only necessary to place it in the jaws and further tighten the carriage-bolt nut with the fingers. This will bring the jaws sufficiently together to permit the wire to hang in them loosely. After the wire has been thus suspended throughout the distance to be covered, the slack can be taken up in the usual way with assurance that the hanger will not be damaged, and the wire will remain suspended thereby until the carriage-bolt nuts are permanently tightened. To do this an ordinary monkeywrench, open end, or socket, wrench, is required. The nuts can be tightened to any degree desired and the jaws powerfully and simultaneously clamped to the insulator and wire. This manner of simultaneously clamping the insulator and wire discloses the underlying principle of the Vise-Grip hanger, for it is nothing more or less than a vise-like action, or the applying of great pressure by means of a screw.

POWER PRODUCTION AT COLLIERIES

At a meeting of the Mining Institute of Scotland, a paper on "Power Production at Collieries, with Special Reference to Gas Power and Electrical Centralization" was read by Mr. Robert Crawford and Mr. Harold Moores.

The authors said that the growing need for economy made the subject of power cost one to which the colliery manager must devote considerable attention, with a view to generating and transmitting his power in such a manner as would insure the highest possible efficiency. The report of the Royal Commission on Coal Supplies pointed out that collieries were extremely wasteful in the amount of fuel consumed in driving their own plants, the annual consumption in British mines reaching the enormous total of 18,000,000 tons. This was equivalent to about 7 per cent. of the total output, whilst it was an ascertained fact that in some individual cases the consumption in the colliery boilers reached the high proportion of 11 per cent. This proved the urgent need for economy, and an effort would be made to demonstrate that very high economies could be obtained by various methods and in varying degrees according to individual circumstances. Of late years the adoption of electricity as a secondary power for pumping, ventilation, haulage, coal cutting, etc., had greatly increased the efficiency of mining power plant by displacing small steam units. Much good work had also been done by the introduction of low- and mixed-pressure turbines so arranged as to utilize the large amount of exhaust steam from the winding engines, and to a less extent from haulage and other engines about the colliery. But while a good deal of attention had been devoted to electrical centralization of power at large collieries by means of exhaust-steam turbines, the authors said it was found that much higher economies could be obtained by converting the fuel into gas for the purpose of driving gas-engine electrical-generating sets, from which energy would be transmitted at high tension to the machinery at outlying pits; and accordingly they described the main types of producer gas plant available, showing the special functions and advantages of each in relation to colliery practice.

In conclusion, they observed that the heat possibilities of gas-engine exhaust did not yet appear to have been fully appreciated. Special exhaust-heat boilers were available, and many of them were in use in connection with gas-engine installations in Great Britain, with such an efficiency that 2 pounds to 2½ pounds of steam per boiler horsepower-hour might be obtained at 130 pounds pressure, without placing any appreciable back pressure upon the engine. Thus, an installation working at an average load of 3,000 kilowatts would evaporate about 9,000 pounds to 11,000 pounds of water an hour, which would easily drive a steam engine continuously at a load of 625 horsepower, allowing 16 pounds of steam per indicated horsepower-hour. This was waste heat converted into useful work, as in the case of the exhaust-steam turbine. It might be wondered why British colliery owners and power users in general had been so slow to take up the large gas engine, but no satisfactory answer was yet forthcoming. Great strides had been made in this matter in Germany, Belgium, and the United States, and large numbers of instances might be quoted in which units of 1,000 horsepower and upwards had been installed, with most satisfactory results. In one case, that of the Indiana Steel Co. of Gary (Indiana), there was an installation of 63,750 horsepower, consisting of 17 engines of 3,751 boiler horsepower each, whilst installations of 30,000 horsepower and 40,000 horsepower might also be mentioned. If did now appear, however, that the day of the large gas engine in Great Britain had at length arrived. Several large sets were at work in the Clyde, Cleveland, and South Wales districts, together with an installation of 20,000 horsepower at Messrs. Brunner, Mond & Co.'s works, Northwich, and one of 10,000 horsepower at the works of the Castner-Kellner Alkali Co., Ltd., Runcorn. It was also worthy of note that one prominent firm of gas-engine makers was building and equipping a special works solely for the manufacture of large vertical gas engines in units up to 3,000 horsepower. Actual experience had undoubtedly proved this system of gas-power production to be convenient and economical, and there were substantial reasons for believing that, as the possibilities of the system became more fully understood, its applications would be increasingly extended. A gas-driven central electrical power station for a group of collieries was the ideal arrangement, where circumstances admitted of the outlay which such a scheme involved; and there appeared to be no doubt that many such would spring into existence during the next few years, with a view to using to the best advantage the heat possibilities of the coal consumed.

A TENT AND SHACK LIGHTING OUTFIT

The John Simmons Co., of New York, have recently adapted their acetylene mine lamp to use as a lamp for tents or shacks in mining camps. A larger generating outfit has been made for this purpose that gives a twenty-five candlepower light lasting eight or nine hours without recharging.

The Baldwin Camp Lighting Outfit

The generating outfit hangs to the tent pole or wall of the shack in a convenient place, and a rubber tube leads to a fixture similar to an ordinary gas fixture.

This light is sold at the price of $5 and, when desired, a portable table lamp is furnished for $1 extra. The outfit is something which, if once used by prospectors or miners in their tents or shacks, will be recognized as one of the greatest comforts and luxuries possible in such a temporary home.

TESTS FOR CARBON MONOXIDE

At the fifty-second general meeting of the Institution of Mining Engineers, in London, June 2, 1910, Dr. J. S. Haldane gave a demonstration of the use of small animals in testing for carbon monoxide, and of a new arrangement for enabling a man equipped with a rescue apparatus to test the air at any time. The experiments were as follows:

1. The fact that small animals may be safely used to indicate the presence of carbon monoxide was shown in a large iron air-tight chamber. 2. A new arrangement was demonstrated for enabling a man with a rescue apparatus to test at any point whether the air is respirable. 3. The improved form of the Hall-Rees apparatus for enabling men to escape from a sunken submarine was demonstrated in an experimental tank.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

Western Editorial Office: 728 Equitable Building, Denver, Colo.
Chicago Office: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to Mines and Minerals, Scranton, Pa.
CABLE ADDRESS, "RBTSOP, SCRANTON"

RUFUS J. FOSTER.......................................Manager
EUGENE B. WILSON, Scranton, Pa.......................Editor
GEORGE F. DUCK, E. M., Denver, Colo.................Western Editor
P. G. MOORE...Circulation Manager
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., Ltd., Dryden House, 43 Gerrard St., Soho, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 Pitt Street, SIDNEY.
SOUTH AFRICAN AGENTS: J. AFFLECK ROBERTSON, CAPE TOWN, Mansion House; JOHANNESBURG, Trust Building; DURBAN, Natal Bank Chambers; BLOEMFONTEIN, Florence Buildings.

SUBSCRIPTION RATES

Single Copies.......................................25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada.......................................2.50
To all other Countries.......................................3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always give your former as well as your present address.

TABLE OF CONTENTS

Miami Concentrating Mill, by R. L. Herrick....................... 1
Geology of the Coalinga Oil Field....................... 4
Brazil Iron-Ore Deposits....................... 7
Fast Tunnel Driving....................... 9
Zinc and Lead in Arkansas, by L. L. Wittich....................... 10
Ore Mining Notes....................... 11
Testing for Protective Alkalinity, by Bede Collingridge....................... 13
The Yampa Smelter at Bingham, by Leroy A. Palmer....................... 14
History of the Rock Drill, by W. L. Saunders....................... 18
Power for Concentrating Mill, by F. C. Bowman....................... 19
Metal-Mine Ventilation, by Evan W. Buskett....................... 19
Circular Copper Blast Furnaces, by Thomas E. Lambert....................... 20
A Durable Triangulation Station....................... 23
The Yerington District, by C. S. Durand....................... 24
Portland Cement Calculations....................... 25
Geochemistry, by Geo. T Holloway....................... 26
The Hecla Mine Hoist, by J. C. McQuiston....................... 28
Stope Timbering....................... 29
Gasoline Mine Locomotives....................... 30
Joplin District Zinc and Lead Ores, by L. L. Wittich....................... 31
Trolley-Wire Support for Mines....................... 32
Power Production at Collieries....................... 33
Editorial—
To Friends—The Coxe and Fritz Laboratories—Teaching Foreigners English—Thesis Prize....................... 34
Book Review....................... 35
Correspondence—
Surveying—Thermochemistry of Anthracite—Determination of the Meridian—Lamp Burns in Water—Pumping....................... 37
Meetings of Institutes....................... 38
Haulage Rope Strains....................... 39
A New Transportation System....................... 39
The Mulga Mine Explosion....................... 40
Mechanical Stokers and Hand Firing....................... 42
Catalogs Received....................... 43
Charging Blasting Holes....................... 44
Evolution of Mine Haulage, by E. B. W....................... 45
Personals....................... 50
Petroleum Deposits About Tampico, Mex....................... 50
Answers to Examination Questions—
Mine Inspectors, Scranton, Pa., May, 1910....................... 51
The Illinois Coal Field, by H. H. Stock....................... 54
Engineering of Modern Coal Plants, by Howard N. Eavenson....................... 57
Teaching English to Foreigners, by Peter Roberts....................... 60
Colliery Survey Notes, by Ralph Shumway....................... 61
The Acid Specific Gravity Test, by A. Langerfeld....................... 62
New Inventions....................... 64
Squibs—
Simultaneous Inventions, 3—Radium in Austria, 6—Tariff on Alloy of Iron and Cerium, 23—Tent and Snack Lighting Outfit, 33—Tests for Carbon Monoxide, 33—Knox Blasting System, 36—Use Care in the Purchase of Brattice Cloth, 44—Losses in Electric Transmission, 53—Weight of Fulminate of Mercury, 59—Coal Mining in India....................... 64

TO FRIENDS

RESIGNATION from Mines and Minerals in the month of June recalls some of the emotions of a June graduation not so long ago. Now, as then, the predominating one is that of regret at parting from steadfast friends. Relations with both the executive force of the journal and with hundreds of friendly correspondents have always been amicable and it will ever be a sincere pleasure to recall that they continued so to the end.

Did leisure time allow, the writer would gladly write a personal good-by to all the hundreds of friends whose contributions of articles and data have in such a large degree been responsible for the success of the western section of the journal. Truly it may be said that, whatever measure of success has been ours, that success is founded upon friendship. In lieu of a personal letter, manifestly impossible to send all those who have contributed to our columns at the writer's request, this editorial is dedicated—to friends in token of a sincere and lasting appreciation.

To friends—scattered from Alaska to Mexico, inclusive, and from coast to coast—yours be the credit. You knew the writer as a miner, a transitman, an assayer, a manager, and finally a "pen pusher." And if in his writings he reflected the enterprise, the foresight, the courage, and the magnificent achievements of the western engineers, it was because in all his vocations he absorbed the spirit that prompted you as well as the information you freely gave him. And lest a mistaken idea be gained that the writer proudly claims as friends only the foremost of the mining and metallurgical professions—a word further. Had the writer depended upon them alone his results had been far less than they actually were. It is to many a graduate student—a mere beginner in the arts—that the writer owes his fullest comprehension of far reaching engineering plans. It is to many a miner, readier with his good-natured tongue than with his pen, that he is indebted for his keenest insight into an accomplished work.

For all the thousands of miles traveled to visit the principal mines of the country, friends have never failed to make possible a rich compensation for strenuous effort. That compensation may be partially viewed in resulting articles scattered over 50 issues of Mines and Minerals, but it is the invisible recompense that goes with the writer into retirement—the memory of your glad-handed welcome and generous entertainment.

As in the past, so in the future, Mines and Minerals must depend upon its friends for its continued business expansion—the success that comes to it must be founded upon the success of its friends. So in extending his best wishes for the future prosperity of his friends in the field, he bespeaks the continued cooperation of them with his successor and with Mines and Minerals, and may the fruits of that friendship continue to enrich and ennoble the mining and metallurgical professions.

R. L. Herrick.

THE COXE AND FRITZ LABORATORIES

LEHIGH UNIVERSITY, a comparatively young educational institution, has made great strides in its progress toward the foremost of technical colleges in the United States. Founded by Judge Asa Packer in 1865, and guided by such men as Eckley B. Coxe, the father of coal mining, and John Fritz, the father of the steel industry, its success was assured in time, but not so rapidly as it has occurred. On June 11 the Coxe Mining Laboratory, so named by the trustees in memory of a trustee and friend, whose services to Lehigh are held in grateful remembrance, was opened.

On the same date the John Fritz Engineering Laboratory was opened. In the course of his remarks Mr. Fritz said: "How these gentlemen have turned out engineers with the things they have is more than I can comprehend."

Having been in touch with both the professors and the alumni of Lehigh University for the past 30 years, we are convinced that Lehigh's success is not due to superiority in courses of study; nor to the superiority of its professors; nor to the erudition of its graduates; but it is due to the united efforts of its alumni in pulling together for each other and their college.

From inquiries made we are able to state that Lehigh professors and assistant professors teach, and do not merely hear students recite, as unfortunately is so often the case in some colleges. In some technical schools, so far as teaching goes, the student would be just as well off reading the subjects at home as to pay for reading the subjects away from home. Any institution which expects to turn out finished pupils must employ teaching professors rather than listening professors.

We are indebted to Mr. Susan K. Huang, of Hankow, China, a student at Lehigh, for the illustrations of the Coxe and Fritz testing laboratories. The Coxe mining

Coxe Mining Laboratory

laboratory will contain in addition to a mining outfit, a milling outfit, by means of which students can become familiar with standard machines.

The Fritz laboratory will contain testing and other mechanical devices necessary to give the budding engineer an insight into practical matters, thereby enabling the recent college graduate to more readily attain to a living wage after graduation.

TEACHING FOREIGNERS ENGLISH

SOME have argued that mine foremen and others employing foreigners should learn the foreign languages in order to direct the men under them. There are two reasons particularly why this is not feasible. First, the mine boss is too busy to study, and second, he would be compelled to study all his life

John Fritz Laboratory

in order to learn the languages and dialects of the various nationalities coming within his jurisdiction. Italy alone has some 70 dialects and the Slavonic language is made up of dialects apparently.

It is proper, therefore, that the foreigner should be instructed in the English language and if the reader will turn to Doctor Peter Roberts' article on "Teaching English to Foreigners" he will find how readily this may be effectually accomplished.

MR. R. L. HERRICK, former Denver editor of Mines and Minerals, conceived the idea some years ago of awarding a thesis prize of $50 to the senior class of the Colorado School of Mines. The thesis that won the prize offered by Mines and Minerals in 1910 was entitled, "The Measurement of the Flow of Compressed Air by Means of an Orifice Meter." It was written by Jean McCallum and Duane Kelso, and will eventually appear in Mines and Minerals. If the plans of the editor can be formulated the prize will be increased in 1911 and be opened to students in all mining schools.

BOOK REVIEW

The Cyanide Handbook is the title of the latest on the cyaniding of gold and silver, by J. E. Clennell, of London. The book contains 499 pages, 10mo, is nicely printed and bound in the regular style adopted by the McGraw-Hill Book Co., who are the publishers. The price is $5.

The book is divided into nine parts headed General, Chemistry, Preparatory Treatment of Ore, The Dissolving Process, Precipitation and Smelting Process, Special Modification of the Cyanide Process, Assaying, Analytical Operations, and Metallurgical Tests. From the captions it is seen that the author is not dealing with the mechanical development of the process.

but with the chemistry entering into and needed in the economical working of the process; in fact, he intimates in his introduction that that is his object. As this is in no sense an elementary book, but one that requires the reader to be thoroughly posted in chemistry and the cyanide process, it seems as if there was too much padding for one thing and irrelevant matter for another. A cyanide man who does not know more about crushing and concentrating machinery than is given in this book would be *persona non grata*.

A chemist or assayer in a cyanide plant who could learn anything from the section on fire-assaying would, to use a "bull," not be there; *contra* there is not sufficient detail to teach one to be an assayer. No one knows better than Mr. Clennell the qualifications necessary for a position of this kind, or the needs of a cyanide mill man. Therefore we intimated that there was padding in the book. It is not, however, to be understood that the book is not a valuable addition to cyanide literature, on the contrary there is much between its covers that cannot be found in any other work on cyaniding, besides, the author has compiled and combined with his own other well-known cyanide men's practice. The section on special methods of assaying and the one on analytical operations will be helpful and a ready reference for mill men who desire to find what kind of ore they are to treat.—E. B. W.

We have received for review the sixth edition of Persifor Frazer's and Amos Peaslee Brown's TABLES FOR THE DETERMINATION OF MINERALS BY PHYSICAL PROPERTIES. The scheme for the first edition, 1874, was that followed by Albin Weisbach in his "Tabellen," 1866. The book is divided into three parts: Minerals of Metallic Lustre; Minerals of Sub and Non-Metallic Lustre; Minerals of Non-Metallic Lustre and White or Light Gray Streak. Two pages are given to columns, headed, Name, Lustre, Color, Streak, Hardness, Tenacity; Crystal System; Habit and Structure; Cleavage and Fracture; Chemical Formula; Remarks on Gravity, Reactions in Open and Closed Tubes Before the Blowpipe, and Associate Minerals. The book contains 125 pages, with flexible cover and is indexed. J. B. Lippincott Co., Philadelphia, publishers. Price $2.50. It is an excellent book for the mining engineer to use in field work.

THE APPALACHIAN MINES RECORD, published at Roanoke, Va., has just issued a new map and directory of all coal mines on the Norfolk & Western Railway. The location of each of the 169 mines is plainly shown by a number on the map referring to the list, which is printed beneath in the style of a calendar. The following details are given for each mine—all in one line across the sheet: Name of the company; name of branch line (if on one); main office of company; mine office; names and addresses of president and secretary, manager, mine superintendent, purchasing agent, and store buyer; station for local and for through passengers; shipping point for freight and express, and whether it is "prepay" or not; telegraph office; method of mining; class of mine; kind of haulage; grades of coal made; number of employes; production in 1909, and selling agents. The map is 24 in. × 36 in., mounted on cloth. The publishers announce that the information given has been gleaned since February 1, 1910, and is very complete. Price of the map is $3.75, postage paid.

COST OF MINING, by J. R. Finlay, Lecturer at Howard University on the Economics of Mining. This book is 12 mo, contains 415 pages, and 22 illustrations. As its name would imply, it deals with the economic side of mining, a subject on which too little has been written, and that at such wide intervals the student has received little benefit. The book is divided into 21 chapters as follows: I, Value of Mining Property; II, Factors Governing Variations; III, Partial and Complete Costs; IV, Statistics of Coal Production; V, Cost of Mining Coal; VI, Cost of Mining Lake Superior Iron; VII, Occurrence and Production of Copper; VIII, Lake Superior Amygdaloid Copper Mines; IX, Conglomerate Copper Mines of Lake Superior; X, Copper Mines on Fissure Veins in Montana, Australia, and Arizona; XI, Various Copper Mines of Arizona and Mexico; XII, Copper Mines in Various Other Districts; XIII, The Copper Mining Business in General; XIV, Lead; XV, Silver-Lead Mining; XVI, The Cost of Silver-Lead Mining; XVII, Zinc Mining; XVIII, Occurrence and Production of Gold; XIX, Quartz-Pyrite Gold Mines; XX, Cripple Creek, Kalgoorlie, and Goldfield; XXI, Silver Mining at Cobalt and Guanajuato.

BOOKS RECEIVED

UNITED STATES GEOLOGICAL SURVEY PUBLICATIONS, Washington, D. C.: Bulletin 415, Coal Fields of Northwestern Colorado and Northeastern Utah, by Hoyt S. Gale; Bulletin 428, Purchase of Coal by the Government Under Specifications, by George S. Pope; Bulletin 430-A, Gold and Silver, by J. M. Hill, F. L. Hess, D. F. MacDonald, and J. T. Pardee; Bulletin 460-C, Lead and Zinc, by Leon J. Pepperberg; Bulletin 430-E, Iron and Manganese, by E. C. Harder, J. L. Rich, A. C. Spencer, and Sidney Paige; Bulletin 430-G, Contributions to Economic Geology, Mineral Paints, by J. C. Stoddard, A. C. Callen F. T. Agthe, and J. L. Dynan; Bulletin 430-I, Contributions to Economic Geology, Salines, by C. L. Breger and A. R. Schultz. Water-Supply Paper 244, Part IV, St. Lawrence River Basin, by H. K. Barrows, A. H. Horton, and R. H. Bolster; Water-Supply Paper 245, Part V, Upper Mississippi River and Hudson Bay Basins, by A. H. Horton, E. F. Chandler, and R. H. Bolster; Water-Supply Paper 247, Part VII, Lower Mississippi Basin, by W. B. Freeman, W. A. Lamb, and R. H. Bolster; Water-Supply Paper 248, Part VIII, Western Gulf of Mexico, by W. B. Freeman, W. A. Lamb, and R. H. Bolster; Water-Supply Paper 249, Part IX, Colorado River Basin, by W. B. Freeman and R. H. Bolster; Water-Supply Paper 250, Part X, The Great Basin, by E. C. La Rue and F. F. Henshaw.

WEST VIRGINIA GEOLOGICAL SURVEY REPORTS OF PLEASANTS, WOOD, AND RITCHIE COUNTIES, with maps, by G. P. Grimsley. Address I. C. White, State Geologist, Morgantown, W. Va.

ACCIDENTS FROM GAS IN THE COAL MINES OF BELGIUM FROM 1891 TO 1909, by V. Watteyne, Inspector General of Mines, and Ad. Breyre, Engineer of Mines, Brussels, Belgium.

ANNALES DES MINES DE BELGIQUE, Volume 15, Part 2, issued by the Administration of Mines, Brussels, Belgium.

BULLETIN OF THE ENGINEER OF MINES OF PERU, No. 75, by F. Malaga Santolalla, Lima, Peru.

THE GOLD HILL MINING DISTRICT OF NORTH CAROLINA, by Francis Baker Laney, Joseph Hyde Pratt, State Geologist, Chapel Hill, N. C.

MINING LAWS OF OHIO, 1910, compiled by George Harrison, Chief Inspector of Mines, Columbus, Ohio.

MARYLAND GEOLOGICAL SURVEY, Volumes VII and VIII, William Bullock Clark, Superintendent of the Survey, R. W. Silvester, secretary, Johns Hopkins University, Baltimore, Md. Volume VII is a resurvey of the Mason and Dixon line between Maryland and Pennsylvania boundary.

The Knox system of blasting is used in quarrying, and consists in drilling two holes in the rock, these holes being located along the line where a fracture is desired, and about 1 foot apart. Black powder is charged in the hole, the quantity being determined by the size of the rock mass it is desired to split off. Tamping is put in the hole, but is not placed in contact with the powder, a space of 1 to 3 feet being left between tamping and powder. This affords an air cushion, which in part receives the shock of the blast. The holes are fired and recharged several times, the result being the fracture of the rock along a line corresponding to the direction of the holes. In a thoroughly homogeneous rock this system works well, and much larger masses can be detached than by feather and plug work, and much more cheaply.

CORRESPONDENCE

Surveying

Editor Mines and Minerals:

SIR:—The length of the curve required by question of Gordon L. Cox, of Wasioto, Ky., in your July number, is readily found in this case, by taking one-quarter the circumference of the circle to the radius 50 feet, thus, $\frac{3.1416\times 50\times 2}{4} = 78.54$ feet for 90°, or the one-quarter part of a circle. The degree of curve, if referred to standard railroad methods of calculation, will equal the number of degrees between the two radii and a chord of 100 feet; in this case the chord of 100 feet will equal the diameter of a circle, and the degree of curve becomes 180°. The deflection angle is one-half the degree of curve, or by deflecting 90° from the tangent and measuring 100 feet for the chord, the arc of the angle included by the radii will include 180°.

Montcalm, W. Va. E. W. Bailey

Editor Mines and Minerals:

SIR:—The following is in answer to question asked by Mr. Gordon L. Cox, in your Correspondence Column of July, 1910, relative to 50-foot radius curve of 90 degree total angle: Total angle, 90 degrees; length of curve, 78.54 feet; degree of curve, 114° 35′ 24″; radius, 50 feet. Grenville Lewis

Pineville, Ky.

Thermochemistry of Anthracite

Editor Mines and Minerals:

SIR:—From a comparison of the relative ash contents and British thermal unit values of the deliveries of furnace coal to the various Government buildings at Washington, tabulated by Messrs. Randall and Holmes in your issue of August, 1907, it would appear that the ash does not act merely as a dilutent, for the lower ash coals show very uniformly a higher calorific value than they should as compared with the mean, and vice versa. It would be interesting to learn whether any of your readers have made a similar observation on other coals, or whether the above is merely a coincidence.

13 James Street, Cardiff R. T. Hancock

Determination of the Meridian

Editor Mines and Minerals:

SIR:—Mr. James Underhill, in common with a good many surveyors, should thank Professor Rowe for drawing his attention to the influence of altitude on the horizontal angle subtended by the sun's diameter. It is of great interest to note that the true position of the sun's center should be worked out separately for each observation, when these are taken to the limbs, and not calculated from the mean. The importance of this increases with increasing altitude and with increasing interval between observations in time. The correction must be worked out on the observed altitudes. At about 45 degrees the sun's semidiameter subtends a horizontal angle which increases or decreases nearly a minute with every interval of two minutes of time.

A clear conception of the effect of altitude can be obtained by imagining the transit set up at the radial center of an immense arch, the width of the arch being such that with the telescope level it subtends a horizontal angle of 32 minutes. If the leveled telescope is sighted to one edge of the arch and then slowly revolved upwards, the intersection of the cross-hairs will gradually leave the edge and travel toward the middle, to which it will point when the telescope is vertical. The horizontal reading has not altered all this time, and as a similar result is produced if the telescope is first directed to the other edge, it follows that the sun, which fitted into a space 32 minutes wide when on the horizon, would have to diminish in size as it rose in order to continue to do so, till it eventually became a mere speck at the zenith. As the sun of course does nothing of the kind, it follows that the horizontal angle subtended between its right and left limbs increases with increasing altitude.

It would be unfortunate if the respect which Mr. Underhill's opinion inspires among your readers should lead any of them to believe that doubled observations correct errors in the vertical angle due to faulty leveling of the instrument. They do not, of course, as any one can convince himself who will take the trouble to set up askew and try it. The only error in the vertical angle thus corrected is the index error of the vertical arc, and in this connection it may be useful to point out that in the plain theodolite, where the telescope reverses in the Y's, it will not even do this, unless the level of the telescope is so adjusted that the bubble remains in the center of its run when the telescope is reversed end for end. This adjustment is effected as follows: Level up the instrument in the manner described in Professor Rowe's article, lift telescope out and reverse. If the bubble moves, take any vertical angle by doubled observation to obtain the index error, which will be half the difference between the two readings. Set the vernier to this error, and with the telescope over one pair of main leveling screws bring the bubble into center by means of these. Lift out telescope and reverse, and bring the bubble back half way to center by these same screws and the remainder by the screws or nuts which attach the level tube to the telescope. If the collimation line has been previously adjusted in the manner customary with this class of instrument, that is to say, by twisting the telescope around in the Y's, this adjustment will not be thrown out. If, however, it has been adjusted by the more accurate method of sighting on to leveled pegs, it will now require altering.

As Professor Rowe says, errors in vertical arc produce an error of somewhat greater magnitude in the sun's azimuth. Errors in latitude or in declination produce their greatest effect near midday, but if of the same sign, tend to cancel each other. This is an additional reason for not trusting to observations which require the prismatic eyepiece. In doubled observations the sun should be observed in diagonally opposite quadrants if there is any doubt as to the cross-hairs being accurately at right angles to each other. Care should be taken that the horizontal cross-hair really is horizontal.

Cardiff, England R. T. Hancock

Lamp Burns in Water

Editor Mines and Minerals:

SIR:—We have a pump at the bottom of a slope which is about 5,000 feet long and delivers about 1,000 gallons per minute. The column pipe is a 10-inch pipe and it gets its water from the lower old workings. I can hold my naked lighted lamp right in the center of the discharge stream of water or I can take it right through the stream slowly and the light will continue to burn. I have held my light in the center of stream for a period of one-half minute time and it still burned. What is the cause of the lamp burning in the water? Student

Pumping

Editor Mines and Minerals:

SIR:—In answer to Wm. H. James' question on "Pumping," in the July number, I give the following solution: $400\times 60\times 24 = 576{,}000$ gallons = capacity of sump. While the sump is being emptied there is flowing into it $400 \times 60 \times 20 = 480{,}000$ gallons, or a total of 1,056,000 gallons to be pumped in 20 hours, making the capacity of the pump $\frac{1{,}056{,}000}{20} = 52{,}800$ gallons per hour, and $576{,}000 \div 52{,}800 = 10.91$ hours, nearly, say

11 hours it will take the pump to empty the sump if it is full and the supply cut off.

The dimensions of a pump to perform this amount of work will depend on the speed at which it is run, which for practical reasons should not exceed 100 feet per minute. The pump has a capacity of $52,800 \div 60 = 880$ gallons per minute. The following formula gives the diameter of the plunger and also makes allowance for slip and leakage past the glands: $5.535 \times \sqrt{\frac{800}{200}} = 11.6$ inches. But the size of the pump can be gotten by trial, thus: Assuming a common duplex steam pump, say 10 in. × 36 in., $10^2 \times 36 \times .0034 = 12.24$ gallons per stroke; $880 \div 12.24 = 72 \times 3 = 216$ feet. Making an allowance of 2 feet per minute for slip makes the speed for each piston equal $\frac{218}{2} = 109$ feet per minute, which is quite common practice. This trial gives the smallest size of plunger that should be used, as a larger size would give better results. The length of stroke is 36 inches, but the diameter of piston cannot be calculated from the question, as this depends on such factors as the vertical height to which the water has to be pumped, and the length of the discharge pipe, together with its diameter and the steam pressure carried. J. T. R.

Editor Mines and Minerals:

SIR:—I would offer the following in reply to Mr. Wm. H. James' question in the July issue of MINES AND MINERALS. The mine makes 400 gallons of water per minute, or $400 \times 60 \times 24 = 576,000$ gallons per day of 24 hours. The pump can empty the sump in 20 hours, $576,000 \div (20 \times 60) = 480$, hence, it delivers 480 gallons per minute. If the supply is cut off the pump can empty the sump, which is full, in 10 hours 54.54 minutes, for the pump can handle 480 gallons per minute, and the supply being 400 gallons per minute, taken together, will equal 880 gallons per minute, or $576,000 \div 880 = 654.54$ minutes or 10 hours 54.54 minutes. A double-action pump with 12-inch diameter plunger and 24-inch stroke making 50 strokes per minute would be able, theoretically, to handle 587 gallons per minute, therefore, a pump of the above dimensions should be sufficient to take care of 480 gallons per minute. E. W. BAILEY

Montcalm, W. Va.

MEETINGS OF INSTITUTES

MINE INSPECTORS' INSTITUTE

The second annual meeting of the Mine Inspectors' Institute of the United States of America, was held in Chicago, June 13–17, inclusive. A meeting of the Executive Board was held Monday, June 13, and by the evening of that day, delegates representing nearly every coal-mining state had registered. The local Committee of Arrangements consisted of the following inspectors, representing the full board of Illinois inspectors: James Taylor, of Peoria, chairman; Thomas Moses, of Westville, secretary; and John Dunlop, of Peoria, treasurer.

Tuesday, June 14, the Institute was called to order at 9.30 A. M. by President Geo. Harrison, of Ohio. The address of welcome on the part of the state of Illinois was given by Mr. David Ross, Chief of the Illinois Bureau of Labor Statistics. Corporation Council Bagby, of Chicago, extended to the Institute the welcome of the city of Chicago. To these addresses Mr. Harrison responded and then delivered his presidential address. Following this, came the appointment of the following committees.

Membership: Messrs. Epperson, of Indiana; Dunlop, of Illinois; Rhys, of Iowa. Executive Business: Messrs. Adams, of Pennsylvania; McDermott, of Montana; Hanraty, of Oklahoma. Constitution: Messrs. Moses, of Illinois; Grady, of West Virginia; Neill, of Alabama. Resolutions: Messrs. McDonald, of Ohio; Johnson, of Pennsylvania; Botting, of Washington.

The afternoon session of Tuesday was devoted to the reports of the secretary and treasurer, and to addresses by Mr. John H. Walker, President of the United Mine Workers of Illinois, and Mr. A. J. Moorshead, President of the Illinois Coal Operators' Association. A very lively discussion took place as to the advisability of endorsing some one for the head of the Bureau of Mines recently created under the Interior Department, but it was voted not to do so.

In the evening of Tuesday, a banquet and smoker was given at the Grand Pacific Hotel.

Wednesday afternoon was devoted to an automobile ride about the city of Chicago. In the evening a banquet was given at the Chicago Automobile Club, the host being Mr. W. W. Taylor, General Superintendent of the St. Paul Coal Co. After this banquet, the following papers and addresses were given: Discussion of the report upon uniformity of mine statistics; "Timbering in Metal Mines," by F. Cushing Moore, State Mine Inspector, Boise, Idaho; "Tests With Explosives in the Field," by Clarence Hall, Pittsburg, Pa.; an address by Mr. John Mitchell.

The program of Thursday was as follows: Address by Mr. G. A. Traer, of Chicago; "What Mine Inspectors Should Not Do," by Hon. David Ross, Springfield, Ill.; "Mine Ventilation," by J. B. McDermott, State Coal Mine Inspector, Helena, Montana; "The Ventilation of Safety Lamps," by E. A. Hailwood, Morely, England; "Mine Explosives," the general discussion being led by Mr. John Verner, of the state of Iowa. In the afternoon: "The Education of Mine Employes," by H. H. Stoek, Professor of Mining Engineering, University of Illinois; "Mine Fires," with general discussion, led by T. K. Adams, Mine Inspector, Mercer, Pa.; "Mine-Rescue Apparatus and Its Use by the Urbana Station, Illinois," by R. Y. Williams, Mining Engineer, Urbana, Ill. Evening—visit to Sans Souci Park. On account of the limited time and the absence of the writers, the reading of several of these papers was omitted and the papers ordered printed.

The closing meeting of Friday morning was devoted largely to business. The Committee on Constitution reported an amendment striking out Section 4. This section read as follows: "And such other professionals as majority of the members of the institute legalize as members for the best interest and advancement of the undertakings of the Institute." The constitution was also changed so that hereafter no officer excepting the secretary can succeed himself in office. Charleston, W. Va., was chosen as the place of meeting for the next annual meeting and the following officers were elected for the ensuing year: President, George Harrison, Chief Inspector Mines, Columbus, Ohio; First Vice-President, Thomas Moses, District State Mine Inspector, Westville, Ill.; Second Vice-President, John Verner, District State Mine Inspector, Charlton, Iowa; Third Vice-President, Robert Irving, District State Mine Inspector, Cayuga, Indiana; Treasurer, Thomas Hudson, District State Mine Inspector, Galva, Ill.; Secretary, James W. Paul, Mining Engineer, Pittsburg Pa.; Assistant Secretary, P. A. Grady, District Mine Inspector, Huntington, West Virginia.

COAL MINING INSTITUTE OF AMERICA

The summer meeting of the Coal Mining Institute of America, was held at Uniontown, Pa., June 28–29, by invitation of the Independent Coke Producers' Association. This meeting was probably the most successful in the history of the Institute, not only in point of numbers, broadness and variety of papers read, but in other matters of interest. President H. H. Stoek delivered an address on "Coal Mining in Illinois." The new constitution and by-laws were discussed and adopted after slight revision. Mine Inspector Thomas K. Adams read a paper on "Mine Fires and Their Prevention," in which he urged carefulness on the part of mine manager and mine boss.

An interesting paper was then read by Sion B. Smith, Esq. of Pittsburg, on the "Legal Responsibilities of Employers and Employes," in which he stated that with no indemnity fund for the injured, they or their families become local or state charges, and increase the taxes of those not especially interested in the mining of coal or making of coke. He suggested a provident savings fund based on wages of men, and a like sum by the company. Mr. F. C. Keighley was toastmaster at the lunch given at the Country Club. Dr. Peter Roberts, secretary of the Y. M. C. A., New York City, described his method of teaching foreigners English. His speech is given elsewhere, although it requires Doctor Roberts' personality to make it fully effective.

Professor Stoek said there had been an awakening in recent years on the scientific phases of coal mining. Prof. W. R. Crane, of Pennsylvania State College, spoke along educational lines. J. W. Paul, among other things said mine-rescue stations would be established in various parts of the United States within the next year. Editors H. P. Snyder and John O'Donnell, spoke briefly, then followed E. B. Day and J. B. Johnston of the *Coal and Coke Operator*. Doctor Johnston spoke on coke waste and made interesting references to the historical side of mining. Attorney James R. Cray, of Uniontown, spoke of the developments in the coke industry about Uniontown during his recollection.

On Wednesday morning the following papers were read: "Mine Layouts," by W. E. Fohl, Pittsburg. His proposed plan of shaft bottom caused some discussion, and will be continued later. W. M. Judd, a member of the engineering firm of W. G. Wilkins Co., Pittsburg, presented a paper on "Mining Towns and Mining Houses." H. N. Fitch, Breta, W. Va., read a paper on "The Systematic Handling, Disbursement, and Accounting of Mine Supplies." Wednesday afternoon a trip was made to the Continental Coke Plant No. 1, of the H. C. Frick Coke Co. The Institute was the guest of the officials of the company, who explained the most modernly equipped coking plant in the Connellsville region, and then took those who cared to go into the dark hole. The next meeting of the Institute will be held in December in Pittsburg.

HAULAGE-ROPE STRAINS

There is probably no greater strain placed upon a haulage rope than that which it is subjected to when starting a heavy load on a steep gradient in a reckless manner. When starting under such circumstances the rope is subject to such a tensional shock that the strands soon become broken, and much expense is then incurred in keeping the rope in repair. The life of the rope is also shortened, a new rope often being required.

To remedy this the rope should be started slowly, and the full pressure not put upon the engines suddenly.

In main- and tail-rope haulage the "shunts" or "stations" should not be made in that part of the road where the gradient is heaviest, but in a less gradient than any other part of the road, so that the load may be taken gradually.

Another common cause of great strains being placed upon haulage ropes, especially in endless-rope haulage, is that of allowing cars to leave the rails, owing to poor tracks. The cars in such cases either travel some distance over the ties or get fast in the timbers at the sides, or collide with the cars traveling in the opposite direction, before it is ascertained that they are off the rails; this subjects the rope to jerks that often prove disastrous to the life of the rope. Another method of straining ropes is to use pulleys, around which the rope has to pass, too small for the size of the rope. The following rules, if carried out, will greatly minimize the strains on haulage ropes, and lengthen the life of the rope:

1. Avoid jerking at starting, or when the engine cannot overcome the load. 2. Have all haulage roads kept in good condition, as upon the tracks depends the amount of strain to which the rope will be subjected and also the amount of output; badly laid tracks are detrimental to a large output. 3. In endless-rope haulage have **V** plates fixed in the roads at short distances apart, so that the cars will not travel far when off the rails. 4. Have pulleys large enough to insure a good turn being given to the rope. 5. Never run an endless rope empty, but keep a load upon it, if possible; this may not always be practicable. 6. Have engine powerful enough to move the load at any time

A NEW TRANSPORTATION SYSTEM

The Automatic Transportation Co., of Buffalo, N. Y., will soon complete the erection of their new system of transportation at Blossburg, Pa., which will be used to transport the output of several mines from the mountain side to the railroad. This transportation line is the first of its kind to be constructed in the state of Pennsylvania.

A new principle in the application of electricity to transportation of coal and ore from mines to railroad or smelters is successfully applied.

The system consists of parallel steel rails elevated from the ground on **Y**-shaped iron posts at a height of from 4 to 30 feet above the surface, as conditions require. The motored cars and carriers operate automatically on these rails at any

Motored Car and Track

desired rate of speed from 5 to 30 miles an hour. The motored cars haul several trailers. The standard gauge is 30 inches.

One of the important points of superiority that this means of transportation possesses is that it may be constructed over the roughest character of country where the problems confronting railroad building make the cost prohibitive. Its operation may be maintained at a minimum expense, and the line can be kept open under all weather conditions.

The ore cars with trailers convey the products of the mines at a high rate of speed, direct to bins or smelters, automatically dumping their loads, reversing and returning immediately to the starting point. The carriers have a capacity of several tons and will climb exceedingly heavy grades.

The important point wherein this system differs from all others in the adaptation of electricity to the problems of transportation is the fact that the carriers take the current into the motors direct from the rails upon which they travel, thus doing away with the third rail and the overhead trolley; another most important point is that all of the operations of transportation are accomplished automatically with the greatest exactness.

The company is closing several important contracts which include the moving of various ores as well as the products of coal mines. The headquarters of the Automatic Transportation Co. are at 2933 Main Street, Buffalo, N. Y., where an extensive factory has been erected. The officers of the concern are William C. Carr, the inventor of this system, president; Joel H. Prescott, secretary and treasurer.

THE MULGA MINE EXPLOSION

Written for Mines and Minerals

On April 20, 1910, an explosion occurred at the Mulga coal mine near Birmingham, Ala., in which 39 lives were lost.

Conditions Existing in the Mine. Probable Cause of the Explosion

Mulga is 12 miles west from Birmingham, on the Ensley Southern Railroad, and the mine is operated by the Birmingham Coal and Iron Co., a corporation which believes in protecting the lives of its employes by systematic work and modern methods. There are two timbered shafts, 720 feet deep, 385 feet apart and cutting two coal beds, the Pratt at 220 feet depth and the Mary Lee at 720 feet, although the latter was not being operated, the top bed only being worked. The shafts are equipped with structural-steel head-frames and double-decked cages. Electric haulage is used in the mine, which has been in operation about two years. The mine is ventilated by a Clifford fan, which at low speed produces 50,000 cubic feet of air per minute. About 10 per cent. of the intake air came through No. 1 shaft, and 90 per cent. through No. 2 shaft. According to Mr. James Hillhouse, Chief Inspector of Mines in Alabama, the mine is worked in two shifts, permissible explosives only are used, the mine being somewhat gassy, and a sprinkling system is installed, to keep down the dust.

The mine was badly wrecked by the explosion which, according to accounts, shot heavy objects out of the shaft high in the air. The cages in No. 1 shaft were wrenched from their guides and could not be used for some time. The cage in No. 2 shaft was at the bottom of the shaft at the time of the explosion, and was so fastened down that it could not be pulled up. The accident occurred at 9.10 P. M. and the fan was put in operation at 11.20 P. M., thus showing it was little damaged.

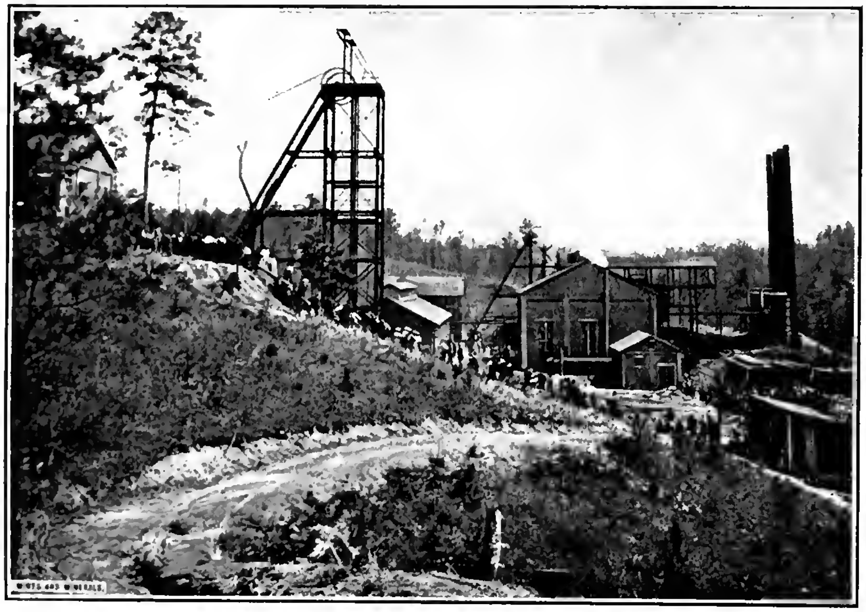


FIG. 1. MULGA MINE

When General Superintendent Hastings reached the shaft after the accident he found a natural air-current going in No. 1 shaft sufficiently strong to flatten out the flame of a driver's lamp, on all sides of the shaft. Immense volumes of smoke were coming from No. 2 shaft, and continued until 7 A. M. the next morning. The smoke smelled of fire and led the manager to believe that if there was fire it was close to No. 2 shaft, and this reasoning was given further weight by the reversal of the air-current from shaft No. 1 to No. 2.

The manager believed, and his belief was later substantiated, that the negroes would from impulse try to reach the shaft through which they entered the mine, and if any of the men reached No. 1 shaft between the time of the explosion and the time when the fan began to operate it would have been death to all had No. 1 shaft been made the upcast.

Arriving at this decision, which was perfectly sound under the premises, and bearing in mind that explosions travel towards the intake, the fan was reversed when repaired and No. 2 shaft made the upcast. Two hours after the explosion, Mr. Johns, the mine foreman, accompanied by another man, went down No. 1 shaft in a bucket and explored for 700 or 800 feet, east, west, north, and south of the shaft without helmets. They then went to No. 2 shaft, and released the cage which was fastened by some timbers that were driven in by the explosion. This occurred about 5 A. M. and Mr. Johns realized that by this time all those that were in the mine when the disaster occurred were dead. By this time assistance had arrived from the Pratt mines, Ensley, Wylam, and Blossburg and all other mines in the vicinity of Mulga. An advance guard with safety lamps and a relief party of 6 or 8 men was organized who went in seach of the dead.

The plan advised by the inspectors was to turn all air into each heading separately and explore them separately. This plan was carried out by cloth brattices carried to the face of each entry. These relief parties worked in alternate shifts of two hours, until they had aggregated 6 hours out of 12 when they were relieved. The position of the bodies represented by the crosses on the map, Fig. 2, would indicate that the men attempted to retreat from the afterdamp or were killed immediately by the concussion. On account of the mine being well ventilated considerable damage was done inside the mine and bodies that were not burned, were badly mutilated. The last of the bodies was removed Sunday evening, April 24, four days after the explosion.

On Monday, April 25, four days after the explosion and after all bodies had been removed, Chief Mine Inspector J. J. Hillhouse and his assistant Robert Neill, accompanied by J. J. Rutledge of the United States Geological Survey, entered the mine for the purpose of ascertaining the cause of the explosion. They attempted to locate a windy shot, but were unable to do so as all coal was found undermined, and the shot firer was found where no mining was going on. Portions of two boxes of permissible explosives with the 15 sand dummies used for tamping were found intact. Firing was done in this mine by connecting the detonator wires with the electric trolley wires, and no indications that this had been done could be found. At the time of the explosion two coal cutting machines were at work undermining and men were engaged in loading the "buck dust" or cutter chips from the machines. This was done daily and, as a further protection against dust explosion, sprinklers were used.

The inspectors having satisfied themselves that the cause of the explosion was not a "windy shot," next turned their attention to the gas. They found some at the extreme points of operation, but the accumulation may have taken place after all the brattices had been blown out by the force of the explosion.

Mr. Hillhouse states that the explosion seemed to have greatest force in locations known to be free from gas and dust, and that ponds of water standing in several places were covered with a solid crust of dust. He gives it as his opinion that a

small body of gas was ignited in some part of the mine, and the flame traveled until it met the dust from the machines, which was suspended in the mine atmosphere, and igniting this increased the force of the explosion. Morgan Johns who was night fire boss and shot firer at Mulga reported at 6:30 p. m., three hours before the explosion, that he found some gas in four headings, but that he left all working places free from gas. He then returned to the mine and was found dead on a haulage heading, showing that no shots had been fired by him at 9:15, the time the explosion occurred. The report sent to Northern newspapers that the government officials were in charge of the mine was wrong, the Alabama State Mine Inspectors were in charge until after the mine was cleaned up and their reports made.

Mulga was a modern mine and a model one. Mine Inspector Neill made an investigation two days before the explosion occurred and found matters satisfactory.

Mulga No. 1 shaft is 15 ft. × 26 ft. in size over all and is divided into three compartments, the north compartment being the upcast air shaft. The other two compartments are used for hoisting coal. The air compartment is 8 ft. 4 in. × 13 ft. 8 in., and the hoisting compartments are 7 ft. 4 in. × 13 ft. 8 in. inside dimensions. Double-decked steel cages are used in the hoisting compartment, the upper deck carrying the loaded car of coal, and the lower deck returning the empty car to the line. No. 2 shaft is similar to No. 1, except the over all dimensions are 13 ft. × 26 ft., instead of 15 ft. × 26 ft. The north compartment at No. 2 shaft is the intake or downcast for the ventilation current. The tipple is equipped with a Heyl-Patterson rotary dump and an endless-belt picking table of steel, together with the necessary coal and rock chutes, etc. No. 2 shaft has a steel head frame and steel cages similar to No. 1 shaft, but no tipple and is used only for ventilation and to hoist men. East of each shaft is located a hoist house, 50 feet square, built of brick with concrete foundations and floors, and structural steel roof trusses. Each hoist house contains two first-motion Vulcan hoisting engines with cylinders 26 in. × 42 in., with coned winding drums having a capacity of 800 feet of 1½-inch wire cable.

The fan house near No. 1 shaft is constructed of brick and concrete. The fan way is built of concrete and connects with the air shaft below the surface of the ground. The top of the air compartment of the shaft is covered with an explosion door. The fan is 15 feet in diameter and driven by a 250-horsepower (230 volts) General Electric motor. There is also an 18″×24″ 155-horsepower engine to drive the fan in emergencies.

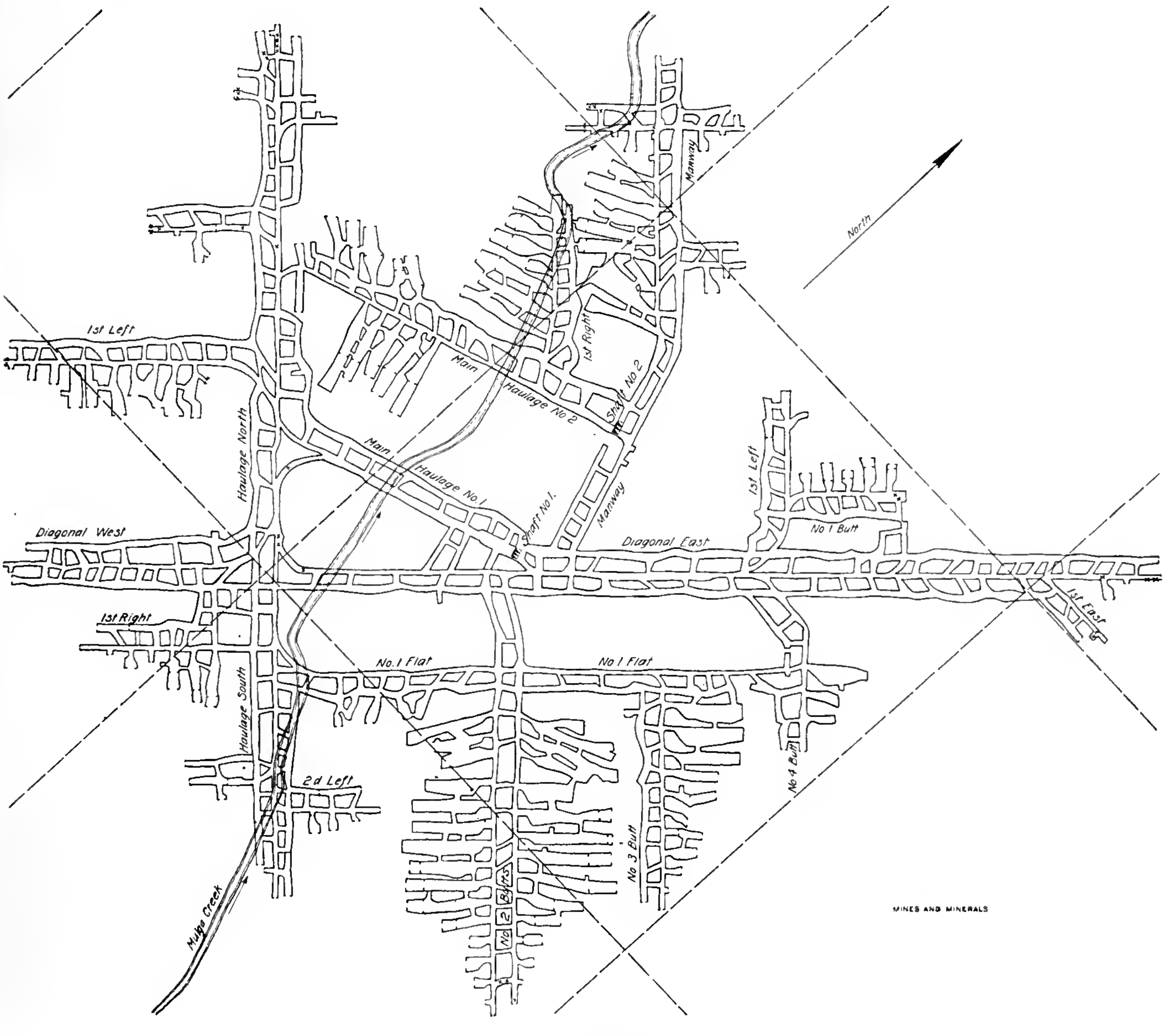


Fig. 2. Map of Mulga Mine

MECHANICAL STOKERS AND HAND FIRING

Results of Tests on Manning Boilers With Taylor Underfeed Stoker and Hand Firing

There has been much written and tabulated on fuel tests, but seldom are comparative tests between stokers and hand firing made and tabulated. It is well known that under certain conditions mechanical stokers are preferable to and more economical than hand firing, but just how much more economical is a question that can only be determined by actual tests. There have been instances where mechanical stokers have been discarded and hand firing substituted, the latter being more economical owing to the kind of coal used for steam raising. There have been cases where preference was given to underfeed rather than overfeed stokers, after the latter had been fully tested. Most engineers and coal consumers are familiar with the underfeed type of mechanical stoker, with its automatic rams, also with the overfeed stoker with its rocking grate and again with the moving chain grate stoker, each type having distinguishing names; it is assumed in this synopsis of comparative tests that the reader is familiar with the constructive details of the ordinary forms of mechanical stokers. The comparative tests were made at the Nashua Mfg. Co., Nashua, N. H. Robert Amory, Jr made the tests, using four boilers, and his results are tabulated below:

Principal conditions governing the trial. Ordinary running conditions of plant with average firing load on at 6 A. M., and off at 6 P. M., pumps, blowers, and stoker feed stopped from 12 to 12:50 P. M.

Stoker Test

Kind of fuel........George's Creek, Cumberland, Jackson Co.
Kind of boiler....Standard "Manning" type mounted on brick setting, fitted with one two-retort Taylor Gravity Underfeed Stoker and forced draft
State of weather..............Cloudy at first, clearing later
Method used in test...Alternate method A. S. M. E. Code 1899
Date of trial... ....November 29, 1906
Duration of trial.........7 A. M. to 6 P. M., 11 hours (figured on basis 10.2 hours, standby losses included)
Stoking surface.. ....Width 3 ft. 10 in.; length, 6 ft., 0 in. Area, 23 00 sq. ft.
Height of furnace..............................7 ft. 8½ in.
Approximate air spaces (stoker)...................136 sq. in.
Ratio of stoking surface to air opening..................24.36
Water-heating surface.......................1,300.17 sq. ft.
Superheating surface..........................453.43 sq. ft.
Ratio of water-heating surface to grate surface...........56.5
Ratio of stoking surface to minimum draft area.........4.68

Hand Fire Test

Kind of fuel..........George's Creek, Cumberland, Jackson Co.
Kind of furnace.......Regular circular water leg Manning type
State of weather........Overcast and damp in morning, later clear and cold
Method used in test..Alternate method A. S. M. E. Code 1899
Date of trial............................December 7, 1906
Duration of trial................7 A. M. to 6 P. M., 11 hours (figured on basis 10.2 hours, standby losses included)
Grate surface.....width circular length...area...28.27 sq. ft.
Height of furnace ....4.76 in.
Water-heating surface...................... 1,300.17 sq. ft.
Superheating surface,.........................453.43 sq. ft.
Ratio of water-heating surface to grate surface.......45.9 : 1
Ratio of minimum draft area to grate surface........1 : 5.48.

Average Pressures

	Stoker Test	Hand Firing
Steam pressure by gauge, pounds	113.06	110.05
Force of draft between damper and boiler, inches of water	.25	.39
Force of, or blast in ash pit, inches of water	2 00	

Average Temperatures

	Stoker Test	Hand Firing
Of external air, degrees F.............	39.0	21.5
Of fireroom, degrees F...............	73.4	67.5
Of steam, degrees F..................	386.7	365.6
Of feedwater entering boiler, degrees F..	104.7	101.0
Of escaping gases from boiler, degrees F	468.7	480.5

Fuel

	Stoker Test	Hand Firing
Size and condition....................	Run of mine good condition	Run of mine good condition
Weight of coal as fired, pounds.........	20,483	21,492
Percentage of moisture in coal, per cent.	2.60	2.60
Total weight of dry coal consumed, pounds..........................	19,950	20,933
Total ash and refuse..................	1,541	1,102
Quality of ash and refuse..............	Soft and clean	Light and full of coke
Total combustible consumed, pounds....	18,409	19,831

Proximate Analysis of Coal

	Stoker Test	Hand Firing
Fixed carbon, per cent.................	78.00	78.00
Volatile matter.......................	13.28	13.28
Moisture.............................	1.00	1.00
Ash..................................	7.72	7.72
Per cent..........................	100.00	100.00

Analysis of Ash and Refuse

	Stoker Test	Hand Firing Per Cent.
Combustible.........................	Not Made	47.7
Earthy matter		52.3

Fuel Per Hour Per Boiler

	Stoker Test	Hand Firing
Dry coal consumed per hour, pounds....	489.000	527.000
Combustible consumed per hour, pounds	451.000	486.000
Dry coal per square foot of grate surface per hour, pounds..................	21.260	18.640
Combustible per square foot of water heating surface per hour, pounds.....	.347	.374

Calorific Value of Fuel

	Stoker Test	Hand Firing
Calorific value by oxygen calorimeter per pound of dry coal, B. T. U...........	14,790	14,790

Quality of Steam

	Stoker Test	Hand Firing
Percentage of moisture in steam.......	None	None
Number of degrees in superheating, degrees F.........................	40.9	21.4

Water

	Stoker Test	Hand Firing
Total weight of water fed to boiler, pounds	198.895	180.233
Equivalent water fed to boiler from and at 212 degrees (K 1.154), pounds.....	229.580	(K1.158) 208.660
Water actually evaporated, corrected for gauge level, pounds.................	199.141	180.012
Factor of evaporation.................	1.178	1.169
Equivalent water evaporated into dry steam from and at 212 degrees, pounds	234.447	210.548

Water Per Hour

	Stoker Test	Hand Firing
Water evaporated per hour corrected for quality of steam, pounds	19.500	17.760
Equivalent evaporation per hour from and at 212 degrees, pounds	22.984	20.642
Equivalent evaporation per hour from and at 212 degrees per square foot of total heating surface, pounds	3.250	2.960

Horsepower Per Boiler

	Stoker Test	Hand Firing
Horsepower developed, (34½ pounds of water evaporated per hour into dry steam from and at 212 degrees, equals 1 horsepower)	166.5	149.6
Builder's rated horsepower	150.0	150.0
Percentage of builder's rated horsepower developed, per cent	111.0	99.8

Economic Results

	Stoker Test	Hand Firing
Water apparently evaporated under actual conditions per pound of coal as fired, pounds	9.71	8.37
Equivalent evaporation from and at 212 degrees per pound of coal as fired, pounds	11.45	9.81
Equivalent evaporation from and at 212 degrees per pound of dry coal, pounds	11.75	10.06
Equivalent evaporation from and at 212 degrees per pound of combustible, pounds	12.73	10.62

Analysis of Dry Gases

	Stoker Test	Hand Firing
Carbon dioxide (CO_2)	8.86	8.10
Oxygen (O)	11.08	12.12
Carbon monoxide (CO)		1.39
Hydrogen and hydrocarbons } Nitrogen (by differences) (N)	80.06	78.39
	100.00	100.00

Calculations From Gas Analysis

	Stoker Test	Hand Firing
Pounds air per pound of combustible, pounds	22.01	21.1
Pounds air per pound of carbon, pounds	26.05	25.6
Dilution per cent. (excess air) per cent	125.50	122.0

Heat Balance

	Stoker Test	Hand Firing
Heat absorbed by water, per cent	76.74	65.68
Heat lost in flue gases, per cent	14.02	13.30
Heat lost by incomplete combustion { on grate / by CO		10.72
Heat lost by evaporation of moisture in coal	.715	.715
Radiation, conduction and unconsumed hydrogen in flue gases	8.52	9.58
	100.00	100.00

Stokers and Fan Blower

	Stoker Test	Hand Firing
Steam used per pound of steam evaporated by boilers per cent. (assumed from Quincy Point). (Test probably too high) per cent	3.2	

The increase in economy of the Taylor stoker plant as compared with the hand-fired plant is 16.84 per cent.

If anything, this figure should be increased, as the hand-firing tests fires were very dirty at the end, as shown by the light ash; there was about 250 pounds of ash in each fire at the end.

Catalogs Received

In writing for catalogs, please mention Mines and Minerals

Merritt & Co., Camden, N. J., Progress in Sewage Treatment, 24 pages.

Foote Mineral Co., Philadelphia, Pa., A Series of American Rocks, 40 pages.

J. H. Williams & Co., Brooklyn, N. Y., Photographs of Chain Pipe Tools and "Ratchetless-Ratchet-Rench."

Paint Manufacturers Association, Philadelphia, Pa., Bulletin 23, The Theory of Driers and Their Application, 20 pages; Bulletin 24, Some Iron Oxides and Their Values, 38 pages; Bulletin No. 25, 1909 Report on Examination of North Dakota Test Fences, 59 pages.

Queen & Co., Inc., Philadelphia, Pa., Highest Grade Engineering Instruments, 20 pages.

Ackroyd & Best, Ltd., Morley, near Leeds, England, Safety Lamps, 34 pages.

Western Electric Co., New York, N. Y., Bulletin No. 1110-1, Telephone and Signaling Apparatus for Mines, Mine Plan and Wiring Diagram, 8 pages; Bulletin No. 5533, Hawthorn Series Incandescent Lighting System With Sunbeam Mazda Lamps, 16 pages.

Hendrie & Bolthoff Mfg. and Supply Co., Denver, Colo., Vulcan Steel Frame Electric Hoists, 16 pages.

Holland Trolley Supply Co., Cleveland, Ohio, Catalog A, Holland Trolley Supplies, 32 pages.

Jeffrey Mfg. Co., Columbus, Ohio, Bulletin No. 12A, The Care of Electric Mine Locomotives in Service, 71 pages; Bulletin No. 13, Jeffrey Industrial Locomotives, 32 pages; Bulletin No. 15, The Jeffrey Crab Locomotive, 16 pages; Bulletin No. 16A, The Jeffrey A-5 Electric Rotary Drill, 15 pages; Catalog No. 31C, Jeffrey Pulverizers and Crushers, 38 pages.

Ingersoll-Rand Co., 11 Broadway, New York, N. Y., Class "O" Duplex Steam-Driven Air Compressors, 24 pages; Class "OC" Duplex Corliss Steam-Driven Air Compressors, 24 pages; "Crown" Sand Rammers, 8 pages.

Joseph Dixon Crucible Co., Jersey City, N. J., Volume XII, Graphite, 12 pages.

The Ore Concentration Co. (1905) Ltd., 4 Broad Street Place, London, E. C., England, Bulletin No. 27, Elmore Vacuum Process, 8 pages.

American Blower Co., Detroit, Mich., Blower Equipment for the Modern Foundry, 23 pages.

Ohio Brass Co., Mansfield, Ohio, Catalog H, Ohio Valves and Steam Specialties, 56 pages; O-B Bulletin of Electric Railway and Mine Haulage Material, 24 pages.

John A. Roebling's Sons Co., Trenton, N. J., circular entitled "By Way of Reminder."

Denver & Rio Grande Railroad, Denver, Colo., With Nature in Colorado, 32 pages.

Cutler-Hammer Clutch Co., Milwaukee, Wis., Lifting, Cutler-Hammer Magnets, 8 pages; Battery Charging Rheostats, 48 pages.

Buettner & Shelburne Machine Co., Terre Haute, Ind., Price Lists of Chain Coal-Mining Machines.

H. W. Johns-Manville Co., Cleveland, Ohio, The J-M Packing Expert, Nos. 2, 3, and 4, describing asbestos and packings; The J-M Roofing Salesman, describing J-M roofing.

General Electric Co., Schenectady, N. Y., Bulletin No. 4736, Lightning Arresters, 28 pages; Bulletin No. 4737,

Electric Hardening Furnace, 8 pages; Bulletin No. 4738, Belt-Driven Revolving Armature Alternators, 4 pages; Bulletin No. 4741, Luminous Arc Lamps for Direct-Current Multiple Circuits, 4 pages; Curtis Turbine Installations, 56 pages; Transformer Manufacturing Facilities, 24 pages; G-I Flame Arc Lamps, 16 pages; Sewing Machine Motors, 8 pages.

DeLaval Steam Turbine Co., Trenton, N. J., DeLaval High Efficiency Centrifugal Pumps, 96 pages.

Hunt Filter Co., San Francisco, Cal., Hunt Continuous Filter, 8 pages.

Wisconsin Engine Co., Corliss, Wis., Bulletin C-4, Heavy Duty Corliss Engines, 24 pages.

Stromberg-Carlson Telephone Mfg. Co., Rochester, N. Y., Bulletin No. 1000, Mine Telephones, 16 pages.

Bristol Company, Waterbury, Conn., Bulletin No. 150, Bristol's Recording Instruments, 64 pages.

CHARGING BLASTING HOLES

Written for Mines and Minerals

The use and abuse of explosives is attracting as much, if not more, attention than any one subject now being considered by mining men, whether considered from the standpoint of the miner or the operator. Elaborate and costly experimental plants have been installed within a few years, by the manufacturers of explosives and by the United States Geological Survey at the Pittsburg Testing Station, to determine the properties of the common explosives used in mining and to demonstrate the proper method of using them. The charging and tamping of the holes drilled to break down coal or rock in a mine are matters of interest and importance, but too often these operations are poorly done, due to negligence, carelessness, or inexperience. It is impossible to see inside a drill hole that has been charged, to determine if the fuse has broken or kinked, the cap pulled out of the cartridge, or to discover other possible accidents that will occur to every practical man. In order to enable the students in mining engineering at the University of Illinois to practice charging holes and then have the charge inspected and criticized, a charging box or "artificial hole" has been built. This is shown in the accompanying illustration and consists of a block of timber 8 inches square and 4 feet 6 inches long, with a hole 2 inches in diameter bored in the center. This block is sawed through the center, hinged and held firmly by clamps while the hole is being charged. It is then opened and the charge easily inspected. By varnishing and greasing the hole, the tamping is prevented from sticking to the wood when the top is raised.

Apparatus to Demonstrate Loading of Holes

The "hole" is set on wooden horses and by raising one horse a hole inclined upward or downward can be obtained. Students are required to charge the hole, using different kinds of tamping, different explosives, and all of the ways of firing by squib, fuse, and electric detonators. Sand or sawdust is used to represent black powder and dummy cartridges of inexplosive dope to represent the nitroglycerine explosives. The charge is examined and criticized by the class and by the instructor.

An experienced miner and former mine manager who was showing a class of students how to load the hole was very much surprised to find upon opening the box that the fuse was kinked into an S shape and broken.

By the use of this box, the effect of using different kinds of tamping can be easily shown, the rate of burning of fuse in a hole can be tested and other experiments with explosives demonstrated much better than in the darkness of a mine where the inside of the charged hole cannot be seen.

A glass tube was first thought of, but the glass breaks readily, is hard to clean from the tamping for another charge, and moreover, as the inside can be seen during the tamping, the actual conditions are not represented nearly as well as with the block.

The blasting box or "hole," which can be easily made and is inexpensive, is adapted for demonstrating to inexperienced men at a mine the proper method of charging. It can also be used for demonstration purposes before mining institutes, etc.

USE CARE IN THE PURCHASE OF BRATTICE CLOTH

Brattice cloth is a comparatively inexpensive item of mine equipment, and as a rule the quantity, weight, and quality furnished by purchasers who buy from sample is not carefully looked into.

Our attention has recently been called to an instance wherein the purchaser of a quantity of brattice cloth, which was quoted at an abnormally low price, preserved the sample sent him. When his order was filled he compared the cloth received with the sample and found that it was of lighter weight, and on measurement the shipment was found to be of less yardage than was charged.

Naturally, no responsible business house will be guilty of such practices, but all sellers of mine equipment are not responsible. Reputable houses, such as those who advertise in Mines and Minerals, welcome the most rigid inspection of the brattice cloth they furnish. It is a rigid rule with Mines and Minerals to deal only with reputable concerns and the utmost care is used before accepting advertisements. This rule has been so well enforced that during the past 20 years there have been but three advertisements thrown out of our columns on account of methods of business that were not strictly honest. Two of these three were doing a legitimate business when their advertisements were first secured, but the third was purely dishonest from the start. The latter, however, did not fool our management only. The advertisement appeared for months in the columns of several mining journals after Mines and Minerals and one other reputable publication threw the advertisement out, and published an expose of the methods of the parties advertising.

EVOLUTION OF MINE HAULAGE

Written for Mines and Minerals, by E. B. W.

(Continued from July)

Different Systems of Endless and Tail-Rope Haulage. Aerial Tramways. Cableways

From primitive methods of mineral transportation, described in the June issue of Mines and Minerals, to the rope haulage described in this article, some years have intervened. Just the year wire-rope haulage was introduced is not known to the writer but he assumes it was between 1870 and 1880. However, since its introduction it has proved both economical and satisfactory and there are conditions where it cannot be replaced to advantage. Some 30 years ago the North of England Institute of Mining Engineers had an excellent paper on underground haulage and from this the assumption is that it was first introduced in that country.

Fig. 38. Endless-Rope Haulage

The two most generally adopted systems are known as the endless-rope and main-and-tail-rope systems, and in coal mines which have low entries and in coal mines which are fiery, these systems offer advantages that other systems of haulage lack.

The endless-rope system, as its name suggests, uses an endless rope which is kept running continuously in one direction by a pair of drums geared together and set tandem. The drums, which are comparatively narrow, are provided with grooves in which the rope runs in its traveling on and off. Two drums are necessary to supply sufficient frictional grip to drive the rope when the trip is attached, and, further, a tension wheel in the rear of the drums is required to take up slack due to the stretching of the rope. The rope winds around the drums several times, then half around the tension wheel (which is weighted and movable back and forth as occasion demands) and then back to the drums and off. To use this system to the best advantage a double track is essential, also a grade which is fairly uniform and in one direction, although with the top grip illustrated in Fig. 38 this is not absolutely necessary.

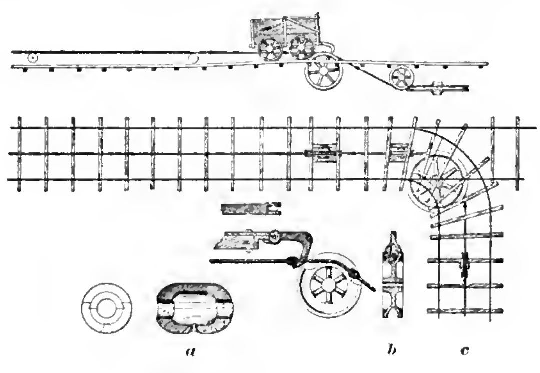


Fig. 39

Where cars are hitched to ropes traveling underneath them, the grips are clamps, which move with the rope. On down grades the clamp holds back the car with a jerk, and on reverse or up grade the car gives another jerk on the rope, thus injuring the rope, particularly where several cars are coupled together. If the trips could be spaced at regular intervals so that the load on the haulage engine could be uniform, this system would be ideal, but owing to the delays in gathering cars and the irregularity in unloading, this is not possible. It is possible to work cross-entries by the endless-rope system as well as main entries if sheave wheels and gearing are arranged for the purpose.

The horsepower required for an endless-rope system may be calculated from the formula

$$\text{H. P.} = \frac{l}{u\,33000}\left[o\left(1+\frac{2\,W_1}{c}\right)+2WV\right]$$

In this formula l = length of haul; V = speed of rope per minute; o = output of mineral per minute; W_1 = weight of empty cars; c = capacity of car in pounds; W = weight of rope in pounds per foot; $u = \frac{1}{40}$ of the total weight to be moved = tension. The horsepower found by the above formula should be doubled to allow for extra pull in case of stoppage.

The Gunckel endless-rope system of haulage consists of an endless-wire rope with hollow, oval, steel knobs a, Fig. 39, attached at intervals. An iron rivet is driven through the rope and knob, after which the hollow is filled with Babbit metal to prevent the ball from slipping. The grip used is shown at b, while at c is shown the means adopted to carry the rope under the track and around a sheave at curves.

One of the difficulties of endless-rope haulage is overcome by placing the rope above the cars. When this system is practiced the cars do not need to be run in trips, and as the cars and rope move slowly there is a minimum of wear and tear. The "grip" for cars being hauled by this overhead system is shown attached to the front of a car in Fig. 40. The empty cars are taken off by simply raising the rope and placing it in a pulley overhead, and as the cars approach this point the rope gradually lifts out of the grab. Where there is any unevenness in the road so that sheaves must be

used to keep the rope from the roof, the plan shown in Fig. 41 was found to overcome the difficulty at the DeBeers Diamond Mine, in South Africa. The two sheave wheels are fixed with hardwood bearings on the ends of two angle irons, which are pivoted on a center shaft of the wooden frame like a balance.

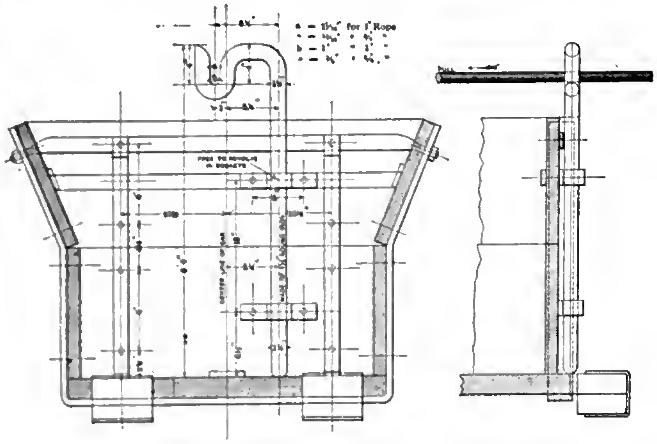


Fig. 40. Grip for Endless Rope

The grip on taking the first wheel lifts it up and depresses the second, thus allowing the car to pass under the first one. When the car passes under the second wheel it raises that and

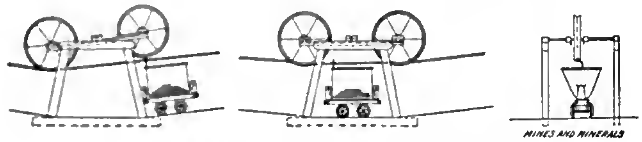


Fig. 41. Arrangement of Sheaves to Keep Rope from Roof

depresses the first wheel. The same arrangement can be used on an incline of 1 in 12 without the rope lifting from the grip.

Much has been said for and against tail-rope haulage, but even then it is preferred, because it may be installed where there are undulating grades, and where for certain reasons the

Fig. 42 Cutting off Cars from Tail-Rope

haulways cannot be continued in a straight line. In tail-rope haulage there are two winding drums on the same power shaft, but so arranged that they revolve in opposite directions. The cars are run in trips and the system may be either single or double tracked, but usually the former. The haulage or main rope is attached to the front end of a loaded trip, while the tail-rope is attached to the rear end of the trip and draws along behind as the loaded cars move to the shaft. As the trip moves either way, one rope plays from its drum while the other is wound on the drum. Fig. 42 shows a car runner cutting off a loaded trip outside the Klondyke coal mine, in Maryland. Whenever an empty trip goes into the mine the tail-rope, which passes half around a pulley at the end of the system, acts as the haulage rope for the empty cars.

On curves, or where it is desirable to carry either main rope or tail-rope across the track, a series of horizontal cast-iron guiding sheaves are used. On curves, a special arrangement such as that shown in Fig. 43 is introduced to carry the main rope. In this particular case the side pull from the main rope is resisted by a series of wooden drums, while the tail-rope is carried on the side near the roof of the mine. Various methods for making attachments to cars and for coupling ropes, when the system is used for side-entry haulage, have been devised. Various kinds of rollers, sheave wheels, and methods of placing them have been tried, so that to go into the details of this matter would be out of the province of this article. Various modifications to the system have been introduced, such as third-rail tail-rope haulage, district tail-rope haulage, etc. The horsepower for the haulage engine of a tail-rope system is found from the formula

$$\text{H. P.} = \frac{f(W_1 N + W_r)\,v}{33000}$$

In which f = the coefficient of friction = tension = $\frac{1}{40}$ the weight of the load to be moved: W_1 = weight of a loaded car in pounds; W_r = total weight of the rope in pounds; v = speed of the rope in feet per minute; and N = number of cars. The rope couplings are attached to the trip which prolongs the life of the rope, in fact it is customary to use old hoisting ropes that have been condemned, as tail-ropes and even for haulage ropes.

One of the most pronounced improvements in mine haulage is that of aerial or wire-rope tramroads, which have made it possible to work mineral deposits which were practically inaccessible. The flexibility of aerial tramways is such that they may be used for carrying supplies to the mine as well as ore from the mine. There are cases where mine cars are carried over aerial tramways and then hauled on surface roads to their destination. Mine timbers, powder, fuel, etc., are also carried from the valley up into the hills, by this means, crossing rivers, valleys, and hills in the ascent. There must be a loading station and an unloading station, and between them a system of towers to support the carrying rope. Aerial tramways are of three kinds, the Bleichert system requiring a stationary and traveling rope, and the one-rope, or Halliday, system, and the cableway. The aerial tramway operated by the Montana Coal and Coke Co. is an excellent illustration of the flexibility of the aerial tramway. For many years the coal was brought from the mines at Aldridge by means of a stream of water confined in a wooden flume lined with sheet iron.* The distance traversed was 1¾ miles with a drop of over 1,000 feet. It was decided to open two new mines and adopt the aerial tramway system of transportation, using three separate lines.

The No. 1 line is 4,000 feet in length and is operated by gravity, conveying the coal from the Newton Mine to the discharge station or coal bins at Electric, 900 feet lower in elevation. See Fig. 44.

The No. 3 line has a total length of 7,200 feet and connects the Foster Mine with the washer at Aldridge, as well as with the terminal of No. 2 line which carries the coal from the loading station, Fig. 45, 7,800 feet, dumping it into coal bins at Electric. The No. 2 line overcomes an elevation of 1,200 feet and traverses a very rough country. The loading terminal consists of a bin of about 100-ton capacity, the bottom of which is so arranged that the coal is delivered in equal quantities to the automatic loader. This loader, shown above the bucket in Fig. 45, is constructed with two pockets of about 12.5 cubic feet

* Mines and Minerals, Vol. 29, p. 531. Robert M. Magraw.

capacity, one on each side of the bucket, each delivering the coal automatically to it without stopping the line or detaching the bucket from the traction cable, although the speed is checked somewhat during the operation. The bucket, after passing around the grip wheel, engages, by means of a short shaft placed horizontally through the trolley frame, two levers which are attached to the loader. This loader is pushed along by the bucket a distance of 12 feet, during which time the gates on the loader are raised automatically, thus discharging into the bucket from both sides and ensuring an even load. The buckets have a capacity of 25 cubic feet, or 1,250 pounds of coal, and are so spaced that when one is loading another is dumping. The usual speed of the buckets is about 250 feet per minute. The No. 2 line crosses a deep gulch, the length of the span being 1,250 feet, the vertical height of the rope at the center being 300 feet. Owing to the length of No. 2 line, 7,800 feet, it was necessary to erect a tension station to break the strain on the track ropes, and to permit of greater ease in taking up the slack resulting from expansion. The two down-hill cables are anchored at this station, the two from the up-hill end being held taunt by weight boxes. Fig. 46 shows the tension station, a high tower and a hill-side tower on No. 2 line.

The tramway described is constructed on the Bleichert system, introduced into this country many years ago by the Trenton Iron Co. Patent locked wire-rope is considered to be the best for this kind of tramway, owing to the comparative smoothness of its exterior, which prevents wear both of rope and trolley wheels. The calculation for the sag in the rope between towers is somewhat complicated, but according to J. H. Janeway, Jr.,† there are but two problems in curves. The first is that of a long curve unloaded from one support on a height to another, maybe 2,000 feet or more away, but subject to a tension coefficient which will be greater than can happen in practice, to ensure the cable always remaining in the saddle. This curve is broken up by interposing intermediate supports making shorter spans, and the use of a span of 500 or 600 feet to avoid several supports. The curve will have practically the form of a parabola, the formula of which is

$$x^2 = \frac{l^4 W^2}{4(T^2 - l^2 W^2)}$$

in which x = deflection at the center or versed sine; l = one-half the span; W = the weight of the rope per foot; T = tension in rope at supports.

This formula is for curves whose supports are on the same level. For curves whose supports are on different levels Rankin's reasoning from Fig. 47 is employed. In the figure, $a\ b\ c$ is the curve of a parabola, with b the lowest point, then s is the span between two points whose coordinates are $x_1\ y_1$ and $x_2\ y_2$. From the equation of the parabola $y^2 = 2\ p\ x$ – in which $2\ p$ = parameter of the curve.

$$y_1^2 = 2\ p\ x_1 \text{ or } y_1 = \sqrt{2\ p}\sqrt{x_1} \qquad (1)$$

and

$$y_2^2 = 2\ p\ x_2 \text{ or } y_2 = \sqrt{2\ p}\sqrt{x_2} \qquad (2)$$

but $s = y_1 + y_2$ in the figure, therefore $= \sqrt{2\ p}\,(\sqrt{x_1} + \sqrt{x_2})$ or by dividing, $\sqrt{2\ p} = \frac{s}{\sqrt{x_1} + \sqrt{x_2}}$. Substituting for $\sqrt{2\ p}$ this value in equations 1 and 2

$$y_1 = s\frac{\sqrt{x_1}}{\sqrt{x_1} + \sqrt{x_2}} \text{ and } y_2 = s\frac{\sqrt{x_2}}{\sqrt{x_1} + \sqrt{x_2}}$$

Making W and T functions of each other and making the proper substitution in the formula, x or the center deflection is determined and a curve plotted from the well-known property of the parabola

$$x : x^1 = y^2 : (y^1)^2$$

† MINES AND MINERALS, Vol. 24, April, 1904, p. 423.

The supports are now located and their heights scaled and marked.

The second problem in curves is that of a long or short single span with one or more carriers on the cable. The weights of the carriers are considered as uniformly distributed loads and

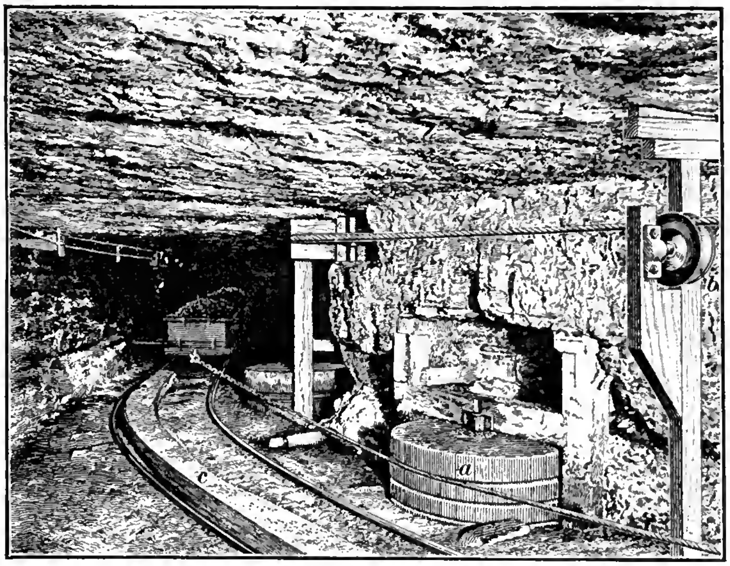


FIG. 43. HAULAGE AROUND CURVES

the solution is as before, bearing in mind that the tension for reasons stated must be less than the actual.

An approximate formula for a single center load is

$$T = 2\,\frac{l + y\ W}{\left(\sqrt{\frac{f^2}{1 + 2\ f^2}}\right)}$$

in which T is the tension of the rope at supports; l = concentrated load; f = ratio between x and y; and W = weight of the rope per foot.*

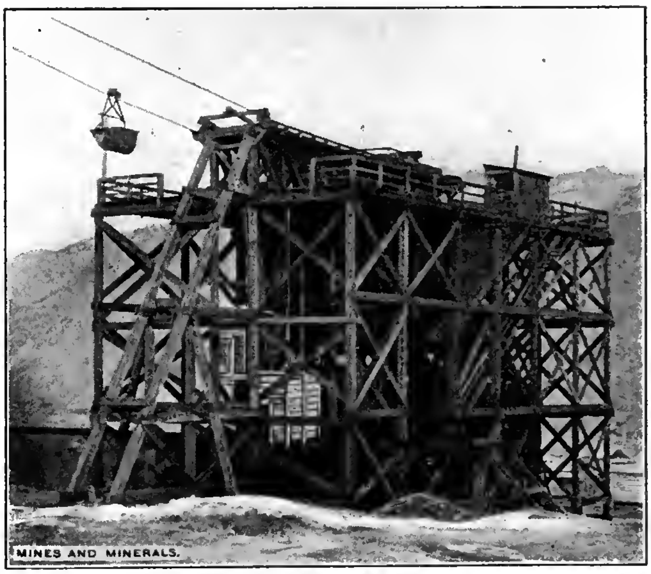


FIG. 44. LOWER DISCHARGING STATION, AERIAL TRAMWAY

The Halliday wire-rope tramway consists of one movable rope from which the buckets are suspended by goose-neck hangers. If the grades are less than 1 to 4, the box-head hangers, Fig. 48, are used, friction being sufficient to prevent slipping;

* The derivation of this formula is given in MINES AND MINERALS, Vol. 24, p. 425.

if, however, the grades are as much as 1 to 4, clamps must be used to prevent slipping. The clamp hanger is shown in Fig. 49. It has a shank *a*, strap *b*, key *c*, and bolt and nut *d*, in order to surround and take firm grip on the rope. The rope and grip travel over the tower sheave wheels, as shown in Fig. 50, which is a tower placed on an eminence. If the grade is 8 degrees or more, the system, being on the endless-rope plan, will move by gravity and necessitate grip wheels supplied with brakes at the terminals. If, however, the grade is undulating, or less than 8 degrees, this system, like the Bleichert, must be driven by power. The cost of wire-rope tramways is practically the same as ordinary surface tramways where the ground is not exceedingly rough. The cost of running a wire-rope tramway depends upon the conditions which it must surmount, and the tonnage carried daily.

An article on rope haulage would be incomplete without some reference being made to the Lidgerwood cableway. This arrangement consists of a wire-rope runway, a carriage, a car, a hoisting rope, and haulage rope. The car is loaded, hoisted to the carrying rope, where it joins the carriage, and from this point on the former fall rope becomes the haulage rope. The first installation of this system was introduced at the Tilly Foster iron mine, near Brewsters, N. Y., where 300,000 cubic yards of rock overburden was removed in order to reach the body of magnetite beneath. The appearance of the mine after the ore was uncovered is shown in Fig. 51.

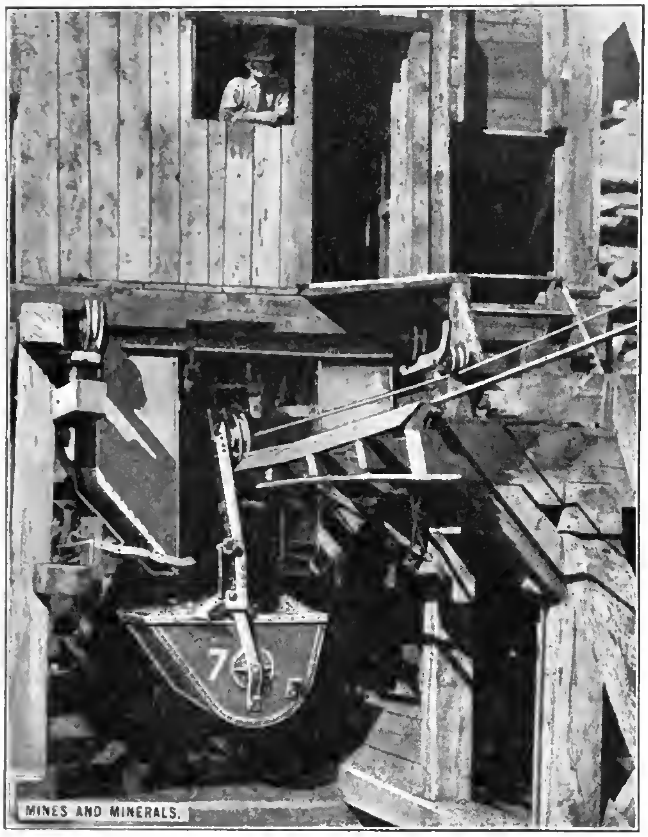


Fig. 45. Loading Station, Aerial Tramway

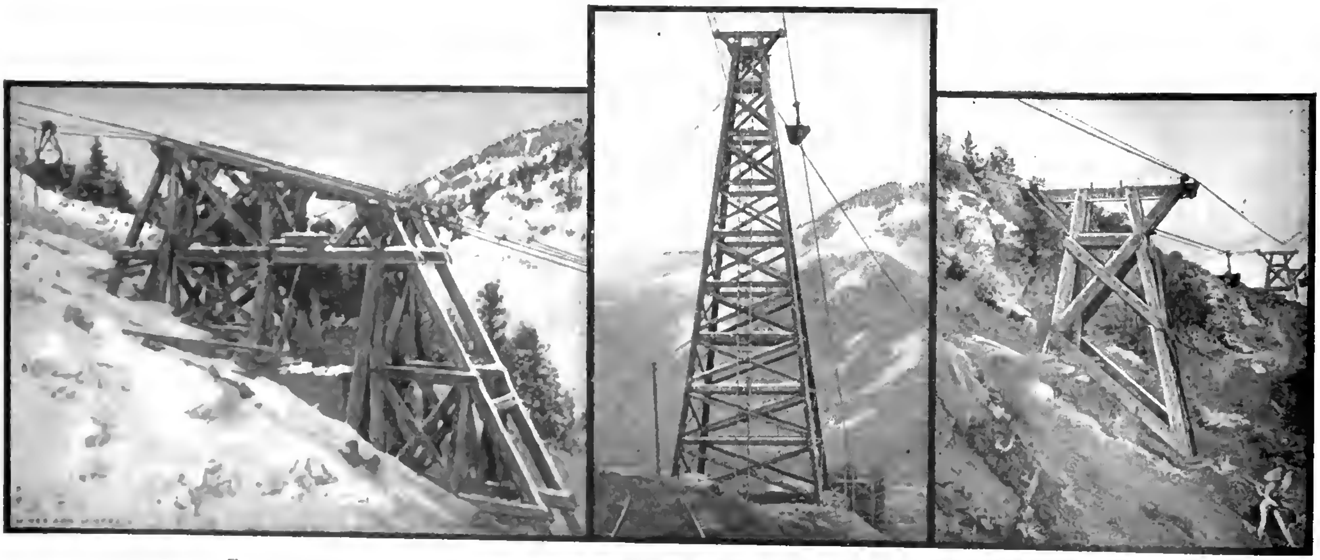
The cableway was invented by Pluchet in 1851, and 11 years later it appeared in the slate quarries in Pennsylvania,* but not with the present improvements in carriage, fall block, head towers, etc. At the Tilly Foster Mine about 600,000 tons of ore were recovered; also by this apparatus dams are constructed, ships unloaded, material transferred across rivers, placer mines worked, coal and iron ore beds stripped; in fact it may be applied to a very large number of useful purposes with economy, particularly in quarrying. The longest cableway constructed on this system had a span of 2,140 feet. It was used in the construction of a dam at Glens Falls, N. Y., and was capable of handling a load of 6 tons. Its value is increased in quarrying by its hoisting up an incline as well as carrying, which feature, in some cases, is almost a necessity.

(*To be continued*)

COAL NOTES

Coal lands are being staked between Mattagami and Missinaibi rivers, near Hudson Bay. The *Cobalt Nugget* says: "To any one but a lumber jack or an Indian it's worth all the coal concessions in the country to wallow in the muskeg around Hudson Bay and be eaten up day and night by flies. There's a fortune waiting for the man who can invent the dope that will stop a mosquito."

We would suggest the plan adopted in Cuba, for killing fleas; viz., first catch them and next put camphor on them. After they are released and attack the next person their bite is harmless.



Tension Station *High Tower* *Hillside Tower*

Fig. 46 Wire-Rope Tramway

*Spencer Miller, Mines and Minerals, Vol. 24, p. 411.

The production of both iron, and consequently coke, has increased over the boom year of 1906 about 10 per cent. The Connellsville coke region produces about 400,000 tons of coke weekly. Recent long-term coke contracts were consummated at prices around $2 per ton. When this price is compared with the recent price paid for Connellsville coal land; viz., $3,300 per acre, it seems ridiculous. The price is the highest ever paid, and figured out, makes each ton of coal in the ground worth about 30 cents, and the coal for a ton of coke worth about 50 cents.

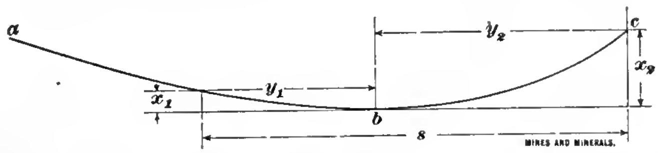


Fig. 47

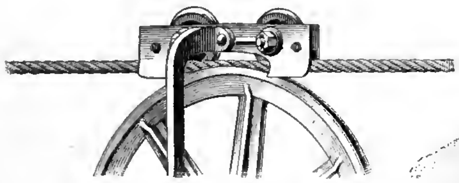


Fig. 48. Box Head for Hanger

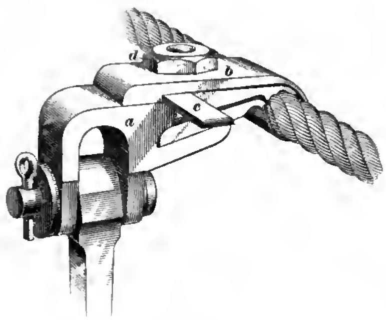


Fig. 49. Grip

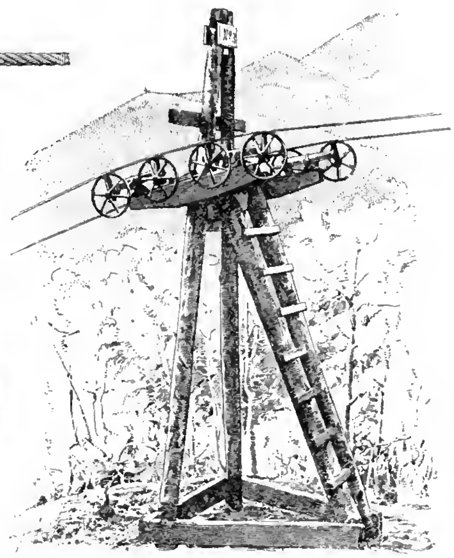


Fig. 50. Tramway Tower

The coal and coke shipments over the Norfolk and Western for the first six months in 1910 were 2.5 million tons more than in the first six months of 1909.

Pottsville, Penna., capitalists who control the Majestic collieries at Majestic, Ky., just across Tug River from War, W. Va., on the Norfolk & Western, according to well-authenticated advices are preparing to spend $600,000 upon their land and operation. A modern steel tipple is now under construction, and other improvements are to be added at once. When these are completed the company will have a daily output of 40 cars, which is among the largest in the lower field. This company controls a coal area of about 10,000 acres on which development has already disclosed several very fine beds measuring from 5 to 8 feet in thickness. It is understood to be the policy of the management to push the work as rapidly as possible, and increase the initial output of the property.

During the first five months of 1910, the Pennsylvania Railroad shipped 28,050,212 tons of coal and coke over the lines east of Pittsburg, which is an increase of 3,868,052 tons compared with the same period in 1909.

A collier entered a book shop, somewhere in England, with his wife and asked: "Can you give me an arithmetic for use in the mine?" The clerk was sorry he could not, and the miner said: "Do you ken what I mean. It is a thing for calculating; something to check the foreman." The clerk again told him he had not got such a thing; and then the collier, looking at his wife in a sheepish sort of way, said: "Man, you have got it in the window." The clerk asked him to point it out—and the collier, taking him into the street, pointed out, "Pitman's Arithmetic."

Drilling for oil in Nescopeck Valley, near Hazleton, Pa., is being prosecuted. "It is said that several bottles of oil have been taken from springs in the neighborhood of the place now being drilled."—*Hazleton Sentinel.* Parenthetically, oil has not been found in the Blue Ridge in paying quantities and it is not likely that it will be, owing to the metamorphic changes in rocks and the dynamic disturbances that occurred in times past which prevented its accumulation.

The trouble between the coal operators and miners over the scale in the Pittsburg district seems to be nearing an end. Motormen are to receive an increase of 10 cents, making their wages $2.80. Wiremen are similarly increased making their wages $2.70. Day laborers also receive an increase. A complete check-off system is established including the collection of an initiation fee. An increased pay of 5 cents per inch for thick slate is allowed. There is to be no charge for safety lamps; three-fourths entry prices for four-run places; no increase in the price of fuel for the home; and no increase in rent for miners' houses.

According to the *National Labor Tribune*, Chairman M. H. Taylor has been placed in an embarrassing position by the quarrels between the miners over their labor union officers, which made it appear as if he was assisting one miners' faction in preference to another. When the quarrel was cleared up, Mr. Taylor found it comparatively easy to treat with the miners' officers. The settlement of the local scale is said to be satisfactory to both miners and operators.

The miners working for the Pennsylvania Coal Co. not being connected with the United Mine Workers, could not be controlled when they decided to strike. The coal company and

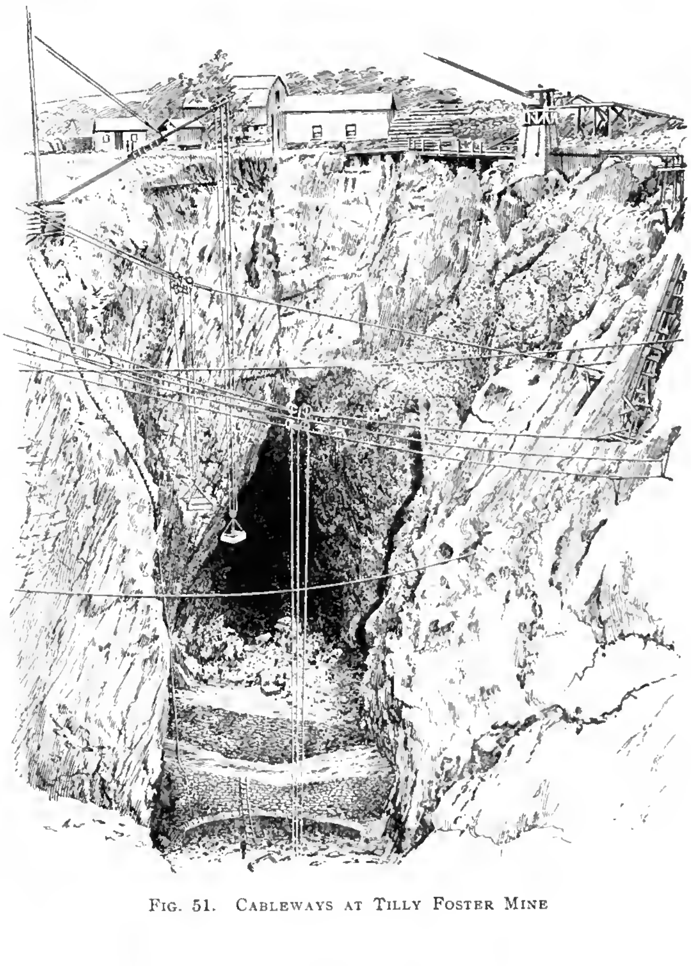


Fig. 51. Cableways at Tilly Foster Mine

unions were in the position of not being able to comply with the award of the Anthracite Coal Commission. After much entreaty on the part of the officers of the United Mine Workers and citizens, the men agreed to place their grievances before the Conciliation Board, and to work pending the decision. The Board was able to obtain concessions at once from the coal company which are satisfactory to the men.

The fiscal year of the Mining Department of West Virginia closed June 30 without one life being lost by an explosion of any kind. Heretofore the mine explosions from gas and dust have attracted the attention of the world. Mines that were thought to be model ones in every particular were visited with the greatest accidents, and the toll of human life was appalling. There were the usual number of deaths from fall of coal and slate. Greater care is being exercised in the mining department, and a higher standard of efficiency is being required among the men that have direct charge in the operation of the mines. Now, all of the mine foremen and fire bosses have to be examined in order to be eligible to hold their positions. The examination is a searching one along practical lines, and any one failing in the test cannot become a fire boss or a mine foreman.

Coal-mine fatalities in the United States in 1909 were fewer than in 1908, notwithstanding an increase of approximately 10 per cent. in the quantity of coal mined. The figures compiled by Edward W. Parker, statistician in charge, Division of Mineral Resources, United States Geological Survey, show the total number of deaths from coal-mine accidents in 1909 to have been 2,412, against 2,450 in the preceding year.

During the last five years the annual reports of the Geological Survey on the production of coal have contained a chapter on coal-mining accidents, their causes, and their relations to the number of men employed and the tonnage produced. These statistics are compiled almost entirely from statements furnished by State Mine Inspectors. It is expected that statistics of mine accidents in future years will be compiled by the new Bureau of Mines. The most serious catastrophe which occurred during the calendar year 1909, was the fire at the Cherry Mine of the St. Paul Coal Co., at Cherry, Bureau County, Ill. It has been reported that in that disaster 393 men were burned or suffocated. The statistics of the holocaust are not, however, included in the preceding general statements, for the reason that the reports of the State Mine Inspectors of Illinois are made for the fiscal year ended June 30, and as the Cherry disaster occurred in November it will be included in the statistics for the fiscal year ended June 30, 1910.

The most serious single accident included in the statistics presented in this statement was an explosion at the Lick Branch colliery of the Pocahontas Consolidated Collieries Co., near Bluefield, W. Va., on January 12, 1909. In that explosion 65 men were killed and 1 was injured. Another explosion had occurred in this mine about 2 weeks before, in which 51 lives were lost. The Lick Branch explosion of January 12 was the only one of serious proportions in West Virginia during the year.

An explosion in mine A of the Chicago and Cartersville Coal Co., at Herrin, Ill., on December 28, 1909, killed 8 men and imperiled the lives of 400 others.

Twelve men were killed by what is supposed to have been an explosion of dynamite in one of the mines of the Cambria Steel Co., near Johnstown, Pa., October 31, 1909.

On April 9, an explosion of dynamite killed 7 miners and injured several others at mine No. 37 of the Berwind-White Coal Co., near Windber, Pa.

On January 25 4 men were killed and 8 injured in Washington mine No. 5 of the Piedmont and Georges Creek Coal Co., in Maryland, as the result of a collision between two trips of coal cars.

On March 19, 5 men were killed and 20 injured in an explosion at the Sunnyside Mine of the Sunnyside Coal and Coke Co., in Vanderburg County, Ind. Twenty nine men were in the mine at the time of the explosion, and all but 5 escaped.

An explosion which caused the death of 20 men occurred on January 10, in the mine of the Zeigler Coal Co., at Zeigler, Franklin County, Ill.

What is supposed to have been a dust explosion occurred in April at the Short Creek Mine of the Birmingham Coal and Iron Co., in Jefferson County, Ala., and killed 18 men.

PETROLEUM DEPOSITS ABOUT TAMPICO, MEX.

Most of the capital employed in the petroleum and asphalt industry near Tampico, Mexico, is American, and the company most heavily interested, and in fact the pioneer in this field, is the Mexican Petroleum Co., an American concern, with headquarters at El Ebano. This company now has over 30 producing wells, a large number of storage tanks all of its departments specially equipped with modern facilities for handling the output, and it is now building a pipe line 62 miles to run from its new fields in the state of Vera Cruz to Tampico. The handling of the most of its output does not require a pipe line, as the wells are conveniently located to rail or water transportation. The output is consumed by the Waters-Pierce refinery, of Tampico, and by the National Railroad of Mexico, from 50 to 55 cents gold being paid for the crude product. Other prominent American operators in this section are the Harriman interests, the Mexican Fuel Oil Co., the Mexican Fuel Co., C. H. Rathbone, and the McKay interests, a collateral branch of the Rio Bravo Oil Co. S. Pearson & Sons (English) are also operating near Tuxpam and building a 6-inch pipe line to run from their fields to Tuxpam. The well belonging to the latter and the Pennsylvania Oil Co., which was brought in on July 4, 1909, at Dos Bocas, about halfway between Tampico and Tuxpam, and which later was completely destroyed by fire, has been pronounced by several experts to have been the greatest well that has ever been known in the history of the oil industry of the world, its output having been estimated from 300,000 to 500,000 barrels daily.

The wells in the vicinity of Tampico range in depth from 800 to 2,500 feet and vary in production from 100 to 2,000 barrels daily. The oil is heavy and viscous and is mostly used in its crude state as a fuel. Under distillation, about 30 per cent. fractionates into good grades of naphthas and illuminants. The heavy or malthitic product is converted into asphaltum by the removal of a 10 per cent. distillate. These asphaltums, for which there is a good demand, delivered in Mexico City, sell for $40 gold per ton. Their exportation, mostly to Germany, is also increasing.—*United States Consular Report.*

PERSONALS

Edgar G. Tuttle, E. M., consulting engineer, is now permanently located at 30 East Logan Street, Germantown, Pa.

L. E. Yoder, chief engineer and superintendent of The New River Collieries Co., Eccles, W. Va., has resigned his position and will resume duties as chief engineer of The New River Co., with offices at Harvey, W. Va.

J. B. Tyrrell has been elected a fellow of the Royal Society of Canada.

J. H. Dietz, formerly mining engineer for the Central Coal and Coke Co., of Kansas City, and for the past 7 years consulting engineer of the United Iron Works, Joplin, Mo., has recently assumed the position of general manager with the Eagle Foundry and Machine Co., of Fort Scott, Kans.

Ernest Russell Woakes left for the Caucasus on July 17 to examine mines on behalf of Messrs. Hooper, Speak, and Fielding.

J. P. Hutchins writes from St. Petersburg, where he is on mining business, "that concomitants essential to doing business in Russia consist in drinking much tea and smoking many cigarettes."

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

Mine Inspectors' Examination Held at Scranton, Pennsylvania, May 25, 1910

QUES. 1.—(*a*) What gases are found in anthracite coal mines? (*b*) How do they affect the lives and safety of men employed in the mines? (*c*) Give the symbols, specific gravity, and properties of each gas. (*d*) Where would you look for each; how is the gas produced; does it affect combustion, and how?

ANS.—These questions can be answered together more concisely by stating the properties and effects of each gas in order. Methane or marsh gas (CH_4), specific gravity .559, is a colorless, odorless, and tasteless gas; combustible, but will not support combustion or life; it suffocates when breathed pure, and extinguishes flame; when mixed with air it can be breathed with impunity but the mixture, in certain proportions, is inflammable or explosive. Marsh gas is to be sought at the roof, or the face of rise headings, or in void places or close to large feeders. The gas issues from the coal or other strata, having been produced in the formation of the coal. Carbon monoxide (CO), specific gravity .967, is a colorless, odorless, and tasteless gas; combustible, supports combustion, extremely poisonous, forms an inflammable or explosive mixture, in certain proportions, with air. This gas is to be sought in poorly ventilated workings in proximity to and in old abandoned places; it is produced by gob and mine fires, or by combustion of carbon in a limited supply of air, or by the explosion of powder in blasting. Carbon dioxide (CO_2), specific gravity 1.529, is a colorless, odorless gas, with slight acid taste; it is not combustible, and will not support combustion or life; it suffocates when breathed, dims and extinguishes flame, and reduces the explosiveness of firedamp mixtures. This gas is to be sought near the floor and in low or dip workings, and in poorly ventilated or abandoned workings; it is produced by the burning of lamps, breathing of men and animals, or any combustion of carbon in a plentiful supply of air. Hydrogen sulphide (H_2S), specific gravity 1.1912 is a colorless gas, having a bad odor like that of rotten eggs. It is combustible, but does not support combustion or life; extremely poisonous, and mixed with air is inflammable or explosive. It is to be sought in damp workings where the coal contains sulphur balls or pyrites; it is produced by the decomposition of pyrites in presence of moisture.

QUES. 2.—State briefly but thoroughly.(*a*) What obligations are imposed on the operator by the mine law? (*b*) What on the mine inspector, and what are his duties? (*c*) What on mine officials? (*d*) What on all employes? State the act, date, article and section, in all cases where memory serves you.

ANS.—The mine operator is requred by law (Art. 3) to furnish an accurate map of the mine as surveyed, and extended at least once in every six months, and to keep one copy at the colliery and file one copy with the mine inspector; also (Sec. 10), to leave a barrier pillar of sufficient width to render the mine safe from inundation by water accumulated in adjoining mines; also (Art. 4), to provide two openings not less than 150 feet apart at the surface, and 60 feet apart under ground, available for each seam in which more than 20 persons are employed; also (Art. 6), to provide suitable wash house at request of 20 employes (Art. 7), a colliery ambulance and two stretchers (Act 1901), waterproof and woolen blankets, olive and linseed oil, bandages, etc., in a suitably clean, lighted, and ventilated room in the mine, (Art. 10) adequate ventilation in the mine, (Art. 11) all necessary timber, ties, rails, etc.; also (Art. 12), to use every precaution to ensure the safety of workmen and employ a competent mine foreman to take charge of the mine. (*b*) The mine inspector (Art. 2, Sec. 11) must enter and examine each mine in his district, and carefully inspect all the workings and the machinery and other appliances in use, methods of ventilation, lighting, and drainage, and inquire into matters relating to the health and safety of the workmen; (Art. 3, Sec. 5) accept, or reject as inaccurate, all mine maps sent him; (Art. 8, Sec. 3) act as member of examining board to examine and certify men for the positions of mine foreman and fire boss; (Art. 13) attend inquests, and investigate all mine accidents, collect and record evidence respecting cause of same; (Art. 15) apply for injunction, at the court of the district, to stop the operation of any mine that does not comply with the mine law, after giving the operator of the mine 24 hours' written notice of his intention; (Act 1901, Sec. 4) visit the medical room of each mine in his district to see that they comply with the law. (*c*) The mine foreman (Art. 12, Rule 3) is charged with all matters pertaining to ventilation; (Art. 10, Sec. 15) the weekly measurement and recording of the quantities of air in circulation; (Art. 11) the supply of all timber, ties, rails, etc., to the men as needed; (Art. 12, Rule 4) the weekly examination of all accessible parts of abandoned places in the mine; the examination, by himself or assistants, of all working places, (Rule 5) daily, before each shift, in mines generating gas, and (Rule 12) each alternate day while the men are or should be at work; (Rule 13) the daily examination of all slopes, shafts, roadways, signal apparatus, pulleys, timbering, etc.; and (Rule 8) he must use every precaution to ensure the safety of the men. (*d*) (Art. 12, Rule 14) Every miner is made responsible for keeping his place securely timbered. (Rule 24) Any employe must report to the mine foreman or other person in charge any danger he may discover in any part of the mine.

QUES. 3.—The diameter of the cylinder of an engine is 16 inches, length of stroke 30 inches, indicated steam pressure 100 pounds, piston speed 225 feet per minute; what is the horsepower of the engine?

ANS.—Assuming the given steam pressure is the mean effective pressure as calculated from an indicator card, the indicated horsepower of the engine at this speed is

$$H = \frac{100(.7854 \times 16^2)225}{33{,}000} = 137 + \text{H. P.}$$

QUES. 4.—What recommendations would you make to secure a large quantity of air in a mine, with the least possible resistance; and how would you handle the air to secure the best possible results?

ANS.—Reduce the resistance by removing every obstruction to the flow of the air-current, and enlarging all cross-cuts, and shortening the distance the air must travel wherever practicable. Clean up the air-courses and avoid standing empty and loaded cars in the airways, where they seriously obstruct the free movement of the air. Also, split the air wherever the conditions will justify doing this, thereby reducing the velocity of the current and the mine resistance.

QUES. 5.—Consider an airway 8 ft.$\times$10 ft. and 5,000 feet long; (*a*) what must be the length of a (similar) airway 6 ft. $\times$8 ft. in section in order that it will have the same rubbing surface? (*b*) If a pressure of 5 pounds per square foot is required to create a certain velocity in the 8'$\times$10' airway, what pressure per square foot will be necessary to produce the same velocity in the 6'$\times$8' airway?

ANS.—The given airways are not *similar*, as the question states; because their corresponding sides are not proportional. But taking the sizes as given, (*a*) the rubbing of the first airway is $2(8+10)5{,}000 = 180{,}000$ square feet; and since the perimeter of the second airway is $2(6+8) = 28$ feet, the required length of this airway, for the same rubbing surface would be $180{,}000 \div 28 = 6{,}428.57+$ feet. (*b*) For equal rubbing surface and equal velocity, the pressure varies inversely as the area; or the pressure ratio is equal to the inverse area ratio. Hence, calling the required pressure x,

$$\frac{x}{5} = \frac{8 \times 10}{6 \times 8} = \frac{5}{3}; \text{ and } x = \frac{5 \times 5}{3} = 8\tfrac{1}{3} \text{ lb. per sq. ft.}$$

If, however, the airways are assumed as of equal length, for

equal velocities the pressure varies as the perimeter and inversely as the area; or, the pressure ratio is equal to the product of the perimeter ratio and the inverse area ratio; thus,

$$\frac{x}{5}=\frac{2(6+8)}{2(8+10)}\times\frac{8\times 10}{6\times 8}=\frac{7}{9}\times\frac{5}{3}=\frac{35}{27}$$

$$x=\frac{5\times 35}{27}=6.48 \text{ lb. per sq. ft.}$$

Ques. 6.—State what measures you would adopt to prevent the occurrence of mine fires from any cause, condition, or circumstances. Should a serious fire occur in the mine, in your efforts to extinguish it, what general points would you observe for the protection of life, limb, and property?

Ans.—Permit no combustible material such as hay, oil, powder, etc. to be taken into the mine except what is required for immediate use, and permit no such material to be placed where it will be exposed to the open lights of workmen. Give strict orders that all supplies of hay, oil, powder, etc. must be taken at once to their destination on arrival in the mine. All powder must be contained in secure metallic cases. Shaft and slope bottoms must be lighted by stationary lamps or electric bulbs; no open lights should be allowed either here or in the mine stables. Allow no accumulations of oily rags or waste in the pump room or lamp station. Where electric power is used particular attention must be given to the arrangement of the switchboard and to the wiring. Enforce strict regulations in regard to the handling of powder in preparing a blast. In blasting in a gassy mine arrangements should be made to have the face examined after the firing of a blast to ascertain that no gas has been fired. Should a serious fire occur in the mine, the men must be notified to withdraw by some safe route if possible. At the same time active measures must be taken to prevent the smoke and gases of the fire being carried into the workings, and get water on the fire by a line of hose laid in the entry; or bring the chemical engine into action. The fire must, in general, be fought from the side toward the air, so as to avoid exposing the men to the poisonous gases and smoke. By what means the smoke and gases may be kept from entering the workings, whether by short circuiting the air, or slowing down, stopping, or reversing the fan, will depend wholly on the conditions existing in the mine, location of the fire, and the headway it has gained.

Ques. 7.—If you were assigned a colliery of 400 acres with five veins, the third and fourth being mined, and the fifth or bottom vein just being opened, and you were about to mine pillars; state briefly what restrictions you would impose for the safeguarding of life, limb, and property, there being no unusual conditions.

Ans.—Assuming that the workings in the third and fourth veins have been systematically planned so that the pillars in the lower seam stand vertically under those in the seam above, the work of drawing back pillars in these two veins should begin in the third vein where the entries and chambers have reached their limit. If the two veins are separated by only a few feet of rock the pillar work in the upper vein should be kept a few yards in advance of that below and this work should proceed uniformly in each vein. Only the most experienced miners should be employed in drawing back the pillars, and they should keep a careful watch for gas, loose top, fault lines in the roof and other dangers incident to the work. As far as practicable the condition of the abandoned workings in the overlying veins should be ascertained and special attention should be given to timbering so that the miners will be warned of any approaching fall of roof in time to escape.

Ques. 8.—(a) What kind of explosives are used in the anthracite mines? (b) What powders are best adapted to various conditions of mining? (c) How should they be handled, stored, etc.? (d) What is meant by permissible explosives?

Ans.—(a) Black powder and dynamite of different grades. (b) Black powder is best adapted to shooting coal, because its action is not as sudden as that of detonating powders (dynamite), and the coal is not as badly broken. The grade or strength of the powder should be adapted to the hardness of the coal. Dynamite or some special kind of detonating powder is used in breaking rock. (c) All explosives should be handled with the greatest of care, and kept in a suitable case or box in a safe place in the mine. Powder must be stored on the surface, in a building set apart from other buildings, and designed for the purpose. No person may have more than one keg (25 pounds) of powder, or what is required for the day's work, in one place in the mine. No lamps or other fire may be brought nearer than 5 feet to any open box or can containing powder. (d) By permissible explosives is meant such explosives as have passed the government tests and been entered on the list of so-called permissible explosives published by the government.

Ques. 9.—How many and what are the essential features of a good safety lamp?

Ans.—The essential features of a good safety lamp may be classified under five different heads; namely, (1) *Simplicity of construction* to facilitate cleaning and avoid mistakes in assembling the parts, and not invite the curiosity of the user or require special oil or special equipment in the lamp room or special intelligence on the part of the user. (2) *Security of the lamp* against damage by accident, improper handling, strong air-currents, sudden exposure to gas, tampering and curiosity. (3) *Good illumination* to assist work and inspection of mine, roof, timbering, etc., and constancy of flame. (4) *Correct indication* of gas and dust, or explosive condition of air by maintaining the same condition within the combustion chamber as exists outside the lamp, by the free admission of air at or below the flame, thus producing an upward circulation, which avoids the tendency of the lamp to smoke, and prevents the products of combustion descending to the flame. (5) *Lightness and portability* of the lamp.

Ques. 10.—Assume that a large territory of a certain mine has collapsed; men are entombed; an inflow of 18,000 gallons of water per hour seriously handicaps the work of rescue, and must be pumped out by a shaft 225 feet deep, with a steam pressure of 50 pounds available at the foot of the shaft. Assuming a piston speed of 110 feet per minute, find the horsepower, size and proportion of a direct-acting steam pump required for this work.

Ans.—To allow for breakdowns and repairs, or a possible increase of demand on the pump, assume all the pumping is done in 20 hours out of 24, or in $\frac{5}{6}$ of the time, making the required discharge in this case $\frac{18,000\times 24}{20\times 60}=360$ gallons per minute. This would require a column pipe having a diameter of $.25\sqrt{360}=4.75$, say 5 inches. Allowing 25 feet over the depth of the shaft to cover the suction and discharge ends, the total length of pipe is $225+25=250$ feet. The total head under which the pump must act is the depth of the shaft to the coal, plus the suction head or depth to low-water line in sump, plus the friction head, which is the extra head required to overcome the resistance of the pipe. The friction head in this case is

$$h_f=\frac{f l G^2}{8 d^5}=\frac{.01\times 250\times 360^2}{8\times 5^5}=\text{say } 13 \text{ ft.}$$

Then, assuming a suction head of say 17 feet, the total head against which the pump must operate is $225+17+13=255$ feet. For this head and an available steam pressure at the pump of 50 pounds, the ratio of the diameter of the steam end to that of the water end is

$$\frac{D}{d}=.7\sqrt{\frac{h}{p}}=.7\sqrt{\frac{255}{50}}=1.58;$$

or

$$D=1.58d$$

This allows for an efficiency of 75 per cent. at the steam end, and .85 per cent. at the water end of the pump, which is good practice. The diameter of the water end allowing this efficiency is

$$d=5.37\sqrt{\frac{G}{S}}=5.37\sqrt{\frac{360}{110}}=9.7, \text{ say } 10 \text{ in.}$$

The diameter of the steam end is then

$$D = 1.58 \times 10 = 15.8, \text{ say } 16 \text{ in.}$$

The horsepower of the pump when doing this work would be

$$H = .00034\ Gh = .00034 \times 360 \times 255 = 31.2 \text{ H. P.}$$

Assuming a 24-inch stroke the pump is making $110 \times 12 \div 24 = 55$ strokes per minute.

Ques. 11.—If you were to make an outside inspection of an average anthracite colliery, state what points would receive your careful attention, and in what features would you be particularly interested? Assume the breaker is over the shaft.

Ans.—It would be important to ascertain the location, character, and equipment of the second or escape opening, and its relation to the hoisting shaft and the breaker building as well as all the other buildings and supply yards that form the surface plant. Also, the methods of lighting, heating, and ventilating the breaker, and the power employed in operating the machinery therein should be carefully investigated. All safety and other appliances should be thoroughly inspected and approved or condemned; this includes brakes on winding drums, safety blocks and catches, cage guides, wings or keeps, gates at top of shaft, signal apparatus, speaking tubes, hoisting ropes and sheaves, hand rails, danger signals, electric wires. The boiler plant should be inspected and the date of the last boiler inspection ascertained; the mine map should also be examined and the data of its latest extensions observed. The age of breaker boys should be noted to see that none are illegally employed.

Ques. 12.—(*a*) If you had reason to question the strength of a boiler to carry, say 100 pounds pressure with safety, in what manner would you satisfy yourself? (*b*) Would you recommend water glasses or gauge cocks on a boiler, and why? (*c*) In what manner could you determine that the steam gauge was recording the pressure correctly? Give a formula by which to calculate the position of the ball of a safety valve so that the boiler will blow off at a given pressure.

Ans.—(*a*) The pressure a cylindrical boiler will stand with safety when the same is in good condition and properly set may be calculated when the thickness and strength of material and the diameter of the boiler are known, by the formula $p = 2$ (thickness$\times$safe strength)$\div$diameter in inches. Thus, a 5-foot steel boiler made of plates $\frac{3}{8}$-inch thick and having a safe strength of 10,000 pounds per square inch, if all right, should withstand safely a pressure of $p = 2(\frac{3}{8} \times 10{,}000) \div 5 \times 12 = 125$ pounds per square inch. However, where any doubt exists as to the strength of a boiler the mine inspector should order its use discontinued until examined by a competent boiler inspector. (*b*) Gauge cocks are preferable to water glasses because of the liability of the latter being broken or becoming clogged, and giving a false indication of the water level in the boiler. (*c*) Multiply the weight (w) of valve, valve stem, and lever by the distance (d_1), in inches, from their center of gravity to the fulcrum of the lever; in like manner, multiply the weight (W) of the ball by the distance (d), in inches, from its point of support to the fulcrum; add these two products together and divide their sum by the distance (d_2), in inches, from the valve stem to the fulcrum. The quotient obtained will be the total upward pressure of the steam on the valve. Finally, divide this total pressure by the area (A) of the valve, in square inches, and the quotient will be the pressure in pounds per square inch at which the valve should blow off. If everything is working correctly the gauge will indicate about this pressure when the valve is blowing off. Expressed as a formula this rule is

$$p = \frac{wd_1 + Wd}{d_2 A}$$

Ques. 13.—If you were conducting a search party through very old and extensive workings and suddenly found yourself bewildered when about to return to the shaft, describe how you would proceed to ascertain the right way back. Assume you have with you the mine map, compass, tape, protractor, and scale, and have found and identified one station at the point where you became bewildered.

Ans.—Set up and level the compass at the point found, and having set off correctly the declination of the needle turn the compass sights in the line of the true meridian. Now spread the map on the floor so that the meridian of the map corresponds with that determined by the compass. In this position the map shows correctly the lay of the workings. Having identified the point found in the mine as a station on the map proceed to measure the bearing and length of any course that is most convenient to travel, leading in the general direction outby or toward the shaft, and establish another point in the mine. With the protractor and scale draw this course on the map, starting from the station found, and mark on the map the point established as substation 1. Having set up and leveled the compass now at substation 1, sight back to the last preceding point and read the needle to detect any possible error due to local attraction. Continue to survey and plat on the map each course leading outby till some place is arrived at that can be recognized.

Ques. 14.—Describe the dangers that attend the use of electricity in and about mines and state the suggestion you would make to safeguard the workmen.

Ans.—The chief danger lies in the liability of workmen or mules to come in contact with live wires or some tool or machine or other metal that is charged with the electric current, and receive therefrom a fatal shock. This danger is greatly increased in the dark and uncertain passageways of mine workings, and the occurrence of roof falls and settlements that often displace the conducting wires and render roads and passages dangerous. There is also danger from short circuiting of currents, blowing out of fuses and sparking of commutators and electric connections by which gas may be ignited or a fire started. To safeguard life use a 250-volt direct current for haulage and permit no travel on the haulways; place conspicious warnings at all road crossings and elevate the trolley wire, at these points out of harm's way; also place danger signals at the mine entrance and the inside parting where trips are made up. Insulate all high-tension transmission lines for conducting power units into the workings and protect these as well as possible against accident.

Ques. 15.—What does the anthracite mine law require in case it is found impracticable to keep the mine free from an accumulation of gas or water?

Ans.—Article 12, Rule 8, of the laws require the person who is in charge of the mine at the time such danger prevails, to use every precaution to ensure the safety of the workmen, and to withdraw every man from the mine, or such part thereof as is found to be dangerous, except those required to remove the danger; and not permit them to return to the mine until the same has been examined by a competent person and reported safe.

(*To be concluded in September*)

LOSSES IN ELECTRIC TRANSMISSION

When figuring upon the installation of an electric transmission plant it is necessary to allow a sufficient factor for losses. An authority gives these as follows: Loss due to friction of steam engine, 10 per cent.; loss in belting between engine and generator, 3 per cent. If direct-connected this is not figured. Loss in generator, 10 per cent.; loss in line, 10 to 20 per cent.; depending on its length, size of conductor, care in building line, etc.; loss in motors, 12 to 20 per cent. depending on kind used and service required by them; 5 to 10 per cent. loss between the brake horsepower and the machine to be driven, to which may be added the loss in the machine itself due to friction. Total efficiency, 50 per cent., taking the lowest estimate of loss. Where the distance is short and great care is taken in making the installation the actual efficiency may be increased to 60 per cent.

THE ILLINOIS COAL FIELD

*Written for Mines and Minerals, by H. H. Stoek**

History and Geology. Trade, Inspection and Mining Districts

Very few Americans probably realize that the great prairie state of Illinois ranks second only to Pennsylvania as a producer of coal. The traveler through Pennsylvania and West Virginia feels instinctively on account of the mountains about him that he is in a mining region, even were he not visibly reminded of it in many sections of these states by the ever-present pall of smoke by day, and the pillars of fire by night, from the coke ovens. Moreover, in these states, the main transportation lines pass directly through the mining section. Not so in Illinois, however, for the traveler to the Far West who goes by way of Chicago usually passes through only the northern part of Illinois, which is the only section not

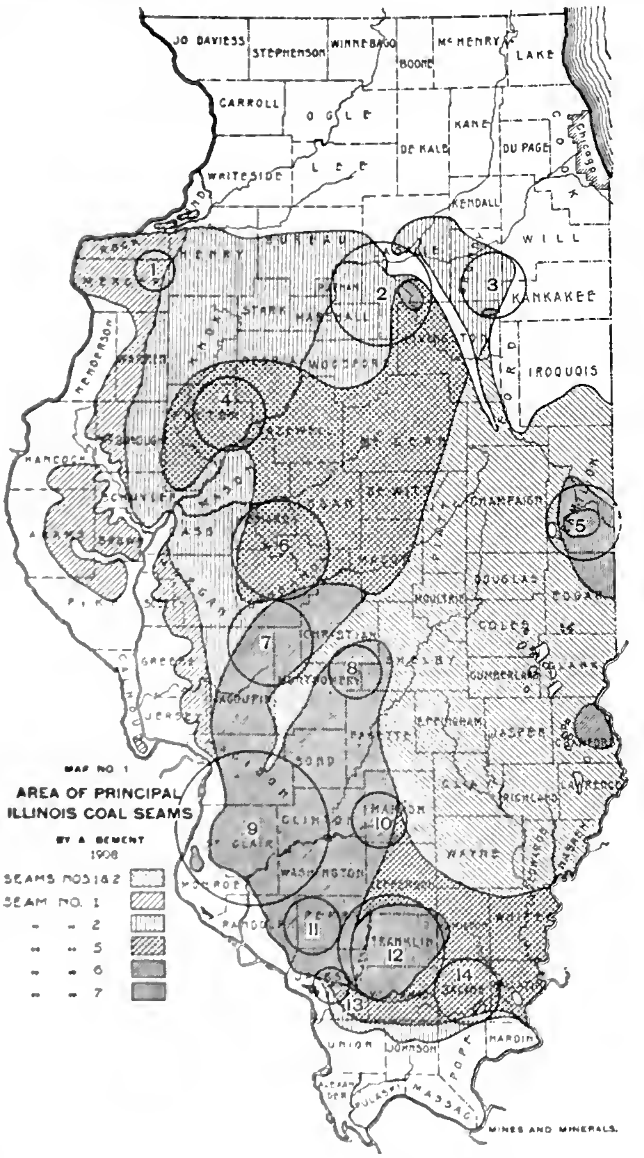


Fig. 1. Map of Illinois Coal Fields

1 Rock Island; 2, Northern; 3, Wilmington; 4, Peoria & Fulton; 5, Grape Creek, 6, Springfield, 7, Virden, 8, Pana; 9, Central Illinois, 10, Centralia; 11, DuQuoin, 12, Williamson and Franklin; 13, Big Muddy; 14, Saline

underlaid by coal. The traveler going southward or southwestward from Chicago to St. Louis or one passing through the southern or central part of the state realizes to some extent that he is passing through a country which has a covering of golden grain and a lining of black diamonds, but even here the surface indications of mining are small compared with those seen in a country where the coal is coked, or in the anthracite region of Pennsylvania where the surface equipment is so much more elaborate.

In Illinois, King Coal cannot be the same despot that he is reputed to be elsewhere, but must reckon with the fair goddess of agriculture who rules over surface lands worth frequently $200 and more per acre, while the coal rights underneath these same lands may be worth only from $10 to $100 per acre.

Coal-bearing rocks underlie three-fourths of the state of Illinois, being under 85 of the 102 counties, the greatest area of bituminous coal in any single state of the United States. The barren areas include a narrow strip along the Mississippi River, a small section in the extreme south, or Egypt, and about one-fifth of the area of the state along the northern boundary.

According to E. W. Parker's "Coal Statistics," "probably the earliest mention of coal in the United States is contained in the journal of Father Hennepin, a French missionary, who as early as 1670 reported a 'cole' mine on the Illinois River above Fort Crevecoeur, near the site of the present city of Ottawa. Father Hennepin marked the location of the occurrence on the map which illustrates his journal. It is also probable that, outside of anthracite mining in Pennsylvania and the operations of the Richmond basin in Virginia, Illinois holds the record for priority of production. The earliest statement that we have in regard to actual mining in Illinois is that coal was produced in Jackson County in 1810 from a point on Big Muddy River. A flatboat was loaded with coal at this place and shipped to New Orleans, but the quantity is not stated. Again, it is reported that in 1832 several boat loads were sent from the same vicinity to the same market. Another record is found stating that 150,000 bushels "or 6,000 tons of coal were mined in 1833 in St. Clair County and hauled by wagons to St. Louis. From 1840 to 1860 the bureau of statistics of the state is without any reliable data in regard to the coal-mining industry, although some scattering statistics are found in the geologic reports published by the state government." Mining is carried on in 52 counties and coal probably occurs in 33 other counties, where it has not as yet been developed. The production of coal from Illinois since 1900 has been as follows in short tons:

Year	Quantity
1900	25,767,981
1901	27,331,552
1902	32,939,373
1903	36,957,104
1904	36,475,000
1905	38,434,363
1906	41,480,104
1907	51,317,146
1908	47,659,690

Illinois has ranked second among the coal-producing states for 23 years excepting in 1906, when on account of a strike in the state, West Virginia secured second place.

Several estimates of the original amount of coal in Illinois before mining began are as follows:

	Tons
F. W. DeWolf, Acting State Geologist	136,960,000,000
M. R. Campbell, U. S. Geological Survey	240,000,000,000
A. Bement	201,299,808,000

Of this amount, according to Mr. DeWolf, 1,041,060,000 tons have been mined and wasted up to the close of 1907, this estimate assuming that 62 per cent. of the coal is recovered in the mining. The output for 1908 was 47,659,690 tons; which is 62 per cent. of 76,870,467. Adding this to the above estimate gives a total of 1,117,930,467 tons to the end of 1908, or a production of about ½ of 1 per cent. of the original content of coal in the state.

The following table gives the production by counties for 1908:

*Professor of Mining Engineering, University of Illinois

Coal Productio of Illinois in 1908, by Counties, in Short Tons

Bureau	1,512,971
Christian	1,377,166
Clinton	1,078,848
Franklin	2,187,383
Gallatin	59,667
Grundy	1,081,442
Henry	141,624
Jackson	624,055
Knox	41,040
La Salle	1,557,173
Livingston	265,666
Logan	372,980
McDonough	17,818
Macoupin	3,894,199
Madison	3,367,820
Marion	981,284
Marshall	393,281
Other counties* and small mines	1,287,937
Menard	355,309
Mercer	376,435
Montgomery	1,410,978
Peoria	921,929
Perry	1,576,891
Randolph	751,605
Rock Island	50,781
St. Clair	3,696,017
Saline	2,552,137
Sangamon	5,015,608
Scott	3,427
Shelby	181,373
Stark	20,351
Tazewell	206,882
Vermilion	2,452,485
Will	162,239
Williamson	5,670,474
	47,659,690

* Bond, Calhoun, Edgar, Greene, Hamilton, Hancock, Jefferson, Jersey, Kankakee, McLean, Macon, Morgan, Moultrie, Putnam, Schuyler, Warren, Washington, White, and Woodford.

There has been a gradual change southward in the center of production and now the counties having the greatest production are in the southern part of the state.

The following table shows the counties having the greatest outputs during the last 20 years:

Counties Leading in Production From 1881 to 1909—Chronological Arrangement

Year	County	Tonnage	Year	County	Tonnage
1881	La Salle	624,900	1895	Macoupin	1,948,992
1882	La Salle	2,365,000	1896	Macoupin	2,097,539
1883	Macoupin	1,233,200	1897	Vermilion	2,000,623
1884	Macoupin	1,164,409	1898	Sangamon	1,763,863
1885	St. Clair	1,202,187	1899	Vermilion	2,221,867
1886	Macoupin	1,085,539	1900	Sangamon	2,519,911
1887	La Salle	1,125,235	1901	Sangamon	2,919,223
1888	St. Clair	1,184,579	1902	Sangamon	3,672,987
1889	Macoupin	1,202,187	1903	Sangamon	4,386,526
1890	Macoupin	1,369,919	1904	Sangamon	4,516,358
1891	St. Clair	1,595,839	1905	Sangamon	4,395,050
1892	Macoupin	1,823,136	1906	St. Clair	4,168,019
1893	St. Clair	2,133,870	1907	Williamson	5,266,452
1894	St. Clair	1,623,684	1908	Williamson	5,367,140

A number of detailed descriptions of the Illinois coal fields have appeared within the past few years, notably, "A Study of Illinois Coal," which appeared in the Transactions of the American Institute of Mining Engineers, Volume XL; another detailed article by Mr. A. Bement, of Chicago, appeared in the Journal of the Western Society of Engineers for June, 1909*. These articles and the reports of the Illinois Geological Survey have been largely used in preparing the following data, and direct reference will be made to them when it is possible to do so without tedious repetition. The writer is also indebted to Mr. F. W. DeWolf for many suggestions and for notes not included in the papers mentioned.

Illinois constitutes the major part of the eastern interior coal field. Of the total area of the eastern interior field, 48,500 square miles, Illinois is estimated to have 35,600, Indiana, 6,500 and Kentucky 6,400 square miles. Before taking up the Illinois field in detail, it is advisable to describe briefly the general characteristics of the entire eastern interior field. The general outline is regular and there are only a few outlying coal areas which are found, mainly toward the northwest. The field may have originally been more extensive, having continued eastward in Kentucky and westward into Missouri, where it may originally have joined the western interior coal field. If this connection originally existed, it has been eroded through the Mississippi Valley.

*Mines and Minerals, July, 1910.

The stratigraphy of the field as a whole has not been worked out sufficiently to definitely subdivide it, and the principal subdivisions now used are based upon state lines, and within the states by commercial or labor districts. Unfortunately, the coal beds have been differently numbered by the State Geological Surveys of Illinois, Indiana, and Kentucky. In Illinois the coal beds are numbered from 1 to 16, 1 being the lowest and bed No. 7 being the highest now worked to any extent, though in Shelby County a bed 2 feet thick and called No. 14 is worked for local trade in a small way. In Indiana, the coal beds are numbered from I to VIII with minor beds marked VI*a*, etc. In Kentucky, the beds are both numbered and lettered, the numbers beginning at the bottom and continuing upward to 12, Nos. 9, 11, and 12 being the principal beds worked. The letters extend from top downward and bed A is the same as bed 12.

At the northern end of the junction between Illinois and the Indiana fields, according to Ashley, No. 6 of the Illinois section seems to correspond with No. IV of the Indiana section. According to DeWolf, toward the south No. 5 Illinois and No. V Indiana are the same. It has not yet been certainly determined if No. 6 Illinois and No. 11 Kentucky are equivalent to No. VI Indiana, or if No. VI Indiana and No. 12 Kentucky are equivalent. A large amount of study and field work have been devoted to the several parts of the eastern interior coal field during the past few years by the State Geological Surveys of Illinois, Indiana, and Kentucky, and the results are being prepared for publication, so that within a comparatively short time, this subject should be cleared up considerably.

The coal-bearing rocks of the eastern interior field belong to the coal measures or Pennsylvanian series of the Carboniferous. The rocks between the coal beds are sandstones, limestones, and shales. The sandstones are not constant in thickness and do not, therefore, give good reference lines, but the limestones, although thin and impure are sufficiently regular to serve as reference planes. DeWolf says: "Probably most of the Pennsylvanian rocks constitute interfingering lenses of comparatively local extent."

The total thickness of the Pennsylvanian rocks probably exceeds 2,000 feet, but around the edges of the basin much has been removed by erosion, and in a large part of Illinois, the basal division is thin or absent. David White, who has contributed largely to recent studies of it, has shown that the earliest beds in Illinois were deposited in a restricted area in the southeastern counties of the state and that the favorable conditions for deposition of the coal measures gradually spread over the state, overlapping the eroded surface of the rocks, which are progressively older to the north.

Ashley says: "The coal measures of the eastern interior coal field rest unconformably on the underlying formations. At the southern margin of the field they lie on the Chester or Kaskaskia limestone, which, in Southern Illinois, has a thickness of 800 feet. Northward, along both the eastern and western borders of the coal field, the Chester thins out, having been eroded before the coal measures' rocks were deposited, and the latter rest on successively lower members of the lower Carboniferous, first, upon the St. Louis, and, as the northern end is approached, upon the Keokuk and the Knobstone. Finally at the northern edge of the field, the Devonian and Lower Silurian formations underlie the coal measures, and consist of alternating beds or lenses of shale and sandstone with which are thinner strata of limestone, coal, and fireclay."

There are three general divisions of the coal-bearing rocks:

First, the base of the measures is a massive sandstone of same age as the Pottsville formations of the Appalachian field. To it belong the Millstone grit of the early Illinois geologist and the Mansfield sandstone of Indiana. According to DeWolf, in Illinois, this section has a thickness of 650 feet or more in Johnson and Hardin counties, but diminishes rapidly in the west and north,

being nearly or quite absent over much of the state. Coal No. 1 of the western counties lies near the top of this formation. Lower coals occur in Southeastern Illinois and Western Kentucky, and some of these were formerly mined.

The second division in Illinois extends from coal No. 9 to coal No. 6, of the western and northern counties, and thus includes all the seams mined for shipment in the state. It is dominated by shale and contains a subordinate amount of sandstone. In age, it corresponds pretty closely to the Allegheny formation or Lower Productive measures of Pennsylvania, since coal No. 6 lies at or near the Upper Freeport, and No. 2 near the Kittanning horizon. This formation extends over nearly the whole area, but its lower beds are not well known in the central part of the basin. At Peoria the total thickness is about 200 feet and at Mattoon it appears to be 300 feet. In Jackson County it is 350 feet thick.

The third and topmost division in Illinois is dominated by shales, and contains no coals of present importance, though some are locally mined on a small scale. This portion of the rocks corresponds to the Lower Barren measures of Western Pennsylvania. It occupies much of the coal area, and reaches its greatest thickness (1,200 feet or more) in the vicinity of Hamilton and White counties. From 275 to 350 feet above its base occurs the Carlinville limestone, which, in the earlier state geological works, was accepted as a dividing line between the upper and lower coal measures.

The Illinois coal field is spoon shaped, the beds dipping gently toward a long axis which lies a short distance west of La Salle, and continues a little east and south to the southwest county of Indiana. According to DeWolf, "the deepest part of the basin is in the vicinity of White County, and from here the strata rise more rapidly to the south than to the north, averaging over considerable distance 40 feet, and locally 100 feet per mile.

The sides of the spoon show some minor longitudinal folds, notably the anticline which runs from LaSalle through the Illinois oil field toward Princeton, Ind., a steep monocline at Duquoin, and a gentle anticline at Belleville. The southern margin of the basin shows numerous minor faults and at least one of consequence, which runs west and a little south from Shawneetown, and has a down-throw to the north of over 1,000 feet. This separates the greater part of the basin from a narrow southern belt of rugged country, characterized by massive sandstones, but containing local areas of thick coal. Igneous dikes and other features along the southern margin of the basin indicate that the structure of the coal field is in part related to the orogenic movements of Southern Illinois and Western Kentucky."

Trade, Inspection, and Mining Districts.—The geological structure of the coal field has largely determined the mining centers, as the first mining has been carried on around the edge of the basin where the coal could be most easily reached. The central part of the basin will probably not be developed to any great extent while there is an abundance of coal about the edge of the basin.

The following brief notice of the several districts will be useful in understanding the references to the different localities and districts of the state.

The state is divided into districts upon several bases as follows:

1. For mine inspection purposes.
2. Geographically, for trade reasons.
3. For labor agreement between the coal operators and the United Mine Workers.

For mine inspection purposes:

District No. 1.—Grundy, Kankakee, La Salle, Putnam, and Will counties, Hector McAllister, Inspector.

District No. 2.—Bureau, Fulton, Henry, Knox, Mercer, Rock Island, and Warren counties, Thomas Hudson, Inspector.

District No. 3.—Livingston, Logan, McLean, Marshall, Menard, Peoria, Stark, Tazewell, and Woodford counties, John Dunlop, Inspector.

District No. 4.—Brown, Cass, Morgan, Sangamon, Scott, Hancock, Schuyler, and McDonough counties, Thomas Weeks, Inspector.

District No. 5.—Christian, Edgar, Macon, Moultrie, Shelby, and Vermilion counties, James Moses, Inspector.

District No. 6.—Calhoun, Green, Jersey, Macoupin, and Montgomery counties, James Taylor, Inspector.

District No. 7.—Bond, Clinton, Madison, Washington, and Marion counties, W. W. Williams, Inspector.

District No. 8.—Randolph, and St. Clair counties, Walton Rutledge, Inspector.

District No. 9.—Franklin, Gallatin, Jefferson, Perry, Saline, Wabash, and White counties, ———, Inspector.

District No. 10.—Jackson, Johnson, and Williamson counties, Thomas Little, Inspector.

In the paper by Mr. Bement already referred to, he divides the state into districts geographically as shown in Fig. 1, and the relative importance of these districts as shown by the output for 1908 is also shown by the numbers.

The districts as set by the Mine Operators and District No. 22 of the United Mine Workers of America for making trade agreements are as follows:

First District.—Bureau, Grundy, Kankakee, Livingston, La Salle, Marshall, McLean, Will, Woodford, Putnam, and Kendall counties.

Second District.—Vermilion County.

Third District.—Brown, Cass, Logan, Mason, and Menard counties; that portion of Macon county lying south of Logan County and west of an extension south of the east line of Logan County; and that part of Sangamon County lying north of a line from east to west, between Chatham and Springfield, located as development of the respective coal seams may indicate.

Fourth District.—Christian, Shelby, Scott, and Morgan counties; that part of Macon County lying east of an extension south of the east line of Logan County; Montgomery County, except mines on the Clover Leaf Railroad as far east as and including Coffeen; also so much of the northwest corner of Macoupin County as to include the mines on the C. & A. R.R. to and including Carlinville, also that part of Sangamon County between Chatham and Springfield not included in the third district.

Fifth District.—Monroe and Randolph counties; that part of St. Clair and Clinton counties south of, but not including the B. & O. S. W.; that part of Washington and Perry counties west of, but not including the main line of the Illinois Central Railroad; also Edwardsville, Glen Carbon, and points intermediate to St. Louis, on the same line of railroad in Madison County.

Sixth District.—Marion and Jefferson counties, and that part of Washington and Perry counties east of and including mines on the main line of the Illinois Central Railroad.

Seventh District.—Wayne, Edwards, Wabash, White, Hamilton, Franklin, Jackson, Williamson, Saline, Gallatin, Hardin, Pope, Johnson, Union, Massac, Pulaski, and Alexander counties.

Eighth District.—Rock Island, Mercer, Henry, Stark, Knox, Warren, Henderson, Peoria, Fulton, McDonough, Hancock, Schuyler, and Tazewell counties.

Ninth District.—Calhoun, Green, Jersey and Bond counties; all of Macoupin County except that part of the northeast corner which includes the mines on the C. & A. R.R. to and including Carlinville; that part of Clinton and St. Clair counties north of and including mines on the B. & O. S. W.; all of Madison County except Glen Carbon and Edwardsville and mines intermediate to St. Louis; also mines on the Clover Leaf Railroad in Montgomery County to and including Coffeen.

ENGINEERING OF MODERN COAL PLANTS

*By Howard N. Eavenson**

This paper will be a general description of the plants of the United States Coal and Coke Co., giving their main features.

A Description of the Plants of the United States Coal and Coke Company

On December 31, 1901, the Pocahontas Coal and Coke Co. executed a lease to the Illinois Steel Co., which was subsequently assigned to the United States Coal and Coke Co., for a portion of their territory containing 50,000 acres of the No. 3, or Pocahontas, seam of coal, 5 feet or over in thickness. The lease contained an agreement that the lessee was to begin work at once, and was to construct 3,000 coke ovens, and to reach an output of 1,500,000 tons of coke at the end of 3 years. Work upon the surveys was started in January, 1902, and construction work on plants Nos. 1, 2, and 3, consisting, as projected, of 950 ovens, was started in May. This work was pushed as rapidly as possible upon these three plants, and in February of the following year, work was started upon Nos. 4, 5, 6, and 7 plants, composing a total, as projected, of 1,100 ovens. In November, 1903, work was stopped upon Nos. 4, 5, and 6 works, and only 100 ovens completed at No. 7 works, which were charged from No. 6 mine. During 1904, a portion of the plants originally projected at Nos. 4 and 5 works were completed and placed in operation, and in 1904, plants Nos. 6 and 7 were completed and plant No. 8 started and completed, making at that time a total of 2,151 ovens. For various reasons, chief among which was the probability of the speedy development of the by-product oven, the original number of ovens was not completed. In the late fall of 1906, the construction of four new plants, Nos. 9, 10, 11, and 12, was authorized, and the construction of these plants was completed last year. The latter four plants are for coal-loading purposes only, and no ovens are projected or intended for them.

With the exception of plants Nos. 1 and 2, all of the mines of the company are drift mines. The No. 1 plant was originally intended to be a drift mine, but on account of a rock fault encountered in what was at that time supposed to be the No. 3 seam, but which was really the No. 4 seam, it became necessary to abandon this plan and to sink a shaft to the No. 3 seam, whose existence in good condition had been found by drilling. The No. 2 mine was originally opened as a drift mine, with a wooden tipple. In 1905 a fire entirely destroyed this tipple, and as it was found necessary, by that time, to use picking tables to clean the coal, the new steel tipple which replaced the wooden one, was built much higher on account of the crushers and picking tables, and a slope was driven to the coal seam a short distance below the original pit mouth. The height of the coal seam above the valley at the various plants varies from good tipple height, at No. 3 works, to an elevation of 600 feet at No. 12 works.

As originally stated, when the first work was started, the largest seam, and in fact, the only one which had been opened in the vicinity of the first operations, was supposed to be the No. 3, or Pocahontas, seam, and work was started with this idea. Subsequent prospecting, however, in the vicinity of No. 4 plant showed the existence of two seams, the lower one of which was found to be identical with the Pocahontas, or No. 3 seam, and the upper one was found to be the No. 4 seam, about 75 feet above the Pocahontas seam. More thorough prospecting demonstrated the fact that, at least along the outcrop, the No. 3 seam had thinned to not more than 3 feet in thickness for a considerable distance on Tug River, and practically the entire length of Sand Lick Creek, one of its main tributaries. Careful prospecting with the diamond drill confirmed the outcrop work, and it was found that this thinning was not merely a local condition, but that over a very large portion of the area which was supposed to contain the No. 3 seam over 5 feet thick, that this seam was less than 4 feet thick. Fortunately, however, over practically all of this same territory, the No. 4 seam, which is of no commercial importance along Elkhorn Creek, excepting in the immediate vicinity of Kimball, had thickened and had become a valuable seam, although it carried more partings than the No. 3 seam, and was also handicapped with what is called a "black-rash" roof. This rash ranges in thickness from a few inches to 2 or 3 feet, averaging about 18 inches. It is composed of thin layers of slate and coal interstratified, is very brittle, and is entirely too high in ash to allow its use in making coke. In some localities both the No. 3 and No. 4 seams are in good condition, and the area of good No. 3 seam coal at No. 1 mine is overlaid with a considerable area of good No. 4 seam coal, and both veins are worked to the shaft. The No. 3 mine is working the No. 4 seam of coal in the territory in which No. 10 mine is working the No. 3 seam, and at No. 11 mine both the No. 3 and No. 4 seams are being worked to the same tipple. The No. 4 seam reaches its greatest development on Sand Lick Creek, and the Nos. 2, 6, 7, and 8, and 9 mines are all working this seam, which ranges in thickness from 6 to 8 feet under the rash roof. At the 12 plants of the company there are 14 mines, of which six are working the No. 3 seam and eight are working the No. 4 seam. At No. 3 and No. 11 mines, in the No. 4 seam, the coal is the thinnest, averaging about 4 feet 8 inches in thickness. The No. 12 mine in the No. 3 seam is the thickest of any, averaging a little over 8 feet. The average thickness of all the coal now being worked is about 5 feet 10 inches. The quality of the clean coal in the two seams is very much alike. The No. 4 seam is a little lower in volatile matter, and a little higher in ash than the No. 3 seam immediately below it, and is a better coking coal, the yield being somewhat better and the structure and looks of the resultant product being considerably better.

The ovens, which have been constructed are of the beehive type, the first ones being 12 feet in diameter. Later this size was increased to 12 feet 3 inches, and the last ovens built were made 13 feet in diameter. A few experimental ovens were made 11 feet in diameter, and practically every type of beehive oven has been tested to see if the structure and yield of the coke could be improved. A number of ovens were built with a flue around the outside of the oven into which air was forced and from which it passed through holes in the crown brick to the crown of the oven. Other ovens were tested with flues under the bottom of the oven, the air entering at the crown, passing over the coal in the oven, thence by a flue at the back under the bottom of the oven and escaping through a chimney. It was found that none of these types gave any better results or any improved yield over the beehive oven when properly operated. An experimental oven was built 16 feet in diameter and still another 25 feet in diameter was tested. Both of these ovens made very good coke, but as they were constructed about the time the last of the ovens were built, no practical results have followed from their test. All of the ovens are charged by electrically driven larries, the current used on the trolley being the same as in the mines, 275 volts. At all of the coke plants, coke-drawing machines have been, or are being, installed, and a little more than two-thirds of the ovens are now drawn by machine. Twelve of these machines use the familiar wedge, which passes under the coke. One employs a scraper which goes into the oven on top of the coke, is forced down through it and operates practically as a hand scraper does. Still another machine which was developed at Gary, uses a shovel, which is forced through the coke and which discharges automatically into a conveyer, from which it passes to a shaking screen and thence to the cars. Fourteen machines are in operation, and four of the plants are entirely machine operated.

As soon as operations were started at the various plants, it was realized that some method must be employed for cleaning the coal, and a Bradford breaker was installed for this pur-

*West Virginia Coal Mining Institute, June, 1910.

pose at the No. 3 mine. After considerable experimenting with this machine, however, it was found that it would not take out the rash, and that in taking out the bone partings which were in the coal, a very considerable amount of coal was wasted. An experimental picking table was then installed at the No. 6 mine, and the results of this were so satisfactory that as rapidly as possible picking tables were installed at all the tipples, the coal in each case being dumped into a small hopper, and being fed from that to the tables. At the last plants, however, it has been found advisable to install bar screens for removing slack from the coal, allowing this to run to a separate conveyer and the coarser coal to be spread on the picking tables. This is a more satisfactory arrangement than to attempt to pick the run-of-mine coal without screening. All of the coal shipped is loaded as run of mine, and no effort is made in the tipple to keep the slack separated from the coarser coal, as it is all run into the same chute and thence into the bin.

The volatile matter in the coal at the eight coking plants does not exceed 18 per cent., and at the No. 1 mine it is as low as 16 per cent.; in fact, the coal which is being coked at our plants is the lowest-volatile coal which is being coked anywhere in the world, either in beehive ovens or by-product ovens, without the addition of some coal of higher volatile matter. On account of this low volatile matter, the coal does not fuse readily, and in order to get a coke of desirable structure, it is necessary to crush the coal. At each of the permanent tipples at each of the coke plants, duplicate crushers are installed through which all of the coal for the ovens passes. The crushers are of the hinged-hammer type. The first screens that were used to regulate the fineness of the coal were bar screens having ¾-inch spaces. It was found at once that this coal was not fine enough to obtain the best results, and after a number of experiments with different kinds of screens, a perforated screen ¼-inch in thickness, having ¼-inch round holes, spaced $\frac{7}{16}$-inch on centers was adopted as a standard. The result of a large number of determinations of the fineness of the coal through these screens is as follows:

Per cent. remains on				Per cent. passes through	
½-inch	¼-inch	$\frac{1}{16}$-inch	....	$\frac{1}{16}$-inch	....
.70	2.23	14.31	....	82.77	....

The crushers are all driven by electric motors, the first ones being belt driven. So much trouble was experienced with bearings and belts with this arrangement, however, that one of the motors was direct-connected to the end of the crusher shaft by a flexible coupling. The results were so satisfactory that all of the crushers are now driven by direct-connected induction motors having a speed of 720 revolutions per minute.

When work was started on the first plants, it was decided to install a central power plant, generating three-phase alternating current at 6,600 volts, and two 400-kilowatt generators, each driven by direct-connected cross-compound engines, were installed in the power plant. On account of the large amount of power consumed in crushing, it was soon found that the power plant, as originally proposed, would be entirely too small, and in 1904 a 750-kilowatt unit, direct-connected to a cross-compound engine was installed. In 1905, a duplicate of this was put in and in 1907, when the four new plants were started, a Westinghouse low-pressure turbine and Westinghouse high-pressure turbine were installed, each of these units being 1,000 kilowatt, and of course, each of them operating condensing.

On account of the scarcity of water, cooling towers were installed to cool the condensing water, and with the exception of a short time during the winter seasons, these towers have been in continual operation. The original boiler installation was four 300-horsepower, horizontal, water-tube boilers. With the additional power units, five 520-horsepower boilers were installed. These are being fired by Jones underfeed stokers, which are supplied with coal from an overhead bin filled by a larry. Duplicate pumps and feedwater heaters are also in operation. The rated capacity of the boiler plant is 3,000 horsepower, and the rated capacity of the power plant is 4,300 kilowatts. Every piece of apparatus at the entire 12 plants, with the exception of the hoisting engine and the fan at No. 1 shaft, is driven by current from the central station. Substations are located at each of the mines in which are installed 15 150-kilowatt rotary converters, furnishing direct current at 275 volts for the locomotives, mining machines, coke-drawing machines, mine pumps, etc. The direct-current load is about 26 per cent. of the current generated, induction-motor load being 74 per cent. In all, there are in addition to the rotary converters,* 8,483-horsepower induction, 6,964-horsepower direct-current motors, and 485-horsepower lighting transformers, a total of 15,932 horsepower, being supplied from the power house. These motors are classified as follows:

Alternating Current

	Horse-power	Per Cent.
Crushers	2,475	15.5
Fans	2,580	16.2
Pumps, outside	1,035	6.5
Compressors	420	2.7
Picking tables	289	1.8
Shops	174	1.1
Conveyers	420	2.6
Exciters	415	2.6
Spares, etc.	675	4.3
Lighting transformers	485	3.0
	8,968	56.3

Direct Current

	Horse-power	Per Cent.
Locomotives	3,215	20.2
Larries	380	2.4
Coke machines	568	3.6
Pumps, mine	260	1.6
Forge blowers	41	.3
Mining machines	2,440	15.3
Hoists	40	.2
Spares	20	.1
	6,964	43.7

The amount of current being generated is approximately 1,200,000 kilowatt hours per month, and the load factor is 39 per cent. The power consumption per net ton of coal produced is 4.7 kilowatt hours. The ratio of motors installed to generator capacity of power plant is 278 per cent., and the boiler capacity is 419 per cent. As previously stated, there are substations at each of the mines, nine of these stations containing one converter and three of them having two each. The substations are usually located at the drift mouth, and in most cases the motor driving the mine fans is in the same building. Usually the power transformers for reducing the high-tension current to 440 volts are located in these substations, although at some of the works it is more economical to have these transformers located closer to the tipples than to the substations, and in these cases they are located in separate buildings. The current is conveyed to the different substations from the power house by wooden pole lines, excepting in the case of the line from the central station to No. 6 works, where, for a distance of 7,000 feet, a very heavy galvanized steel tower line has been constructed. This line carries now two separate circuits, one for Nos. 6 and 7, and the others for Nos. 8 and 9 mines, and provision is made for a third circuit to proposed plants on the waters of Dry Fork. The wooden poles are located from 100 feet to 150 feet apart, while the steel towers are about 520 feet apart. The voltage used on the transmission lines is 6,600 volts, excepting on the line to No. 12 works, where 22,000 is used, the step-up being made by oil-cooled transformers in the power plant. The distance of this transmission is approximately 9½ miles. All of the transmission lines are calculated for 5 per cent. line loss.

The mines are all ventilated by fans, and excepting in the case of No. 8 mine, where two small wooden fans are in use, these are Clifford steel fans. Five of them are 20 feet diameter by 6 feet wide, one is 16 feet diameter by 6 feet wide, and six are 13 feet diameter by 6 feet wide. The delivery of these fans is, at full capacity, from 300,000 to 400,000 cubic feet of air per minute against a mine resistance of 4 inches. Excepting one fan at No. 1 shaft, all of these are motor driven, seven being belt driven, and five being direct connected. In every case the motors are two-speed induction motors, being run usually for 4 or 5 years at the lower speed before increased size of the mine demands the full capacity of the fan. The direct-connected fans are driven by motors having a capacity of 150 horsepower at 120 revolutions, and 300 horsepower at 240 revolutions per minute. These, of course, are now being operated at the lower speed. On account of the large percentage of induction load, the power factor conditions at the central station are not as good as they should be, and during this summer, two synchronous motors, to be used as rotary condensers and also to drive two of the mine fans, are being installed. The motors will be direct-connected to the fans by flexible coupling, will be self-starting, and will each have a capacity of 400 kilowatts. The useful work in each case will be about 200 horsepower, leaving the remaining capacity to improve power factor conditions. The motors are to operate at 187 revolutions per minute, and will replace two of the belt-driven units.

The gauge of track in all of the mines is 48 inches, and the original intention was to keep all of the mine cars uniform, the capacity selected being 93 cubic feet. On account of the low coal at Nos. 3 and 11 mines, however, it has been necessary to put a smaller car at these places, and at these two mines, cars having a capacity of 62 cubic feet are in use. These smaller cars have 14-inch wheels, all of the larger ones having 18-inch wheels and all $2\frac{3}{16}$-inch cold-rolled steel axles. The No. 11 mine is equipped with steel cars throughout, and at No. 9 mine, about one-half of the cars in use are steel of the semicylindrical type. At No. 10 mine a few steel cars are being tested, but at the remaining mines the cars are all of wood.

At six of the mines steel tipples have been erected, and two more are under construction this year to replace temporary tipples at Nos. 4 and 5 mines. At No. 3 mine a wooden tipple was erected originally on account of the time necessary to secure the steel, and at No. 8 mine a wooden tipple was erected to work out a small piece of coal which could not be conveniently worked to the permanent steel tipple which will be erected later. At Nos. 11 and 12 mines, the life of the plants are not sufficient to justify steel tipples. All of the tipples use Phillips automatic cross-over dumps and in every case the coal is dumped into a small hopper and from thence passed to the picking tables, all of which are 4 feet wide and are motor driven, running at a speed of 50 feet per minute. These picking tables at all of the coal plants feed directly into the chutes and then into the bins; at the coke plants the coal is fed from the picking tables to chutes to the crushers and from thence to the bins. At the No. 2 mine the dump is located at the foot of the slope, from which point an overlapping bucket conveyer carries it about 260 feet to the tipple. This conveyer is on a grade of 40 per cent. At No. 9 plant the difference in elevation between the mines on opposite sides of the hollow is so great that the coal from one is brought directly to the top of the tipple, while the coal from the other one is lifted to the same elevation by a rubber-belt conveyer 210 feet long, on an incline of 28 per cent. At No. 12 mine the coal is 600 feet above the loading point, and about 1,200 feet horizontally from it. The dump and picking tables are located at the top of the hill and the coal is brought down the hill by means of three retarding conveyers which discharge into each other and into the bin. At all of the later bins the gates are operated by compressed-air cylinders, and, as the bins are just the length of a car, it is no trouble to load a car of mine-run coal in 5 minutes. The air cylinders are controlled by three-way valves, operated by levers.

Excepting for the first two openings, which were timbered on account of lack of time, all drift mouths and all fan openings are arched with concrete.

At all of the coke plants the water supply for the ovens is obtained from the river or from the branches on which the plants are located when these afford water. The water is pumped from the streams by direct-connected motor-driven centrifugal pumps to large steel storage tanks. All plants have an auxiliary deep-well supply, which is used during the dry months of the late summer and fall. Three of the plants have deep-well pumps, but at all of the others air lifts are used, the water being blown into small tanks, from which the centrifugal pumps handle it. At four of the plants, large sumps have been excavated in the rock and concreted. In these the ground and stream water collects, and the well water is blown. At Nos. 9, 10, 11, and 12 works, the water supply is for house use only, and at these plants nothing but deep-well water is pumped.

The houses erected for the miners are all well built, of good size and conveniently arranged. They range in size from a single two-room house to a double five-room house, but by far the greater number are single three-room or double three-room houses, as this type seems to appeal more to the miners in this field than any other. Most of the houses are plastered, about 60 per cent. have water in the kitchen, about 80 per cent. are lighted by electricity, and a large percentage are fenced. The foremen's houses and houses of a better grade are built with cellars, bath rooms, and all the usual modern conveniences. At Gary a number of the better grade of houses are heated by steam from the main boiler plant. Prizes are offered each year for the best kept gardens and the best display of flowers around the houses. The lighter side of life has not been entirely neglected and at different works are ball grounds, tennis courts, etc., and at Gary the company built and operates a bowling alley and pool room. Under a lease from the company, a skating rink has been built and is operated at Gary, and furnishes a very convenient place for dances and meetings of all kinds. A moving-picture show was established by a lessee of the company, but this venture did not prove a success financially and has been abandoned. A bank has been organized by the employes of the company, and in a little over 3 years its savings account, on which interest is paid semiannually, shows deposits of more than $70,000 by the miners and other employes around the works.

WEIGHT OF FULMINATE OF MERCURY

The importers, E. I. du Pont de Nemours Powder Co., contended that the weight of fulminate of mercury on which duty was assessed was excessive. Protest was sustained. No attempt was made to weigh the article exclusive of the water accompanying it, on account of its highly explosive nature. Duty was therefore assessed upon the invoice weight of the merchandise. The merchandise was imported in canvas bags, which were themselves contained in rubber bags. These rubber bags were placed inside of small casks and each of these casks was contained in a larger cask. Water was placed in the rubber bags so as to prevent friction in transportation, and thereby prevent explosion. The decision says:

The invoice weight of the merchandise is shown to be 12,000 pounds. The weight of the fulminate after the extraction of the water is shown to be 11,661 pounds and 15 ounces, showing a difference between the article containing the water and without it of 338 pounds 1 ounce. The importers claim that no duty should be assessed on this amount of water. We think the contention is well taken and therefore reverse the decision of the collector and authorize a reliquidation of the duty so as to make an allowance by way of reduction from the invoice weight of 338 pounds.

TEACHING ENGLISH TO FOREIGNERS

By Peter Roberts *

A Practical Method for Use of Adult Foreign-Speaking Persons

The great need of the foreign-speaking mine workers is a working knowledge of English. They are anxious to understand the boss, to learn the art of mining, and know how to best preserve life and limb. A system which gives these men a working knowledge of English quickly and interestingly must commend itself to employers in the mining industry; such a system is the one issued under the caption: "English for Coming Americans."

Children of every nation learn with equal ease their mother tongue. They can do it because they depend wholly upon the organ of language—the ear, and practice their vocal organs to reproduce what they hear. This is nature's way to give her children practical knowledge of language; no sooner does a child learn a word or sentence than he uses it to make known his daily wants. By the time he is four or six years of age he can talk fluently, he understands what he is told, and when he begins his school life the teacher has a medium of communication through which knowledge may be given the child.

The foreign-speaking, when they come to America, are like the babe, deaf and dumb to the English language. They possess the organ of language—the ear; they have articulating organs. What they need is a wise and sympathetic teacher to train their ears to the English sound and help them to adjust their vocal organs to utter it. This is what the mother does, and every normal child talks; if it were systematically done with the foreign-speaking in the mines, every one of them would talk and understand every-day English within six months.

The foreign-speaking is like the child—in not knowing our language; but unlike him in possessing a mature intellect. Children's books don't interest him; lessons for children do not appeal to him. He has a logical mind, his daily life is rich in experience, he is employed in a specific industry, and he lives in a society that carries on its affairs by every-day English. These facts should be taken into account when we plan to give this man a working knowledge of English. The lessons should recognize the laws of logic; they should appeal to the adult mind; they should hitch on to his daily experience; they should pertain to the work he does; they should contain every-day English, such as is heard in the shop and on the street, in the mill, and in the mine.

The preparatory course in "English for Coming Americans" comprises 30 lessons, divided into three series:

A.—Domestic:

1. Getting up in the morning.
2. Getting wood to light the fire.
3. Lighting the fire.
4. Preparing breakfast.
5. Table utensils.
6. Eating breakfast.
7. A man washing.
8. A family of eight.
9. Welcoming a visitor.
10. Going to bed.

B.—Industrial:

1. Going to work.
2. Standing a prop.
3. Guarding against fire.
4. Cleaning and loading coal.
5. Drilling a hole.
6. Preparing a cartridge.
7. Fixing and starting a shot.
8. A man injured.
9. Looking for work.
10. Quitting work.

C.—Commercial:

1. Writing a letter.
2. Buying stamps.
3. Buying a ticket.
4. Going by train.
5. Pay day.
6. Buying a hat.
7. Going to the bank.
8. Sending money to the Fatherland.
9. Buying a lot.
10. Building a house.

Each lesson deals with every-day experiences, the sentences are simple but practical, the lessons conform to the laws of economics, the industry is carefully studied in the second series and the scholars are taught how to perform work as well as how to talk English. A new lesson is given each evening and the men are brought under new relations so that their interest is held.

A sample lesson will illustrate how the lessons are built. Take lesson first in Series *A*:

GETTING UP IN THE MORNING

awake	I awake from sleep.
open	I open my eyes.
look	I look at my watch.
find	I find my watch.
see	I see what time it is.
is	It is six o'clock.
must get up	I must get up.
throw back	I throw back the bed clothes.
get out	I get out of bed.
put on	I put on my pants.
put on	I put on my stockings and shoes.
wash	I wash myself.
comb	I comb my hair.
put on	I put on my collar and necktie.
put on	I put on my vest and coat.
open	I open the door of my bedroom.
go down	I go down stairs.

The language is simple, given in sentence form, which is the unit of language; it deals with a daily experience which it clothes with a new garment; the words are such as men use a thousand times a day, and the adult is interested and his attention held.

In our classes no books are given the pupils. Before the learner sees a word or writes a word of the lesson, he knows it all and can rehearse it as the teacher goes through the pantomime suggesting the various stages in the act of getting up. When the pupil is able to do this, he is given a sheet containing the lesson and he reads it. On the reverse side of the sheet the lesson is in script, and he writes it out copying from the sheet. Then the lesson is reviewed and comments such as "good," "that's right," "splendid," "fine," "first-rate," etc., thrust in by the teacher as the scholars rehearse, so that soon the pupils learn the proper use of these words and bring them to practice.

How long will it take a foreign-speaking man to learn such a lesson? About 30 minutes. Our classes last about an hour and a quarter, and if the teacher knows his business there is not a dull moment in the session. The scholar begins to talk the very first lesson and when he is through he has more than 50 English words which he can pronounce and use correctly in the connection in which he learned them. In subsequent lessons he meets words he knows, but under different relations, and he is interested in meeting old friends in new connections. The teacher reviews each lesson as he goes along, so that soon teacher and pupil converse in simple sentences, and by the time the 30 lessons are completed, the scholars will have command of between 600 and 700 words, used daily in the affairs of life. With this equipment he will be able to understand the foreman, do his work more intelligently, know the routine of the industry in which he is employed, and be better able to take care of life and limb in hazardous work.

*An address delivered before Mining Institute of America, June 28, 1910, at Uniontown, Pa

The system of teaching is not a philosophical scheme although based on sound philosophy. It has been in practice for two years, and last year nearly 10,000 men learned the English language by it. There are around each colliery in the country men who are able to take this system and by it teach English to the foreign-speaking employe in the mine. All that is needed is an organized effort. If the men in charge of the mining industry say, "it will be done," within a year every man now employed in the mines would have a working knowledge of English, and once this is realized the new men of foreign speech, entering the industry, can easily be taken care of.

I cannot point out the beneficent results that would follow. A few may be mentioned: fewer accidents; less destruction of property; less friction between employes; better understanding between employer and employe; the disappearance of the straw boss, the interpreter, the demagogue, and racial cliques; greater intelligence, better understanding of American industrial methods, and a sense of greater responsibility in social, political, and moral affairs.

COLLIERY SURVEY NOTES

Written for Mines and Minerals, by Ralph Shumway *

As a comparison with Mr. Harrison's system of note-keeping, described in the December, 1909, issue of MINES AND MINERALS, the following system may be of interest:

Method of Recording Notes Used by the Rocky Mountain Fuel Company

This form of notes is now being used very satisfactorily in all of our mines in Colorado. It is only intended for double-entry system of mining with room-and-pillar work; with all entries and rooms carried on sights.

Three sets of books are necessary to complete the record of the mine surveys. These books are standard I. P. loose-leaf books, provided with indexes and sheets which have been printed with the forms shown in the succeeding sketches. These three sets are:

1. Room books, using sheets shown in Fig. 1.
2. Entry books, using sheets shown in Fig. 2.
3. The office record, sheets shown in Fig. 3.

In using the room books, we record certain definite information at every extension of all working rooms. This information would be the pluses on all cross-cuts either from the mouth of the room or from the survey station in the room; the thickness of pillar at every cross-cut and the width of both rooms at every cross-cut, also the plus to the face of the room and the coal measurements at the face. The outline form, as printed on the room sheet, is intended to be used only where the rooms are run regularly. In some cases rooms are cut up very badly, necessitating the use of free-hand sketching, which is usually done on the opposite side of the page.

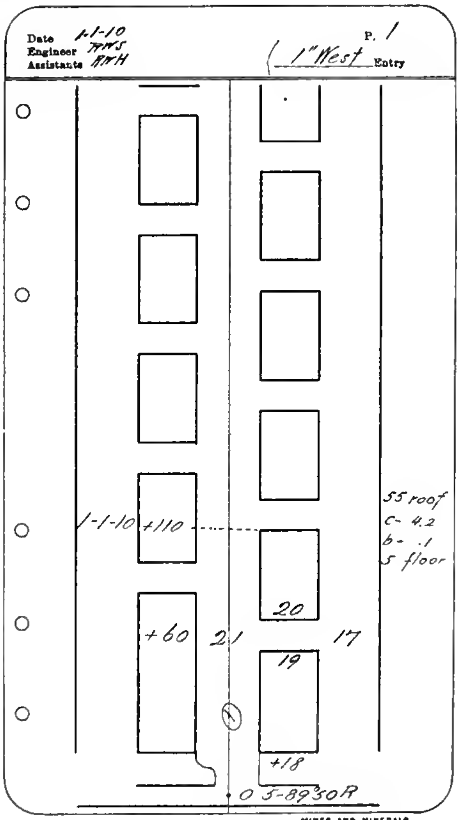


FIG. 1

In entry work the angles are read by deflection and doubled at each station. The record at each station includes the description of the station, needle course, vertical angle, including H. I. and F. S., horizontal angle, and slope measurement. The

P.	Alpine Mine				1" West Survey		
Station and Hor. Dist.	Hor. Angle and Mag. Bear.	Slope Dist.	Ver. Angle	Calc. Bearing	Lat.	Dep.	Elev.
* 5° 139.32 * 4°	N10°10W 0°00	139.37	+1°31	N10°00W	N8776.57 +137.20 N8639.37	E2589.45 −24.19 E2613.64	+365 7637.63

FIG. 3

slope measurement is read going in the entry and again on leaving the entry. As side notes to the regular transit notes, are measurements of the thickness of coal at the face, including all impurities; the width of entry; the pluses on all cross-cuts and rooms; the thickness of pillar at the cross-cuts and the center to center of the entries. As in the case with the room notes, the printed form is used only in routine double-entry work. In case of cut-offs or single entries, sheets similar to Fig. 1 are used with sketch and the same notes, as described above, using the dividing line as the survey line.

In transferring entry notes to the office book, the essential notes of the transit survey are copied and the latitude and departures figured in the office book. This does away with the greater part of the labor of preparing an office copy. If it is necessary to use the latitude and departures at the mines, the office book can be taken along. In that case all the notes are in the field at one time, which is somewhat objectionable, but with a small engineering force it is found that the small amount of labor in copying notes more than compensates for this objection.

The advantages of loose-leaf notes are: All entry and room notes are continuous; compactness; ready accessibility; clearness, as it reduces to a minimum the necessity for free-hand sketching; readiness of expansion, as in the case of royalty measurements; a minimum amount of copying of notes.

As each entry or room is completed, the complete notes for it can be filed in the office under similar indexing.

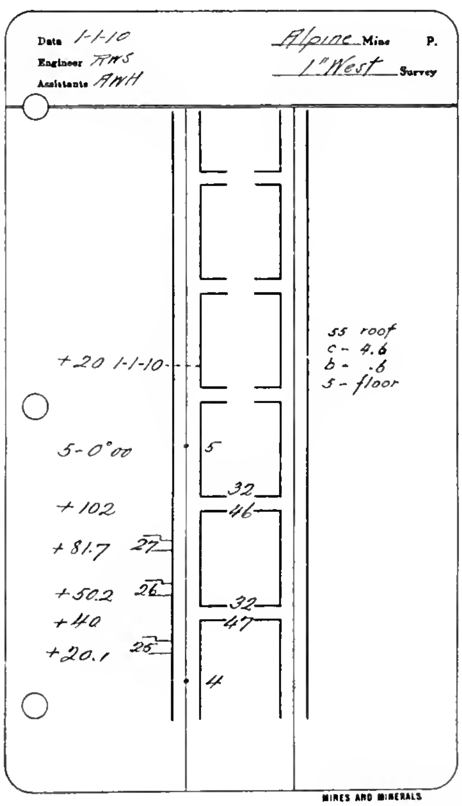


FIG. 2

*Chief Engineer, Rocky Mountain Fuel Co., Denver, Colo.

THE ACID SPECIFIC-GRAVITY TEST

Written for Mines and Minerals, by A. Langerfeld

Amount of Impurities Allowed in the Various Sizes of Anthracite. Methods of Testing for Them

Anthracite to be marketable must be sized and cleaned. At every colliery there is an inspector who examines each car loaded for shipment, to ascertain its fitness for market. If he finds the coal contains too much rock, slate, or bone, or that it is rusty or muddy looking, or contains over or under sizes, he condemns the car, and the coal must be run through the breaker again.

For the sake of uniformity certain percentage quantities of coal under or over the size being shipped are made allowable; also certain percentage quantities of impure coal called "bone," and of slate or rock.

These are as follows:

1. Steamboat. Very little of this size is now made, and some collieries make none at all. The size is generally larger than 6 inches and every piece must be clean coal, and not much under size. It must be free from mud or discoloration.

2. Lump coal at some collieries is from 3-inch to 6-inch diameter, and at others, from 4-inch to 5-inch diameter. All pieces must be nearly pure coal, not much under size, and all free from discoloration.

3. Broken, or grate coal, must pass over about 2¼-inch round or 2¾-inch square holes, and the largest pieces must pass through about 4½-inch round or 4-inch square holes. It may contain an unobjectionable admixture of larger or smaller pieces; 1 per cent. of pieces of slate or rock (stones), and 2 per cent. of pieces of bone.

Pieces of material containing less than 40 per cent. carbon are classed as "slate," and pieces containing hardly any carbon are classed as "rock." Pieces containing from 40 per cent. to 65 per cent. of fixed carbon are classed as "bone," and pieces containing over 65 per cent. of fixed carbon are classed as coal. At some collieries 70 per cent. carbon, instead of 65 per cent., is classed as coal. The inspectors use their judgment in deciding the quality of each piece, but if a dispute arises, samples of the disputed kinds of pieces are sent to a chemist and analyzed. A sample piece of the analyzed quality is then kept on hand for comparison. This is generally accomplished by comparing the colors and apparent specific gravities by holding the pieces in the hand.

There is generally a close agreement between inspectors as to the quality of a car of coal, or of any one piece, but not always. One piece of coal in my possession was pronounced good enough to pass as coal by one inspector, as bone by another inspector, and as slate by another one. But this is a peculiarly exceptional piece. Sometimes there is an equally wide difference of opinion as to the quality of an entire car of coal, for instance, a car of washery buckwheat coal was billed by the inspector as containing 8 per cent. slate, but at the destination the receiving inspector stated it contained 40 per cent. slate. This is an exceptional case, due to discoloration after drying, but it proves that for small sizes of coal the "hand test" alone is unreliable, and therefore the more positive specific-gravity test is in use at some collieries.

Egg coal which is next smaller in size to broken coal varies at different collieries. The smallest pieces pass over 2-inch square holes and over 2¼-inch to 2⅝-inch round holes; the largest pieces pass through 2¾-inch square holes or 3¼-inch round holes. Egg coal may contain up to 5 per cent. oversize and 10 per cent. undersize; also 2 per cent. slate or rock and 2 per cent. bone. It must be fairly free from mud or discoloration.

Stove coal passes through 2-inch to 2⅛-inch square holes and over 1⅜-inch square or 1½-inch round holes. It may contain 5 per cent. egg and 10 per cent. chestnut; also 4 per cent. slate or rock and 3 per cent. of bone. This appears to be a transposition of percentages because there is always more bone than slate in anthracite, but the standard specifications are written that way for this size. Stove coal must be free from mud, and fairly free from discoloration.

Chestnut or nut coal passes through 1⅜-inch square holes or 1½-inch round ones, and over ¾-inch square holes or ⅞-inch round ones. It may contain 5 per cent. oversize and 10 per cent. of undersize, called pea coal, and 5 per cent. of undersize called buckwheat No. 1. It may contain 5 per cent. slate or rock and 5 per cent. bone. Chestnut coal must be free from mud and discoloration.

Pea coal passes through ¾-inch square holes or ⅞-inch to $\frac{15}{16}$-inch round holes and over ½-inch square holes or $\frac{9}{16}$-inch to ⅝-inch round holes. It must not contain an objectionable quantity of larger sizes, over 15 per cent. of buckwheat No. 1, 5 per cent. of the size termed rice, nor more than 10 per cent. of slate or rock, and not much more bone than slate. Its color must be fairly bright.

Buckwheat No. 1 passes through $\frac{7}{16}$-inch to ½-inch square or $\frac{9}{16}$-inch to ⅝-inch round holes, and over ⅜-inch to $\frac{5}{16}$-inch round holes or ¼-inch square ones. It may contain an unobjectionable admixture of oversize but not more than 15 per cent. of smaller coal, nor more than 15 per cent. of slate or rock, and not so much bone as to make the mass look dull in color. It must be washed free from mud and not contain more than 2 per cent. of ashes if it is washery coal. Ashes are also counted in as part of the allowable slate.

Buckwheat No. 1 is the largest size to which the specific-gravity test is applied. One large coal company has adopted the rule that from 15 per cent. to 18 per cent. of the pieces in buckwheat No. 1 size may be heavier than 1.7, so that in an "acid test," made by throwing a sample into a liquid of 1.7 specific gravity, at least 82 per cent. of the sample will float.

Rice coal passes through $\frac{5}{16}$-inch to $\frac{11}{32}$-inch round or ¼-inch square holes, and over $\frac{3}{16}$-inch round or ⅛-inch square ones. The quality of rice coal is not specifically defined, but must present a good appearance and be free from mud. It is generally understood that it may contain up to about 20 per cent. slate. Rice coal is sometimes subjected to the acid test, and is then generally allowed to contain up to 25 per cent. of pieces heavier than 1.7.

Barley coal passes through $\frac{3}{16}$-inch round or ⅛-inch square holes and over $\frac{3}{32}$-inch round holes. It must present a fairly bright appearance and is not generally tested specifically.

Culm or "gun powder" size is all that passes through $\frac{3}{32}$-inch round holes. It is also called slush. Lately it is being burned in many of the steam plants near the mines, and in some chemical works; some of it is also being used for making briquets.

Birdseye coal is a mixture of rice and barley sizes. Chestnut and larger sizes are called "prepared sizes." Pea is in a class by itself, while buckwheat, rice, and barley, are called steam sizes.

When a car of coal contains considerably less slate or rock than the maximum allowable percentage, then for each per cent. less slate 2 per cent. more bone is generally permissible in that car. Approximately the percentages of the different sizes produced in a breaker are: Steamboat, little; lump, 3 per cent.; broken, 10 per cent.; egg, 13 per cent.; stove, 20 per cent.; nut, 19 per cent.; pea, 18 per cent.; buckwheat No. 1, 10 per cent.; rice, 3 per cent.; and barley, 4 per cent. These percentages depend on the hardness and friability of the coal, and also on the demand for prepared sizes. Often there is no demand for stove and larger sizes which are produced in the mines, and then it becomes necessary to break and so reduce them to nut size. The percentage of coal lost by this method of reduction varies from 5 per cent. to 8 per cent. of the coal mined. This totals an enormous loss during a year, which will, it is to be hoped, be reduced to less than 2 per cent. by the use of modern breaker machinery.

The impurities contained in anthracite are slate, rock,

fireclay, pyrite, and bone. Rock is any kind of stone nearly free from coal; fireclay is also nearly free from carbon; slate may contain as much as 40 per cent. coal; pyrite is generally diffused through pieces of coal and slate, or occurs in small cubical crystals. In some coal beds there are occasional pieces of pure white stone which are principally either lime or magnesia. This is generally produced by percolation into cracks or fissures.

Fireclay is usually immediately below the coal, but it also occurs in bands in the coal. Therefore it comes into the breaker and is classed as slate or rock. It varies in hardness from clay to slate and in specific gravity from 1.9 to 2.5 according to the quantity of coal it contains. Slate in the anthracite measures varies in specific gravity from 1.75 to 3, according to the admixture of carbon or pyrite.

The rock varies in specific gravity from 2.5 to 4, according to the admixture of pyrite.

Pieces of pyrite run from 3 to 5 in specific gravity according to the proportions of coal and slate they contain.

The specific gravity of the lightest pieces of pure Bernice anthracite is only 1.38, but most of the hard anthracite runs from 1.45 to 1.5. Analyses made by Pennsylvania Second Geological Survey give the specific gravity of hard anthracite at from 1.54 to 1.68. This is inclusive of bone and slate, producing from $5\frac{1}{2}$ to $13\frac{3}{4}$ per cent. of ash. The specific gravity of bituminous coal is often only 1.27.

Pieces of anthracite containing only 65 per cent. of carbon theoretically vary in specific gravity from 1.77 to 2.72, because pieces consisting of 65 per cent. of the lightest pure anthracite of 1.38 specific gravity and 35 per cent. of the lightest slate of 2.5 specific gravity, have a specific gravity of 1.77, and pieces consisting of 65 per cent. of anthracite of 1.5 specific gravity and pyrite of 5 specific gravity, have a specific gravity of 2.72. There are few such heavy pieces that could be classed as coal, but in the Buck Mountain coal bed there are many pieces of passable coal whose specific gravity is nearly 2, which in weight is equal to much of the light slate. For this reason a specific-gravity test usually differs materially from a hand test depending on looks and feeling. There is generally a difference of from 3 to 5 per cent. between a hand test and an acid test of buckwheat No. 1. The hand test is usually lower for impurities than an acid test, because many pieces are specifically as heavy as slate and will therefore sink with slate, while in appearance they are good coal. For this reason an acid test made by floating the coal in a heavy liquid only is not very definite. It only shows the percentage of pieces which are lighter than the liquid used, and the percentage of pieces that are heavier. The lighter pieces are sure to be coal, but of the heavier ones some are usually also passably good coal containing some sulphur and iron, or other substances whose specific gravity is higher than that of coal. The method of testing coal by floating it has been called the acid test because sulphuric acid, properly diluted with water, was mostly used as the heavy liquid.

When a dry sample of coal is tested it will show a considerably smaller percentage of slate than when tested damp, because dry slate is lighter than damp slate; and when a wet sample is tested, the adhering water reduces the specific gravity of the liquid, and therefore shows a larger percentage of pieces over 1.7 than the same sample would show dry or damp. In order to obtain uniformity the sample should be wet and the liquid should be tested for its specific gravity after putting in the sample of coal, and then brought up to 1.7 specific gravity before taking out the sample.

If in a piece of coal containing pyrite the coal part has a specific gravity of 1.5, then only 6 per cent. of pyrite makes the specific gravity of this piece 1.71, and the piece will sink in a 1.7 liquid. The acid test applied to such a piece therefore classes it as slate or refuse. But any inspector would pass such material as good coal by its appearance. Similarly a piece containing only 3 per cent. of pyrite and about 11 per cent. of slate would sink in a 1.7 liquid although it contains 86 per cent. of coal and is therefore commercial coal. A piece containing only 1 per cent. of pyrite and about 18 per cent. of slate would also sink in a 1.7 liquid although it would contain 81 per cent. of coal and is therefore also commercial coal. Such a piece of coal is classed as slate or refuse by an acid test, but would be classed as coal by a hand test.

These are the reasons why there is generally a difference of several per cent. between an acid test and a hand test, and why the hand test is more nearly correct. But a hand test is not as uniform as an acid test, because a hand test is a matter of personal judgment of the inspector, and this is almost sure to vary as much as anything else that depends on the personal equation, as illustrated by the tests mentioned for the car of buckwheat coal. As a rule the difference in hand tests is less than 1 per cent. for prepared sizes and seldom over 3 per cent. for steam sizes.

For testing steam sizes the greatest advantage of the acid test is that it can be made very quickly, while a hand test of a sample containing so very many small pieces takes considerable time.

The quickest and best way to test steam sizes is to first separate all the pieces lighter than 1.7 by floating them, and then pick the heavier pieces over by hand to complete the separation of all the pieces that are commercial coal. In most cases the retesting by hand of the pieces heavier than 1.7 gives about the same result as an acid test in a 1.75 liquid.

When coal is tested only for slate, or pieces heavier than 1.7, the test does not show the actual quality of the coal, because the percentage of bone also makes a material difference in the heating quality of anthracite. But there is a far greater variety of bone than there is of slate. It is claimed that certain kinds of bone actually produce more heat when burned than pieces of pure coal, generally called "glassy" coal. Glassy coal is jet black and lustrous. Until a few years ago the so-called "blue" coal was all classed as bone. "Blue" coal is dull black, bluish black, or grayish. It occurs in all grades from nearly pure coal to nearly all slate. It is a substance in which coal and clay or other materials are homogeneously mixed, and it often contains considerable pyrite and some graphitic coal. Blue coal is therefore always heavier than glassy, or pure coal, and its specific gravity is from 1.5 to 2. When it contains no iron or sulphur its acid test agrees closely with a hand test, but when it contains considerable pyrite or other heavy materials, its specific gravity is increased far more than its appearance as being bone or slate. The variance in classification between such blue coal tested by specific gravity and by appearance is therefore greater than for bone, in which the slate occurs in layers.

In order to ascertain by specific gravity the percentage of bone or bony pieces in a sample of coal, it is necessary to use two heavy liquids; one for floating coal, and one for floating bone and sinking slate.

To do this it must first be decided what the specific gravity of the heaviest pieces of coal may be, and then the liquid for floating this coal should be a little lighter. Having already adopted 1.7 as the limit for pieces passable as coal and good bone of buckwheat size, it is necessary to adopt a lighter liquid than this for floating only coal, because in testing buckwheat all the coal and fairly good pieces of bone are classed as coal. So far none of the coal companies have used two heavy liquids for one test, but they will probably do so when coal prices go higher for steam sizes, because the consumers will be more particular about wanting to know the quality of the coal.

A piece containing 66 per cent. of coal of 1.45 specific gravity and 34 per cent. of slate of 2.5 specific gravity will have a specific gravity of 1.8 theoretically, but the actual specific gravity of a piece of that composition or quality is usually considerably less than that. The reason for this is the presence of from 4 to 6 per cent. of moisture and some occluded gas. If only $\frac{1}{10}$ per cent. of marsh gas of about .00065 specific gravity

(water being unity) is in a piece containing 66 per cent. coal, 5 per cent. water, and the rest slate, then the specific gravity of this piece is only 1.74. In practice, all the pieces that float in a 1.75 liquid are classable as either coal or good bone in buckwheat No. 1 size, and this generally gives a test of about 3 per cent. less slate than a test in a 1.7 liquid. In view of the fact that pieces containing from 40 to 65 per cent. of pure coal are classed as bone, the liquid for floating bone should be at least 1.8, because a piece containing 40 per cent. of pure coal, or fixed carbon as is usually said, of 1.4 specific gravity, 6 per cent. moisture of 1.1 specific gravity, $\frac{1}{10}$ per cent. marsh gas of .00065 specific gravity (water unity), and the rest slate of 2.5 specific gravity, would be of 1.973 specific gravity. A liquid for floating this would therefore have to be of about 1.98 specific gravity. In practice it is not necessary to make the liquid quite so heavy. This is probably because there is always some air in the crevices and recesses in pieces of anthracite. For testing with two liquids it has been found that 1.85 and 1.65 gravity gave good results.

Experiments made to test coal for specific gravity by displacement in water were not successful, because when a number of small pieces are immersed in water it is almost impossible to force all the air bubbles from between the pieces and out of the crevices in the pieces, and the presence of a small quantity of air makes such a test very inaccurate.

The rule for calculating the specific gravity of material containing various substances is simple if the quantity or proportion of each substance and the specific gravity of each are given. It is as follows:

RULE.—Divide the weight of each ingredient by the weight of the whole piece, and multiply each quotient by the specific gravity of the respective ingredients. Add these products together. Divide this sum into the weight of the piece. The quotient gives the specific gravity of the piece.

EXAMPLE.—A piece of bone weighs $2\frac{1}{2}$ pounds and contains $\frac{1}{2}$ pound slate of 2.5 specific gravity, and $\frac{1}{10}$ pound of pyrite of 5 specific gravity. The remainder, 1.9 pounds, is coal of 1.45 specific gravity.

$$\frac{1}{2} \div 2\frac{1}{2} = \frac{.5}{2.5} = \frac{.1}{.5} = \frac{1}{5} \text{ and } \frac{1}{5} \times 2.5 = .5$$

$$\frac{1}{10} \div 2\frac{1}{2} = \frac{.1}{2.5} = \frac{1}{25} \text{ and } \frac{1}{25} \times 5 = \frac{5}{25} = \frac{1}{5}$$

$$1.9 \div 2\frac{1}{2} = \frac{1.9}{2.5} = \frac{19}{25} = .76 \text{ and } .76 \times 1.45 = 1.102$$

$.5 + .04 + 1.102 = 1.642$ and $2.5 \div 1.642 = \frac{2.5}{1.642} = 1.523$, the specific gravity of the piece.

By simple equation any one unknown ingredient can be found. If it is desired to find out how much of any one ingredient must be added to give the composition a certain specific gravity, the calculation is easily obtained from the formulas:

$m_1 + m_2 = 100$, and $S_1 m_1 + S_2 m_2 = S_3$,

m_1 and m_2 are the percentages of the materials in the composition, and S_1 and S_2 the respective specific gravities. S_3 is the specific gravity of the composition. Then: $m_1 = 100 - m_2$, $m_2 = 100 - m_1$, and then $S_1 m_1 + S_2 (100 - m_1) = S_3$.

There are many ways of making heavy liquids or solutions. The cheapest is sulphuric acid diluted with water, but it is so dangerous to handle and destroys so much, that it is better to use something else. The next cheapest heavy liquid is a saturated solution of calcium chloride with enough zinc chloride added to give the double solution the desired specific gravity. A saturated solution of commercial calcium chloride at a temperature of about 65 degrees has a specific gravity of 1.41, and a saturated solution of commercial zinc chloride 1.85 to 1.9. The specific gravity of a saturated solution of calcium chloride can be raised to 1.55 by adding zinc salt, but no higher, except by warming it and keeping it warm. It is most convenient to use only zinc chloride, and this is not expensive.

The Thoulet solution described in MINES AND MINERALS of April, 1910, has a specific gravity of 3.2, which is more than is required for coal, bone, or slate. It also costs very much more than zinc chloride.

A saturated solution of copperas with 1 part common salt for 25 parts of the copperas will make a heavy solution if kept hot, but is only heavy enough for floating coal, and not bone.

COAL MINING IN INDIA

Statistics which have been published by the Board of Trade relating to coal mining in India show that the quantity produced has steadily risen since 1886. In 1878 the total amounted to 1,015,210 tons, whereas in 1908 it had increased to 12,769,635 tons. Up to 1905 the Raniganj field held the premier position as regards quantity, but it is now second, its output in 1908 being 4,221,781 tons, or 33 per cent. of the year's total. The Jherria fields had the largest production in 1907, and in 1908 it increased its lead by yielding nearly $6\frac{1}{2}$ million tons, or about half the total production. About 90 per cent. of the coal mined in India during 1908 was produced in Bengal. The remainder was raised principally in Nizam's Territory, Assam, and the Central Provinces. During the years 1904-5 to 1908-9 the exports, including bunker coal, amounted to 7.28 per cent. of the production.—*London Times.*

NEW INVENTIONS

Complete specifications and drawings for any of the following patents can be obtained from the COMMISSIONER OF PATENTS, WASHINGTON, D. C., at the following rates:

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
At entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED JUNE 7 TO JUNE 28, 1910, INCLUSIVE

No. 960,867. Giant powder cap, James Henry Fahy, Goldfield, Nev.

No. 960,852. Mine car wheel, Thomas Donohoe, Avella, Pa.

No. 960,958. Coal-mining machine, Lotte Kovatch, Oliveburg, Pa.

No. 960,782. Feeder for tube mills, Everest A. Baylis, Madera, Mexico.

No. 960,441. Production of fine metal tungsten, Elihu Thomson, Swampscott, Mass.

No. 960,940. Art of constructing tunnels in the earth, George W. Jackson, Chicago, Ill.

No. 961,298. Prospector's needle, Dell W. Jewell, Kalamazoo, Mich.

No. 962,040. Apparatus for electrical treatment of ores, William B. McPherson, Los Angeles, Cal.

No. 962,383. Process of treating ores, Elizabeth Barnston Parnell, Carshalton, England.

No. 961,846. Sink-and-float testing apparatus, George R. Delamater, Strong, Colo.

No. 962,493. Method of making sulphuric acid from smelter gases, John Parke Channing and Frederic John Falding, New York, N. Y.

No. 962,612. Means for extracting gold from river beds, John H. Batten, Jamestown, Cal.

No. 962,636. Gold-saving apparatus, William H. Hackney, National Soldiers Home, Cal.

No. 963,002. Mine ventilating apparatus, William Dunn, Wheatland, Pa.

No. 962,678. Ore concentration, Henry Livingstone Sulman, Henry Howard Greenway, and Arthur Howard Higgins, London, England.

No. 962,990. Ore concentrator, Seth R. Swain, Denver, Colo.

No. 962,998. Ore, mineral, and coal breaker, Isaac Christ, Tamaqua, and Henry K. Christ, Mahanoy City, Pa.

No. 962,575. Ore separating or concentrating machine, Walter R. Lins, Philadelphia, Pa.

Mines and Minerals

Vol. XXXI—No. 2. SCRANTON, PA.—SEPTEMBER, 1910—DENVER, COLO. Price, 25 Cents

THE WIND ROCK COAL MINE

*By W. S. Hutchinson**

The Wind Rock Coal and Coke Co. derives its name from Wind Rock, Anderson County, Tenn., which is about 4½ miles north of Oliver Springs. Wind Rock No. 1 Mine is on the southern edge of the Wartburg coal basin, at an elevation of 2,400 feet above the sea level. Its location commands a splendid outlook over the valley to the south. Wind Rock can be reached from Khotan, on the L. & N. Railroad, or from Oliver Springs, on the Southern Railroad.

Undermining by Continuous-Cutting Machines. Use of Coal-Loading Machine. Haulage Systems

The mine was opened in the early part of 1904 by the Wind Rock Coal and Coke Co., composed of Knoxville business men who were considerably delayed in the development of the property by encountering a split in the coal bed composed of hard fireclay, which has since been exploited and found to be confined to 200 acres. The extent of the split, which is lenticular in shape, with its major axis line approximately parallel to Walden's Ridge, was ascertained by driving a pair of entries on the bottom bench of coal. Its maximum thickness was found to be 35 feet. Further exploiting was carried on by skirting the edge of the split and by outcrop prospecting. In 1905 the Wind Rock Mine was purchased by the Bessemer Coal, Iron, and Land Co., of Birmingham, Ala., which now operates the property, consisting of 4,000 acres, as measured within the 30-foot crop line. The entrance to the mine is shown in Fig. 1.

Fig. 1. Entrance to Wind Rock Mine

Geology.—The seam of coal operated probably belongs to the lower Kanawha measures and is known locally as the Dean seam. It is high in the mountains and extends throughout the Wartburgh coal basin, having extensive workable areas in Campbell, Anderson, Morgan, and Scott counties in Tennessee, and also in Bell County, Kentucky.

The inclination of the seam is from 1 to 2 per cent. to the northwest, which is practically in the same direction as the course of the main entry of the mine, shown in Fig. 1, which unfortunately furnishes a grade against the load. This, however, is unavoidable, because the location of the railroad relative to the mine and tipple renders this arrangement necessary. The average height of the coal bed on the entire property, as determined by outcrop openings and prospect holes, differs slightly from the thickness of the coal in the mine proper, which averages 4 feet 9 inches. The roof is exceptionally good, being of solid blue slate from 15 to 30 inches in thickness; and the floor is a massive sandstone formation. At certain points in the mine a soft, white, fireclay, which lies just below the coal, causes trouble and expense by softening and heaving the floor of the excavation as increasing pressure comes on the pillars. This condition of creep seems to be unavoidable, and is not uncommon in mines.

* Knoxville, Tenn.

The analysis of the coal is as follows: Moisture, 1.90 per cent.; volatile matter, 38 per cent.; fixed carbon, 56.64 per cent.; ash, 3.46 per cent.; sulphur (taken separately), 59 per cent,

Coal from this mine is an excellent steam fuel and the entire output is readily disposed of to the railroads in Georgia and Florida.

Mining System.—The mine is developed on the room-and-pillar system, as indicated by the plan, Fig. 2. The main entries and air-course are driven almost due northwest. On the main entry, all cross or butt entries were turned off at right angles to the main entry, but considerable trouble was experienced from cracks in the roof, causing bad top in the butt entries.

Later this trouble was successfully avoided by turning all cross-entries at an oblique angle to the main entries. On entries next to the outcrop, rooms and pillars are each 21 feet wide, with the room necks 21 feet long by 10 feet wide. On all other entries the rooms are driven 30 feet wide with 25-foot pillars, and room necks 10 to 12 feet wide and 21 feet long. In all cases from 150 to 250 feet of solid coal (depending on the amount of cover) is left on each side of the main entries before the first rooms are turned on the various cross-entries.

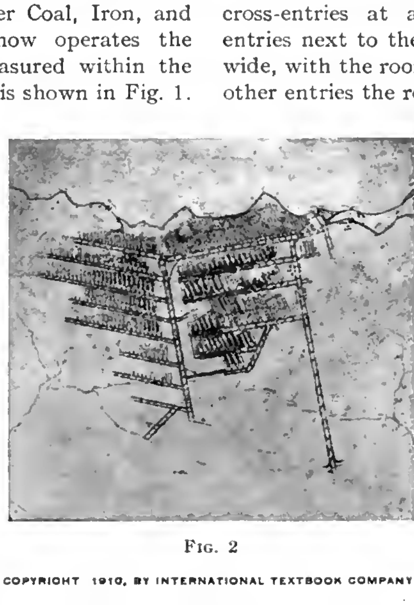


Fig. 2

In all cases, except as noted above, the rooms are driven in 215 feet, leaving 20-foot barrier pillars between the faces of the rooms and the air-course to the next pair of cross-entries beyond. The pillars are at once robbed back to a point within 65 feet of the entry rib. The cross-entries are driven on 40-foot centers, leaving a barrier pillar on the lower side of the air-course. About 75 per cent. of the pillars are cut with machines of the Sullivan

type, one of which is shown in Fig. 3 to be half-way through the pillar on a corner cut, the remainder of the coal being won with hand picks for the sake of safety.

An original modification of mining on a retreating-block system is now being tried. Narrow rooms are driven in pairs on cross-entries on 175-foot centers. When the rooms are driven up, the room pillars are undercut so long as it is safe to do so,

Fig. 3. Making the First Cut

after which the remainder is undercut by hand. The track is advanced after each cut to facilitate loading. The advantage gained lies in the long faces, which are worked quickly and effect a saving of time by concentrating labor, equipment, etc. The roof is carried on timbers until the coal is removed and retreat is made behind a new pillar.

Undermining.—The output is at present about 1,000 tons of coal per day of 9 hours. The coal is all undercut before it is shot, 70 per cent. of the total being mined by Sullivan continuous-cutting electric chain machines, shown in Fig. 5, to be crossing the face of the room under its own power without changing jacks or backing from the face. This machine is the one shown in Fig. 3, disconnected from its starting frame. The remainder of the coal is mined with hand picks and with two chain breast machines. The Sullivan machines are of the low-vein pattern, four cutting to a depth of 6 feet 3 inches, and the fifth, 5 feet 3 inches. They make a cut 5 inches high and are used largely for cutting the long room pillars, where their continuous-cutting principle of operation renders them both rapid and economical. The machines are also employed in mining rooms previous to slicing the pillars and in narrow work in their vicinity. The average rate of cutting is seven to eight 30-foot rooms per shift of 9 hours, each machine furnishing coal for from 20 to 25 loaders. One machine runner recently cut eight rooms 30 feet wide and three narrow places in one shift.

It is found that the best results are obtained by undercutting from 2 to 3 inches above the floor. Owing to its friable nature the coal left at the bottom of the cut is shattered so that it can readily be shoveled up from the floor wherever it has been undercut.

The cost of undercutting the coal with the Sullivan machines is $3\frac{1}{2}$ cents per ton run-of-mine, for room work, with a bonus of 30 cents per place for narrow work. On pillar work and on all cuts 60 feet long and over, the cost per ton is 3 cents, run-of-mine. These costs, of course, include both the machine runner and helper. Twenty-two cents per ton is paid for loading coal over 4 feet in height, with 10 per cent. added when the coal is under 4 feet. This last price includes shooting, loading, and timbering. In blasting in 30-foot rooms, 12 to 14 inches of powder is used in each rib, and in entries about the same amount of powder is required. The average cost for powder per ton of coal is from $\frac{3}{4}$ cent to 1 cent. This low powder cost is due to the fact that undercutting causes the coal to fall readily.

The undercutting is done entirely on the night shift, except the narrow work, which is double-shifted and the coal is loaded out during the day time. This prevents the mining machines being delayed by car blockades or locomotives, when being moved under their own power from place to place, as shown in Fig. 7. It also permits the use of a smaller power plant than would be possible if the undercutting and transportation were carried on at the same time.

Loading Machine.—A coal-loading machine, one of the first built for use at a coal face, is an interesting feature of the underground equipment. This apparatus is practically a dirigible shovel, mounted on a self-propelling truck, which carries also a front and rear endless conveyer. The latter is also dirigible through about 11 feet of arc. The shovel is designed to work in a seam $4\frac{1}{2}$ feet in thickness or higher. It has a capacity of 250 pounds per stroke and makes from 18 to 20 strokes per minute, the machine cleaning up the room from the two tracks, one beside each room pillar. Electricity, gasoline, air, or steam, can be used in this machine for power. Four electric gathering locomotives, one electric train locomotive, two reserve locomotives, and two mine pumps complete the mine equipment.

Haulage.—The method of moving loaded and empty cars is as follows: The cars in the rooms are trammed from the room

Fig. 4. Incline, Wind Rock Mine

neck by the miners and are brought to the mouth of the cross-entry by a gathering locomotive. The tram locomotive coming in switches its empty cars to the gathering motor and picks up the loaded cars. No side tracks are used.

Changing cars is usually accomplished in the following manner: The tram motor, pulling an empty trip in, deposits its empty cars on a cleared cross-entry track, then goes forward

to the next cross-entry and picks up a load. A gathering locomotive distributes the empties left on the first cross-entry and substitutes loaded cars. When a cleared cross-entry is not to be had, the tram locomotives makes a flying switch of its empties to the main entry and switches itself into a loaded butt-entry track. The empties are then delivered to the various rooms, as required, by the gathering motors while the tram motor takes its load to the outside. When the grade permits, the tram motor comes in ahead of its empties, gathers its load as it advances along the entry, and pushes them ahead, distributing the empties as it goes and then returning with the loads.* These methods of course necessitates a large supply of mine cars, some 300 being in use.

Two and one-half per cent. is the steepest grade against the loads on the main entry. The cross-entries are very undulating with a maximum grade of 13½ per cent., which, however, is in favor of the load. On the cross-entries wrecks were frequent formerly, but this trouble has been partly eliminated by replacing the 20-pound rails first used by 35-pound rails, and maintaining a solid roadbed.

On reaching the surface the cars are hauled over the weighing scales to the tipple, where the coal is dumped into monitors, which run by gravity over a standard-gauge inclined plane, to the lower tipple, where the coal is dumped into railroad cars and weighed for shipment. The monitors have a carrying capacity of 10 tons each, and are run double or in tandem, i. e., each trip carries a net load of 20 tons to the lower tipple.

The total length of the incline, a part of which is shown in Fig. 4, is 4,350 feet, and the difference in elevation between the top and bottom is 1,300 feet. The average grade of the incline is 32 per cent., with a maximum grade of 54 per cent. The average time required to run a trip over the incline is 3½ minutes.

The haulage drums are arranged in tandem, the front drum being 8 feet in diameter with four grooves, and the rear drum, 9 feet in diameter with five grooves. The drums are provided with differential groove rings and with 10 inches of brake surface on each side of the grooves, so that the speed of the monitors may be governed with a hand-brake, operated by a man in the tipple. The rope used is 1⅜-inch plow steel, "lang-lay" type so well adapted to inclines where the rope comes in contact with rollers and ties.

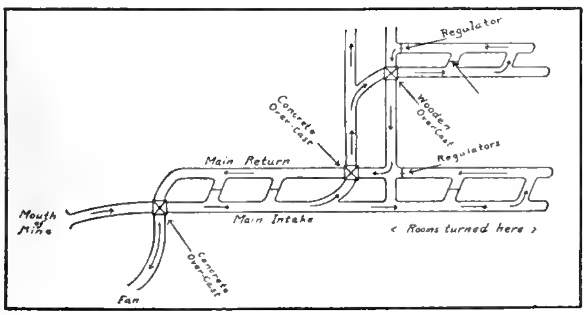


FIG. 6. PLAN OF VENTILATION

Ventilation.—Ventilation is accomplished by means of the split system. The air-current is, so far, split eleven times. Cross-entries, which are driven 1,800 feet as the standard lengtn, off the butt panels, are furnished with wooden overcasts built of 6"×8" framing, boarded on both sides, with clay tamped between the boards. For permanent overcasts, however, quarried sandstone, laid in Portland cement, is used. A plan of the ventilation is shown in Fig. 6.

In the air stops on cross-entries 6"×8" framing is employed, boarded on both sides, with clay filled in between the boards. On all main entries sandstone and Portland cement air stops are used, these being 18 inches thick with a buttress on each side, having a base of 3½ to 4 feet. The buttresses are used for stiffening where the air stops are over 5 feet in height. The regulators are ordinary wood air stops, with a slide-door device that can be locked for any required opening.

FIG. 5. CUTTING ACROSS THE FACE

The fan is of the disk type, belt driven by an electric motor and runs exhausting. It displaces 50,000 cubic feet of air against a ¾-inch water gauge with 310 revolutions per minute.

Surface Plant.—The power house is 250 feet from the pit opening. Direct current at 250 volts is furnishd by two units, one a 150-kilowatt engine generator, direct-connected for the day load; the other a 200-kilowatt engine generator, direct-connected, used for the machine load on the night shift.

A boiler room, equipped with two 150-horsepower high-pressure water-tube boilers adjoins the power house. Coal for the boilers is hoisted from the outside tram road level to the boiler room by means of a small steam hoist.

The water supply is obtained from three small mountain streams which empty into a reservoir constructed in a convenient hollow in the mountain side. The retaining wall is 35 feet in height and has a 15-foot base, channeled 2 feet deep out of solid rock. The wall has a batter of 3½ inches to the foot, giving it a crest nearly 5 feet wide.

The reservoir pumping plant consists of a 5"×8" triplex, electrically driven pump, which raises water through a 2-inch pipe line to a vertical height of 400 feet and discharges it into a 16,000-gallon tank. It is rain and surface water, well suited for boiler use, as it causes little or no corrosion or scale. It is interesting to note that the reservoir has been well stocked with black bass by the government by way of furnishing diversion for the company men during a scarcity of orders.

A well-equipped machine and blacksmith shop, motor barn, fan, oil and tip house, and saw mill, comprise the rest of the surface buildings and equipment.

The writer is indebted to Mr. C. H. Thompson, General Manager of the Wind Rock Coal and Coke Co., for data used in the

*This method of switching is not recommended, as there are too many flying switches and the locomotive should never be behind the trip either going in or coming out.—EDITOR.

preparation of this article and to Messrs. Doyle, Mine Foreman, and Scarborough, Night Foreman, for assistance in securing photographs.

BUREAU OF MINES RESCUE STATIONS

The location of three of the nine new rescue stations recommended for the coal fields of the country by the Secretary of the Interior as a means of reducing the number of deaths in coal mines has been determined upon by George Otis Smith, acting director of the Bureau of Mines.

The first will be at Birmingham, Ala.; the second at Huntington, W. Va., and the third at Wilkes-Barre, Pa. The Alabama station will be accessible to the coal fields of Alabama, Southeast Tennessee, and Northwest Georgia. The Huntington station will cover the coal fields of Southern Ohio, Western West Virginia, and Northeastern Kentucky. The Wilkes-Barre station will cover the entire anthracite fields. Other stations will be established throughout the country as soon as the plans are prepared and the best locations decided upon.

Each station will be in charge of a foreman, a man with practical mine experience, who has been a miner, a fire boss, mine foreman, manager of a mine, or inspector of mines. It will be his duty to train the miners from the coal fields within his district in rescue work.

It is proposed that the miners who work in the nearby mines will, under the guidance of the foreman of the station, form a volunteer rescue corps, ready to respond at once to any emergency call within the district.

The principal equipment at the station will consist of eight so-called oxygen helmets, breathing apparatus, which will permit trained men to enter the most deadly gases in a mine and remain for 2 hours without danger from asphyxiation. An air-tight room will be fitted up and filled with smoke. The miners who are sent to the station will practice with the oxygen helmets in this room until they are perfectly familiar with the apparatus.

It is expected that it will take two weeks to thoroughly train a miner for this work The men will be housed and fed at the station during the training period. When they return to their respective mines it is expected that they will form rescue corps and have properly equipped stations of their own.

At a mine disaster the foreman of the station will have charge of the rescue work A mining engineer will be stationed in the district, whose duty it will be to examine the physical condition of the mines and have general supervision over the training in rescue work

The Bureau also has ordered two portable rescue stations fitted up on specially built railroad cars and ready for immediate response to calls for assistance in the event of mine disasters One car will be stationed at Billings, Mont., to answer calls in Montana and Northern Wyoming. The other car will be for service in the coal fields of Colorado and Eastern Utah. The purpose of these cars will be the same as the rescue stations being established in various parts of the country. They will be fully equipped with apparatus, such as oxygen helmets, and will be sent from one mining camp to another to instruct the miners in their use, as has been done for 2 years at the Pittsburg rescue station.

Fig. 7. All Aboard for the Next Room

One end of each car will be fitted up as an air-tight room to be used in training the men in the use of oxygen helmets. This room will be filled with noxious fumes and the miners wearing the helmets will remain inside for a period of 2 hours in an atmosphere that would kill without the helmets. There will also be sleeping arrangements for 12 men, to accommodate the rescuers at a mine disaster. Each car will contain eight oxygen helmets, a supply of oxygen in tanks, one dozen safety lamps, one dozen electric lamps, one field telephone outfit with 2,000 feet of wire, automatic resuscitating outfits and a limited outfit for use in demonstration and actual practice of equipment in first-aid to the injured in connection with mine accidents.

In addition to the location of branch rescue stations at Birmingham, Ala.; Huntington, W. Va.; and Wilkes-Barre, Pa , mentioned above, the Bureau has decided to have stations at Trinidad, Colo., accessible to the coal fields of New Mexico and Southern and Central Colorado, and at Rock Springs, Wyo.

The erection of the Bristol, England, tower for shot making is credited to William Watts, plumber, in 1769. It was "built" by sawing a square hole in the center of the various floors of his house, and locating the well in the cellar. This tower is still in use, although it has been heightened by the addition of some stories. Watts secured a patent on December 2, 1782, and sold his London rights in 1800 for £10,000 ($48,665).

The lead from which the shot is made is smelted by the present company, and alloyed by their secret process, at their establishment. When molten it is poured into a sieve-like receptacle at the top of the tower and these molten drops, falling into a well, 120 feet below, form the shot, which are then passed through a polishing grader. They are then spilled from a hopper on to an inclined plane; the perfect shot run on to a second plane, while the imperfect drop into the opening between; only the perfect shot reach the last of the four planes.

The tower being only 120 feet in height, a larger size than BBB cannot be manufactured by this process. The larger shot, including shrapnel for the British Government, are prepared in two different ways. For the medium-size shot a wire of the proper material is fed into a machine which, simultaneously, mashes it into a ribbon shape and punches irregular formed shot. The largest shot are made by pouring the molten metal substance into long bullet molds, which, cooling, form irregular shot. The various sizes are then placed each by themselves in gins which are revolved for 6 hours, when the shot come out perfectly smooth spheres.—*United States Consular Report.*

STRIPPING COAL BEDS

Written for Mines and Minerals

Methods in Use in Anthracite Region of Pennsylvania and in Bituminous Coal in Illinois

In a number of places in Northeastern Pennsylvania the coal crops so near the surface it pays to remove the covering and work the deposit by open-cut mining. At Hollywood and Milnesville, 4 miles northwest from Hazleton, open-cut mining has been carried on for at least 35 years. Dr. H. M. Chance described the Hollywood strippings in Volume A. C. of the Second Geological Survey so long ago as 1883. In the vicinity of Hazleton the Carboniferous strata have been folded and subsequently eroded so as to form a number of canoe-shaped basins. In some places the Mammoth bed has been folded, as shown in Fig. 1, so that in the center of the fold there is a mass of rock; in other places the strata have been folded and the rocks eroded so that there is scarcely any cover above the coal, as shown in Fig. 2.

The thicknesses of these deposits vary from 135 to 80 feet, depending on the extent of the folding and erosion they have undergone. The cover above the deposits varies from soil to hard sandstone; however, the coal is in excellent condition, even although at times it is rusty or water stained. The rock walls forming the sides of these basins have steep inclinations

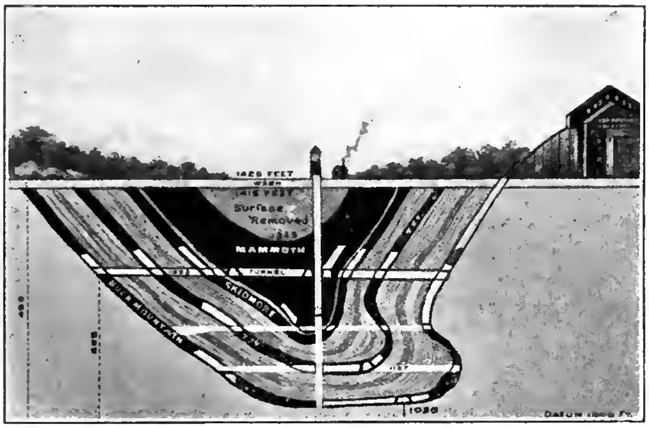


FIG. 1

and this, with other conditions, determines whether the coal is to be mined by the room-and-pillar method, as in Fig. 3, or by the quarrying method, as in Fig. 2.

In case the coal basin is not too long, the breaker is located at one end, and a gangway driven in the bottom of the basin to the other end. In case the basin is long, the breaker is placed at a suitable distance from one end, or possibly on the boundary line between two properties.

In the Hollywood basin the coal was worked from the gangway up each side, by the pillar-and-room system of mining, after stripping off the surface dirt. The rock covering followed the coal into the rooms and was then hoisted out by means of the plane shown in Fig. 3. After the end of the basin was reached the coal pillars were shot down; first, however, removing the rock cover on the coal roof left over part of the excavation. This method of mining was satisfactory and large quantities of coal could have been mined in a short time had there been a market for the product. Mr. Chance, in writing on the cost of mining the Hollywood coal stated: "It will probably pay to remove an average of 2 or 3 cubic yards of debris for every ton of coal." Later the work of stripping was systematized so that it now pays to remove 6 cubic feet of dirt for every cubic foot of coal where room-and-pillar mining is followed. The coal basins, which are practically level at the surface, may carry from 6 feet to an indefinite number of feet of dirt and rock above the coal, but even in the latter case the depth of cover that may be profitably stripped will depend largely on what it is composed of and the facilities at hand for its removal and wasting.

At the Morea colliery, in Schuylkill County, the cover above the coal is soil, and below this rock, which, in some places in the basin, as shown in Fig. 1, reaches a thickness of 80 feet near the center, and will average about 20 feet. In other places in

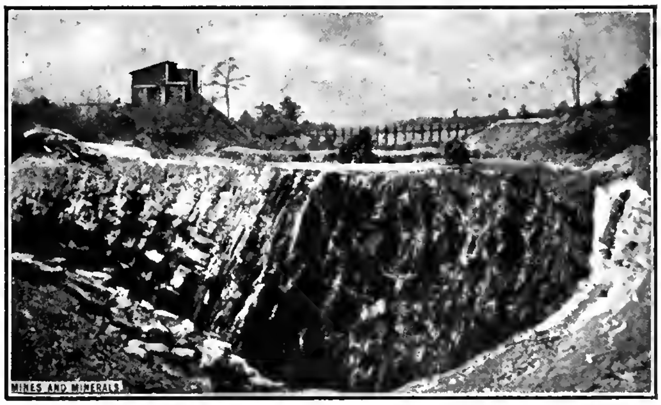


FIG. 2. MAMMOTH VEIN, MOREA COLLIERY

the basin, as shown in Fig. 2, there is only a thin cover, but fortunately where the cover is thinnest the coal is thickest, thus balancing up the differences in the cost of removing the thicker covering. At Morea the soil is removed from the rock by a steam shovel. The rock is then broken with explosives, after which it is loaded into cars by steam shovels and hoisted out of the pit. After the coal has been cleared of rock for a sufficient distance vertical drill holes are put in the coal. As there is a vertical wall of coal a small quantity of powder effectively placed will break down a relatively large quantity of coal, which falls to the coal floor covering the gangway, and from which it is sent down chutes to the mine cars.

There are a number of places in the anthracite fields where flat coal beds outcrop so near the surface that they may be stripped for 100 feet before the cover attains so great thickness as to make the work unprofitable. The Hilldale strippings are

FIG. 3. STRIPPING AT HOLLYWOOD COLLIERY

about 1 mile from the Pennsylvania Coal Co.'s No. 14 breaker and have a section from the surface down about as follows:

Soil, 4.5 feet; coal *A*, 4.5 feet; rock 3 feet; coal *B*, 1 foot; rock, 5 feet; coal *C*, 2 feet; rock, 3 feet; coal *D*, 12 feet.

Coal bed *A* has been exposed so long it is worthless, but it is expected that it will become marketable where it has a rock cover.

Bed *B* is good coal and although somewhat rusty adds to the total tonnage recovered.

Bed *C* is a little short of 2 feet thick and under present mining conditions could not be worked at a profit underground; in the stripping operations, however, it is, like bed *B*, a source of income. Bed *D* is the coal aimed for, and is as good as coal can be, although carrying a slate parting.

The method of stripping followed by Mr. Kinsley, the contractor, is as follows:

First, the top soil and poor coal *A* is removed from the top rock by means of a clam-shell bucket and locomotive crane. The boom on this crane is 42 feet long and can place the top material where it will be out of the way once for all. The dirt is first removed ahead of the crane in the direction it is moving; next from the side of the crane where the mining is to be carried on. The rock and coal are then broken down to the coal bed *D* by blasting. The coal from *B* and *C* is picked out by hand, while the rock is wasted and piled back by the bucket to form the track on which the coal car is shown in Fig. 4. The coal bed *D* is next broken by powder and loaded into the bucket by hand. The bucket is then raised by the crane and the coal dumped in the car as shown in Fig. 4. Anthracite coal is so brittle it breaks in handling, and while it was at first intended to use the bucket to pick up the coal, it was found inadvisable to do so. The method of stripping followed by Mr. Kinsley is equivalent to making a side cut along the crop for the crane track, then excavating below this cut, and filling in. The fill will furnish the road for the next side cut when coming back from the boundary line of the property, as it does for the coal cars in going forward.

Fig. 4. Dumping Coal from Bucket

Near Danville, Ill., there were about 35 acres having a bituminous coal bed 8 feet thick, which was approximately horizontal. The coal bed outcropped on all sides of a flat-topped hill, and had an overburden from 38 to 40 feet thick, composed of coal clay, gravel and about 20 feet of shale. It was decided to strip the coal with a single cut and to take a wide cut so as better to provide for the efficient mining of the coal. The plant decided on was a steam shovel having a dipper of 2 cubic yards capacity, and mounted on a movable platform, provided with a belt conveyer for disposing of the material. The platform, which was 30 feet wide, was mounted on four trucks that were moved as desired on two tracks. The machine, as it appeared in operation, is shown in Fig. 5. After excavating the material above the coal with the bucket, it was swung to a large steel hopper and discharged. From the bottom of the hopper the material was carried on a steel cross-feeder to the lower end of the belt conveyer. The latter was a 40-inch wide belt traveling on a steel arm 105 feet long. The arm was supported by wire ropes from a tower built above the platform to a height of 48 feet. By this arrangement the waste clearance at the outer end of the arm was about 60 feet above the tracks.

The machine is said to have had no difficulty in excavating heavy pieces, stumps, and logs, and depositing them in the space where the coal had been mined. After the overburden had been removed the coal was quarried and loaded into cars on a track laid in the place from which coal had been taken previously.

This is the only reference the writer has to stripping bituminous coal and it has been taken from Volume 28, page 139, Mines and Minerals. As the shovel made a cut, the debris was deposited in the cavity made by removing the coal. The tracks for the machine were laid on top of the coal, which was mined so as to leave a bench for the shovel to come back on.

Fig. 5. Stripping of Consumers Coal Co., Danville, Ill.

It would not be advisable to use steam shovels for scooping up anthracite for two reasons: First, anthracite must be cleaned of slate or adhering rock before it reaches the breaker, and this can be accomplished only by hand. Second, the coal is too brittle and would spall badly.

EVOLUTION OF MINE HAULAGE

Written for Mines and Minerals, by E. B. W.

(Continued from August)

Wire-rope tramways were used 200 years ago, according to Spencer Miller;* cableways, 60 years ago; wire-rope haulage underground about 60 years ago in Scotland; and steam locomotives have been in use nearly as long, at least prior to 1870.

Locomotive Haulage—Steam, Electric and Compressed Air. Calculations

Steam locomotives were first used on return airways. The class of 1883, Columbia College, will doubtless remember receiving a few sparks down their necks and being nearly smothered in No. 1 Drifton, Pa., mine. Outside of a mouthful of sulphur gas, there was no harm, but there have been fatal cases of asphyxiation due to steam locomotives underground.

While steam locomotives are not desirable in gassy mines there seems to be no record which shows they were the direct cause of a mine explosion. Their introduction made it possible to greatly increase the output of drift mines and marked an epoch in the reduction of haulage costs. Where the cost was about 8 cents per ton by mule haulage the locomotive reduced it to 4 cents; at the same time the increased output reduced fixed charges. The first steam locomotives weighed about 6 tons, but as the hauls became longer, necessitating a larger number of mine cars in a trip to maintain or increase tonnage, the locomotives were enlarged. The heaviest underground steam locomotive is that shown in Fig. 52, which was constructed for use at the Pocahontas Mine, in Virginia.

FIG. 52. STEAM MINE LOCOMOTIVE

It is not good practice to fire locomotives underground with soft coal on account of the smoke and gas emitted; therefore, the locomotive shown in the illustration was designed to make a round trip of 7 miles without firing.

To obtain satisfactory work from mine locomotives resistance offered by grades, rolling stock, and tracks must be considered. On grades the resistance due to gravity increases in exact proportion to the steepness of the grade; that is, for a 1-per-cent. grade it is 20 pounds per ton; for a 1½-per-cent. grade it is 30 pounds per ton, etc. The resistance due to friction depends on the physical condition of the track, whether it is wet or dry, clean or dirty, properly gauged, well tamped so as to be solid, or whether the rails are in alinement, and curves made to conform to the locomotive wheel base. The kind and condition of the rolling stock used exerts an influence; for instance, cars in good condition, well made and rigid, may offer a resistance of 8 pounds per ton on a good track, and 20 pounds on a bad track; again, cars in bad order may offer a resistance of 30 pounds per ton on a good track and 60 pounds or more on a bad track. Cars with wheels scraping against the sides may develop 60 to 80 pounds per ton frictional resistance on a good track. Frictional resistance may be determined by noting on what grade a car once started will just keep in motion. If a car will barely keep moving on a 1½-per-cent. grade its frictional resistance is 30 pounds per ton. The tractive force of a locomotive is its pulling strength measured in pounds by a dynamometer. The larger the cylinders of the locomotive are made the greater the tractive force; and the larger the diameter of the driving wheels used on a locomotive the less the tractive force. The first statement is based upon piston area and the steam pressure; the second statement is based upon the crank leverage; for with the connecting-rod close to the axle of the driving wheels there is not so much effective leverage exerted as when it is further from the axle, the cylinder and steam pressure being the same in both cases. The tractive force of a locomotive is the force

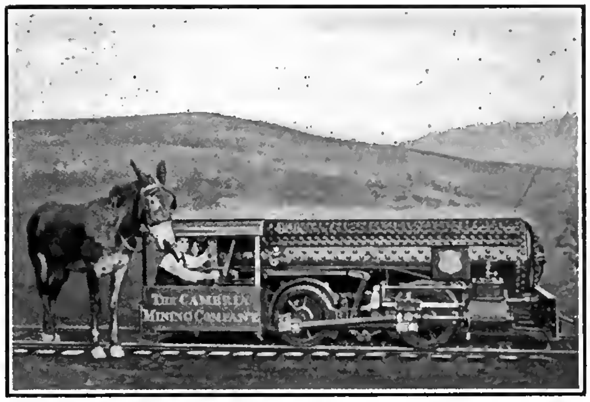


FIG. 53. COMPRESSED-AIR MINE LOCOMOTIVE

necessary to drive the locomotive and to pull the train. It is calculated from the formula:

$$T = \frac{D^2 \times L \times .85\,P}{d}$$

In this formula T = tractive force; D = diameter of each cylinder in inches; L = length of the piston stroke in inches; $.85\,P$ = the effective cylinder pressure which has been found to be 85 per cent. of the boiler pressure in pounds per square inch.*

The hauling capacity of a locomotive is found by dividing the tractive force by the sum of the resistances due to gravity and to friction, and then deducting the weight of the locomotive. This gives the weight in tons of 2,000 pounds which the locomotive can haul.

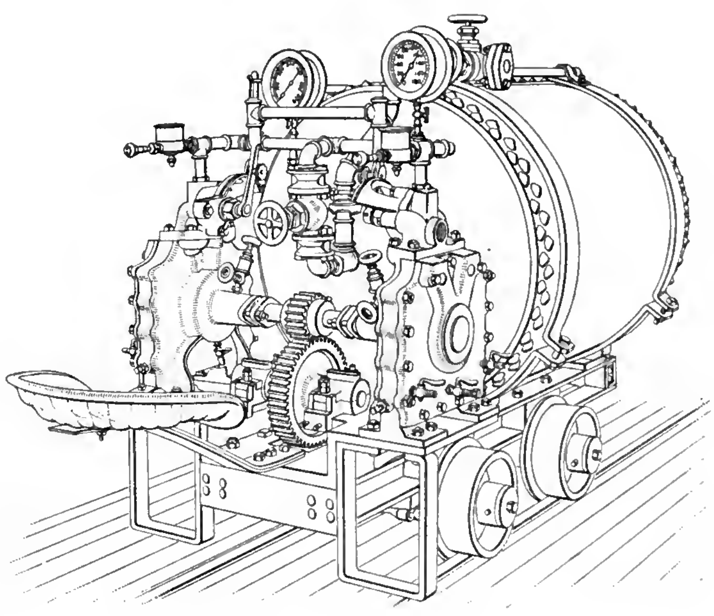


FIG. 54. COMPRESSED AIR LOCOMOTIVE USED IN TUNNEL

*MINES AND MINERALS, Vol. XXIV, page 412.

* H. K. Porter Co.

Example.—What is the hauling capacity of a 20-ton locomotive, 8-inch diameter cylinders; 12-inch stroke, 24-inch diameter driving wheels, working under 150 pounds steam pressure per square inch in boiler; the frictional resistance being 8 pounds per ton, and the grade $1\frac{1}{2}$ per cent.?

Solution.— $\frac{64 \times 12 \times .85 \times 150}{24} = 4{,}080$-lb. tractive force.

$\frac{4{,}080}{8+30} = 107.4$ and $107.4 - 12.2 = 95.2$ tons as the hauling capacity of the locomotive under the above conditions.

(The weight on the driving wheels to produce satisfactory results with mine locomotives must be 6 times the tractive force, hence, $\frac{4{,}080 \times 6}{2{,}000} = 12.2$-ton locomotive.)

It is cheaper to operate a light locomotive wherever the haul exceeds $\frac{1}{4}$ mile than to use mules. However, with steam locomotives, gathering mules must be used to deliver empty cars to the rooms and collect the loaded cars on the partings where the locomotive may reach them. In comparing the cost of mule haulage with locomotive haulage, the tonnage hauled a given distance must be considered. In one instance the anxiety of the buyer of a coal company to purchase something led him to order a locomotive a year before it could be used to advantage. When the cost of hauling 300 tons on the main road to the dump reaches $1,800 per annum, then locomotives can be substituted to advantage. When from any cause the mine is idle the up-keep of the locomotive is practically nothing, while the mules have to be fed and cared for.

In ore mines, where the ventilation is not good, and in coal mines where it would be dangerous to adopt either electricity or steam, compressed-air locomotives are used to advantage. In some respects compressed air as a means of haulage possesses advantages over other kinds of power, but the same can be said of other prime movers under certain conditions. Compressed-air locomotives have been in use for 25 years, and are liked wherever adopted. The great objection to them at first was their length, but this has been partly remedied by using tank tenders to carry a supply of air which would propel them a given distance. To further overcome the difficulty of large tanks an air-pipe storage system was devised, which paralleled the track. This pipe storage is now used almost universally. One feature in connection with air haulage is that small gathering locomotives may be used in place of mules. Fig. 53 shows one of five gathering locomotives adopted to replace mule haulage at the Antelope Mine, Cambria, Wyo. The reader will note that the mule does not seem worried over the change.

It is to be understood that each compressed-air locomotive must be designed to comply with the particular mine road it is to travel, also the air tanks and air-storage system must be designed for the length of haul, and the pressure to be used.

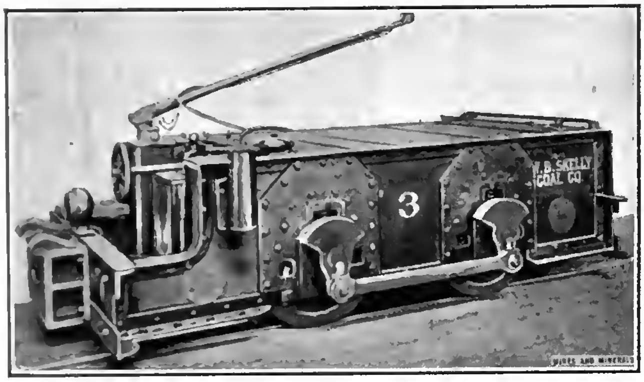


Fig. 55. Baldwin Side-Rod Electric Mine Locomotive

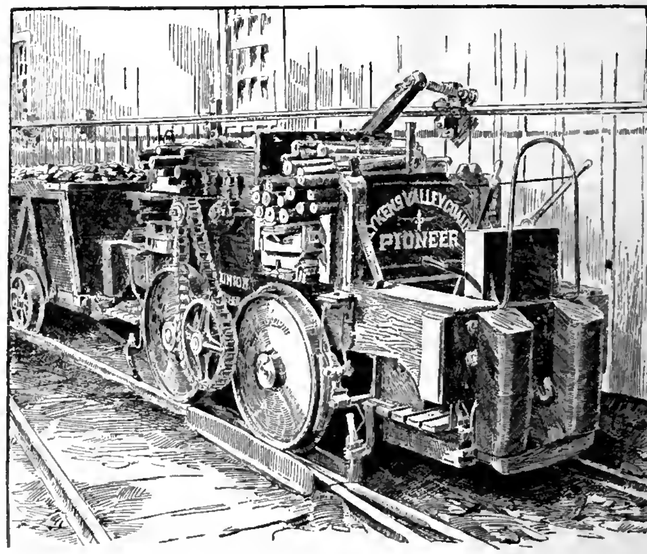


Fig. 56. First Electric Mine Locomotive Used in America, 1887

To install a compressed-air locomotive plant, it is necessary to know the approximate mineral output; design the locomotives of suitable weight and power to overcome the resistances, and then construct the plant to produce and carry compressed air suitable for requirements. The haulage problem includes an air compressor whose capacity can supply the demands of the locomotives for compressed air and furnish a pressure of from 500 to 800 pounds per square inch. Rather than charge direct from the compressor, suitable stations are arranged where the locomotive tanks can be filled with compressed air almost instantly. What is known as the stationary storage system is a pipe line extending from the compressor to the charging station. This system enables the compressor to be run continuously and restore the depleted air pressure after a locomotive has been charged. In order to insure the requisite pressure the relative volume of the locomotive and storage pipes must be compared, as follows:

$$v'(p' - p) = v(p - P)$$

In this equation v = volume of locomotive tanks; P = absolute initial pressure in locomotive tanks; p = absolute final pressure in the

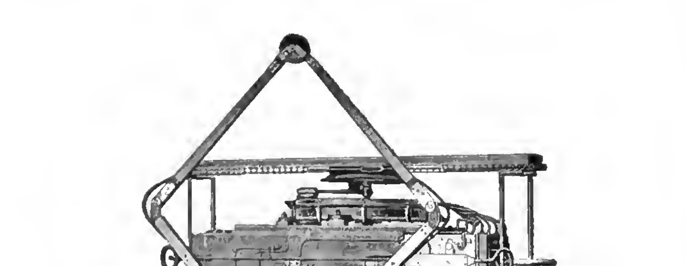


Fig. 57. Electric Locomotive Built 1889, Still in Use

locomotive tanks, usually corresponding to cylinder working pressure; v' = volume of the pipe line; p' = absolute pressure in pipe line to give initial pressure in the locomotive tanks.

To insure the proper pressure, say 800 pounds per square inch, in the locomotive tank, a somewhat higher pressure must be maintained in the pipe line.

To find the volume of the pipe line use equation $v' = \frac{v(p-P)}{p'-p}$, or to find the absolute pressure in the pipe line use $p' = \frac{p(v+v') - vP}{v'}$.

Assume the capacity of the locomotive tank to be 150 cubic feet, with a final pressure of 140 pounds left on the tank at the time of recharging to 800-pounds pressure, and also assume that the pipe line has a pressure of 1,000 pounds above the atmosphere; then, $v' = \frac{150(815-155)}{1{,}015-815} = 495$ cubic feet for volume of the pipe line. Again, $p' = \frac{815(150+495) - 150 \times 155}{495} = 1{,}015$ pounds, absolute pressure, or 1,000 pounds gauge pressure, to give 800 pounds air pressure in the locomotive tank. It is to be expected that various other questions will arise, among them the capacity of the compressor capable of supplying the necessary quantity of air for a haulage system. This is found by calculating the number of cubic feet of free air per minute required to make a given run in a given time. Using the same figures as before and assuming the time occupied in making the run to reduce the tank pressure from 800 to 140 pounds per square inch to be 20 minutes, then at the beginning of the run the volume of free air would be $\frac{(800-140)150}{14.7} = 6{,}734$ cubic feet. Substituting this in the equation $C = \frac{c}{t} = \frac{6{,}734}{20} = 336.7$ cubic feet of free air per minute, as the capacity the compressor must have to fulfil the requirements.*

To ascertain approximately the amount of air needed to perform a given amount of work, tables have been constructed, based on the cylinder working pressure, grades, and resistances. The tractive force of compressed-air locomotives is found from the formula $T = \frac{D^2 \times L \times .98\,P}{d}$, in which .98 is the effective cylinder pressure, there not being so much loss in pressure in air passing from the tank to the cylinders as with steam. For compressed-air locomotives it is always best to install a three-stage air compressor, for with it the final temperature of the air may be kept down to 250° F., provided the plant is not too small. The air locomotive shown in Fig. 54 was used at the Empire Gold Mine, in Grass Valley, Cal., for hauling five 1-ton ore cars on the 2,000-foot level. The distance traveled in a round trip is 5,000 feet. The tank measures about 36 inches in diameter by 48 inches long and carries a pressure of 500 pounds per square inch. The dimensions over all are 5 feet long, 4 feet 4 inches high, and 38 inches wide.

While the locomotive in Fig. 54 resembles a stranded whale, looks do not cut much of a figure in mine locomotives. One of the most picturesque steam locomotives was in business at the Clark Mine, near Allisonia, Va. Its work was done on the surface, although it ran through a small tunnel which also sufficed as its stable. It consisted of a small flat car, carrying an upright boiler, a small horizontal boiler-feed pump, and a small horizontal engine. The latter was geared to one car axle. This locomotive answered every requirement and could whistle with the best of them.

In the anthracite mines of Pennsylvania there were in 1900, 365 steam locomotives, 30 air locomotives, and 38 electric locomotives. In 1906 there were 445 steam locomotives, 108 air locomotives, and 223 electric locomotives. To show the general trend toward air and electric haulage the following statistics are given: In 1900, 84.4 per cent. were steam locomotives, 6.9 per cent. air locomotives, and 8.7 per cent. electric locomotives. In 1906 the percentage of steam locomotives in use was 57.3; air locomotives 14; and electric locomotives 28.7. It is interesting to note that while there was an increase of 158 air

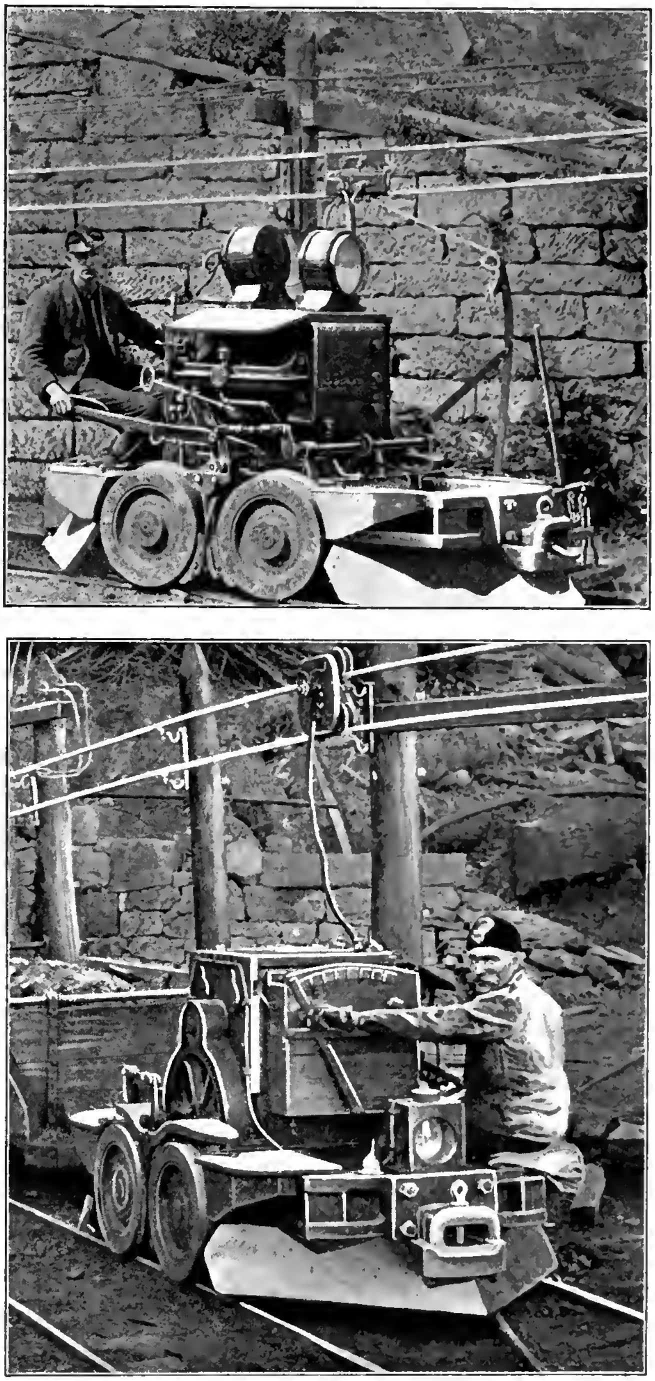


FIG. 58. ELECTRIC LOCOMOTIVES BUILT 1888, STILL IN USE

and steam locomotives, there was also an increase of 185 electric locomotives.

Much can be said from an economical standpoint in favor of steam, air, and electric haulage, but economy cannot always enter into the subject, so that comparative statements given for any one mine are not so absolute as the conditions that govern their use. For this reason it is stated that each kind of locomotive may be better adapted to a certain mine than either of the others.

* See article on "Compressed Haulage," MINES AND MINERALS, Vol. XXI, page 188, November, 1900.

The most critical point in designing a mine locomotive is to make the dimensions a minimum. The haulways in mines are never of more generous dimensions than are necessary, hence the minimum dimensions for mine locomotives are as small as 2-foot wheel base; 8-foot length overall, and 3-foot width. Scarcely two orders carry the same dimensions, and it is necessary for the makers to have a variety of motors suitable for track gauges as narrow as 18 inches, and for wheels 20 inches in diameter. Since the introduction of electric locomotives coal mines are worked to advantage whose beds are 4 feet thick,

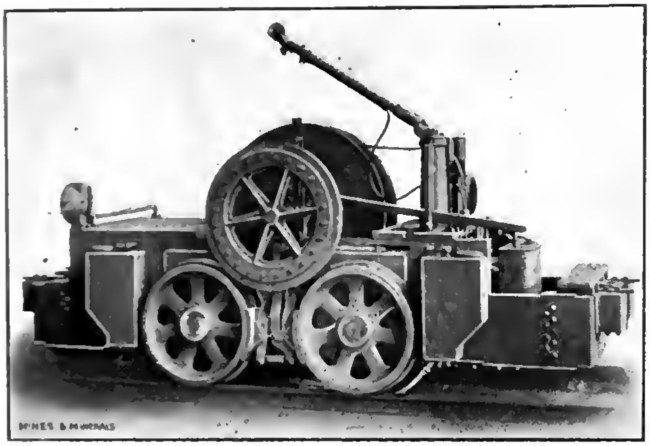


Fig. 59 Wampus Electric Locomotive

and in which mules could not work unless the entries had the roof brushed. Such mines are now competing successfully with mines whose coal beds are 6 and 7 feet thick.

The introduction of electric locomotives has improved the rolling stock, and where formerly it was customary to find as much as 60-pounds per ton car resistance on the level, at present it is found as low as 8 pounds. The several electric locomotive manufacturers have taken advantage of the weak and strong points developed in practice until there are as many styles and constructions as one could desire. The frames have been changed, the motors improved until it would seem as if nothing were left to change, and yet one of the latest improvements, Fig. 55, is to couple the wheels with side rods and use driving axle cranks

The first electric locomotive was built in 1887 for the Lykens Valley colliery of the Pennsylvania Railroad Co. Since that time very rapid progress has been made until the modern locomotive bears little resemblance to the Pioneer shown in Fig. 56.

The locomotive shown in Fig. 57 was built for the Hillside Coal and Iron Co. in 1889, and was still in use in 1910 It does not resemble the earlier one in any respect

The two electric locomotives shown in Fig. 58 were the first built by the Jeffrey Mfg. Co. and were the first ever used in a bituminous coal mine. They were built in 1888, and are still in use at the XX mine near Shawnee, Ohio

For a number of years the manufacturers of electric locomotives have endeavored to eliminate the mule as a car distributor and collector. The first attempts were made with storage-battery locomotives, after which reels carrying wire cables were placed at the rear end of the small locomotives.

The Wampus electric locomotive, shown in Fig. 59, is a gathering locomotive supplied with a reel for a double cable.

Fig. 60. Electric Gathering Locomotive

The reel is turned to wind and unwind the cable by a friction wheel which can be dropped on a drive wheel of the motor.

The Pocahontas Collieries Co., at Pocahontas, Va., has 12 of these machines in use, that seem to be giving general satisfaction. The gathering cable reel is set between the axles with the reel center several inches above the top of the locomotive frame. When it is desired to enter a room, the cable is attached to the trolley wire in the entry and to one rail. The lever shown is then pushed up, which allows the friction wheel to come in contact with the driving wheels and turn the reel at a slightly greater speed than the speed at which the locomotive moves, thus insuring the cable winding on the smallest diameter.

This arrangement can only be used in thick-bedded coal seams, and to work in thin beds the drums are placed on the body of the locomotive, as shown in Fig. 60.

In some cases it is considered better to carry a rope into the room face and by means of a motor mounted on the electric locomotive in the entry pull the cars to the entry. In such cases traction reels are run by a small independent motor connected with them by gearing.

Fig. 61 Jeffrey Storage Battery Locomotive

Fig. 62 shows an electric locomotive on the entry with a man unwinding and hauling a small $\frac{3}{8}$-inch steel rope from a drum into a room in order to attach the rope to the car to be pulled out. The reel, which contains 300 feet of rope, is mounted vertically at the rear end of the locomotive, and can be used for pulling cars to the face of

sloping rooms as well as pulling them out of flat rooms. If there is considerable up grade from the entry, the rope may be run through a pulley at the face and the car hauled up by the locomotive. Fig. 63 shows the traction reel pulling a loaded car from a room.

Where the hauls are not too long and the tracks are approximately level, storage-battery locomotives are sometimes used.

Fig. 62. Taking Hauling Rope Into Room

These locomotives weigh from 2½ to 7 tons and cost from $1,800 to $4,000, depending upon their size.

Fig. 61 shows a 12-horsepower locomotive equipped with series-wound, railway-type motor, and a 42-cell, 16-kilowatt-hour battery, which in service will give an operating range of about 300 ton-miles on a single charge when the tracks are approximately level. The locomotive shown was constructed for hauling material excavated in the extension of the Comstock tunnel.

The economy in actual cost of operation is often secondary to the saving that results in several departments from promptness in handling material, particularly in tunnel driving.

If frictional resistance be neglected the pulling force necessary to draw a load up a grade of given percentage is equal to that same percentage of the weight of the load. That is, a 20,000-pound locomotive has its draw-bar pull decreased 400 pounds on a 2-per-cent. grade. The draw-bar pull for a traction locomotive is taken at from 20 to 25 per cent. of the weight on the drivers, and if there be a deduction of 2 per cent. of the weight of the locomotive from the draw-bar pull in ascending a 2-per-cent. grade, there is a total deduction of from 8 to 10 per cent. of the draw-bar pull. In mine haulage, the train resistance on level roads varies according to the condition of the track, cars, and lubrication, but is generally between 1 and 2 per cent. of the gross weight.

When the increased draw-bar pull required by the train is considered the results are still more serious. Take for example a traction locomotive weighing 30,000 pounds; at 20 per cent. rating its draw-bar pull on a level track is 6,000 pounds. Assuming a train resistance of 2 per cent., this draw-bar pull provides for hauling a train of 300,000 pounds. On a 2-per-cent. grade the locomotive draw-bar pull is reduced to 5,400 pounds, and the total train resistance is increased by 2 per cent. of the weight of the train, becoming 4 per cent. The locomotive now can haul $5,400 \div .04 = 135,000$ pounds, which is a loss of 55 per cent. compared to what it can haul on a level track.

Traction locomotives may be used on short grades of 5 per cent., but above that they are not to be considered. The Goodman rack haulage system is able to haul a certain trip of cars over a given roadway regardless of variations in track or car resistance, conditions that change from day to day to affect traction locomotives.

Fig. 64 shows a Goodman plain rack locomotive climbing a 16-per-cent. grade with 25-ton trips. Since traction locomotive wheels will slip under certain conditions, embracing bad rails, grades, and loaded cars, they are made heavier on this account than they would need to be otherwise. With a rack rail into which driving sprockets mesh for positive pulling, there is no slipping, and hence the locomotive weight can be greatly reduced and retain the same draw-bar pull. The essentials in the Goodman rack-haulage system are: A strong and durable rack rail, supported by the track ties and securely anchored to them. A powerful electric locomotive whose motor drives steel sprocket wheels attached to the axles and which engage the rack rail to produce the forward motion.

The rack rail may be either live or dead. In Fig. 64 the locomotive is working on a live rack rail, which is also the conductor for the power current.

In Fig. 65 is shown a rack rail locomotive which receives its power from a trolley. The Goodman company makes also a combination rack and traction locomotive which is claimed to be preferable in haulage work where there are prevailing or continuous grades, even though they be quite within the practicable or possible limitations of traction locomotives. The combination rack and track locomotive is arranged to run as a rack locomotive on grades, and a traction locomotive on level stretches where no rack rail is needed.

Once the opinion generally prevailed that gasoline locomotives were not suitable for underground work. The Germans thought differently and have been so successful with them that they are being considered in England and the United States. In the August, 1910, issue of Mines and Minerals a general illustrated description of an internal combustion locomotive was given.

There are probably some methods of mine haulage which have been overlooked in this article, because every kind of animal or machine large and tractable enough has been used for the purpose. We might have alluded to horse whims, and water wheels, but these are reserved for another article. The subject of mine haulage has been treated as one of evolution, but in practice the subject includes items which require the skill demanded of mining engineers. For instance, the extent of the haulage, the output of the mine, the grade and direction of the haulage road, the gauge of the track, the construction of the

Fig. 63. Pulling Car Out of Room

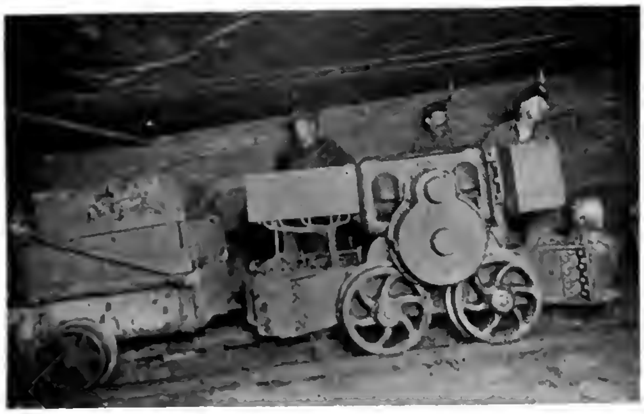


Fig. 64. Plain Rack Locomotive on 16-Per-Cent. Grade

track, with turnouts, curves, switches, etc. Then it embraces the choice of haulage system, whether man, animal, or power or all three, which will of course depend on the mineral, the extent of the mine and its capacity. Each system will require thoughtful consideration and engineering skill, which, if not difficult to the trained man, will at least make him do some calculating to get things right. Take, for instance, an ordinary coal mine. The road is surveyed; the grades and the curves calculated and laid out; the number of ties and the number of tons of rails with fishplates figured. This is not all, the road must be ballasted; the ties centered; the rails gauged, spiked, and joined; the ties tamped and the road lined up. After these preliminaries have been finished the haulage system adopted will soon show weak points that will demand attention. When, however, matters are once working right as far as roadbed is concerned, the rolling stock will require attention, and repairs commence. Mine haulage is a more difficult problem to solve than surface railroad haulage, because the wear is more on the rolling stock, the wrecks more frequent, and darkness enshrouds it all, unless, as at large operations, the haulways are lighted by incandescent lights.

Fig. 65. Goodman Combination Rack and Traction Locomotive

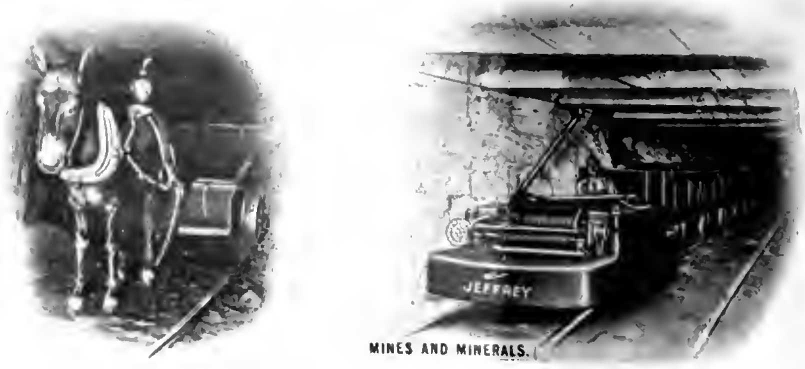
FULMINATING *VS.* WHITE PHOSPHORUS FOR IGNITERS

In *Annales des Mines* Mr. G. Chesneau gives the results of a series of experiments made after the accident at the Lievin colliery to determine the relative safety of fulminating igniters and white phosphorus igniters.

This series of experiments shows conclusively that when lamps, fitted with fulminating-pellet igniters, are introduced in a lighted state into an explosive atmosphere, or are lighted therein, an explosion of the external gases can be caused by particles of the pellet traversing the gauze or deposited on the gauzes and then dislodged by shock. It also indicates the possibility of this phenomenon being reproduced in the course of testing in the mine for firedamp, all the conditions, heated state of the gauze, a stationary explosive mixture, shock caused by the lamp touching the walls of the recess, being present simultaneously. The number of projected particles found after once lighting the lamps shows that even scrupulous care in cleaning the lamps is no guarantee of security, the only effective precaution being to abolish the use of fulminating pellets unless they can be manufactured in such a way as to entirely prevent the projection of unconsumed particles. Further tests point to the danger of firedamp explosions by unconsumed particles of the igniting pellets being greater than the risk of a similar result ensuing from the "Marsaut effect," i. e., the expansion of the flame produced by the explosion of the gaseous mixture in the inner gauze.

There is a fundamental difference between the fulminating and the phosphorus pellet, as the particles from the former are hard and able to pass through a mesh of their own size, while the particles of the latter are viscous and stick to the wire gauze, only the extremely fine ones passing through, and it was only with coarse, unsifted particles that an ignition of gas could be obtained.

These experiments fully confirm the results of previous workers who have tested the phosphorus igniters, and enable the author to recommend the adoption of the latter. This is the more welcome since, without special lighting devices, it would be impossible to retain lamps burning light mineral oil, which lamps—apart from their greater tendency to blow out in a draft—present considerable advantages over colza-oil lamps, both in respect to constancy of illuminating power throughout a whole shift and of their greater sensitiveness in detecting traces of firedamp (1 per cent., as compared with 2½ to 3 per cent. in the case of the oil lamp). The substitution of white phosphorus igniters for those using fulminating caps should not be an expensive operation, at least for the large majority of lamps fitted with vertical igniter boxes and lateral operating mechanism. There is very little difference in the size of the boxes, and it should not be difficult to arrange for the phosphorus igniter boxes to be made of the same size as the others and operated in the same way and thus one of the dangers from igniters would be avoided.



As It Was *And Is*

Fig. 66. Mine Haulage

WORKING A STEEP COAL SEAM

*By Austin Y. Hoy**

The Pacific Coast Co., of Seattle, owns and operates through subsidiary companies (The Pacific Coast Steamship Co., Pacific Coast Railway, and Columbia & Puget Sound Railroad) over 26 steamships in the coastwise trade with Alaska, British Columbia, and California points, and about 165 miles of railroads. This company built the first railroad in the state of Washington, 38 years ago, and is today one of the largest concerns of the Pacific Northwest.

Mines of the Pacific Coast Coal Co., at Coal Creek, Washington

The operation of these steamers and railways requires a large amount of coal. When the Alaska trade is in full swing, over 3,000 tons daily are used to supply the demands of this traffic.

The Pacific Coast Coal Co. (another subsidiary incorporation) owns and operates coal mines at Franklin, Black Diamond, Coal Creek, and Burnett, all less than 30 miles south and east of Seattle. The Coal Creek Mine adjoins the old Newcastle Mine, which was opened some 30 odd years ago, worked to a depth of 2,000 feet, to the boundaries of the property, and is now closed down.

During the last decade, the production of coal has increased so greatly in the United States that the employment of a large percentage of unskilled men as miners has been necessary. Since native-born Americans rarely become coal miners, this labor has been supplied largely by immigration from Italy and the Slav countries, where practically no coal is mined. These circumstances have necessitated the extensive use of machines for doing the work formerly done by skilled miners; namely, "mining" the face of the coal, so as to avoid "shooting off the solid" and the consequent production of great quantities of slack and unmarketable sizes of coal.

Chain machines and compressed-air "punchers" have solved this question under the conditions existing in most of the coal fields of the United States, but in some places the seams pitch so sharply, see Fig. 1, as to render the use of these types out of the question.

Fig. 1. Loading Coal from Chute

This was the situation that confronted the Pacific Coast Coal Co. at the Coal Creek Mine. Skilled miners were almost unobtainable, and as the coal is a non-coking, semibituminous variety, adapted for steam and domestic purposes, shooting off the solid made much unsalable product.

Accordingly, over 2 years ago, experiments were begun to increase as much as possible the percentage of lump (which brings about $2 a ton more in the Seattle market than the smaller sizes).

Compressed-air post mining machines of the radial type were installed, and it was found that after a mining was put in with these machines the coal could be sent down the chutes with only a little pick work, and without any powder, except an occasional light shot at a corner or to shoot out a "nigger head." In fact the powder consumption was reduced over 95 per cent.

Fig. 2. Cutting Coal with a Number of Chisels in Bit

The lump coal was increased in this way from about 25 per cent. to about 60 per cent., and the practical elimination of powder made the mine much safer.

The first machines used developed serious mechanical troubles, due to the unusual stresses imposed by frequent falls of coal. About a year ago the company therefore accorded a thorough test to the Sullivan post puncher, and after several months use an order was placed for 30 machines of this type and make.

Figs. 2 and 3 show a "post puncher" in operation in the No. 4 vein of the Coal Creek Mine, where the coal is 4½ feet high, and pitches at an angle of 38 degrees. As this type of mining machine is practically new in American practice, a brief description may be of interest.

The "post puncher" resembles a rock drill in the fact that it is mounted on a post or column and uses bars or steels of various lengths in doing its work. It resembles the "puncher" or pick machine in its ability to mine across the face, or to shear a room from top to bottom. For this purpose the machine and shell are mounted on a gear segment, controlled by a worm and crank. In mining, the operator swings the bit in an arc back and forth across the face. By loosening one nut, the segment may be set in a vertical plane, and a shearing cut made in a similar manner.

This feature is a very great convenience when the coal will not shoot readily unless both mined and sheared. A valuable feature of the "post puncher" is the fact that the machine does not swing around the post, but around the center formed by the socket, in which it is clamped, about 6 inches from the post. It is, therefore, unnecessary to set the column exactly square with the intended mining, but the machine may be set in a few seconds to mine wherever desired, regardless of careless placing of the post. This is a patented feature, possessed by no other similar machine. Dismounting the machine when swinging from the mining to the shearing position is also unnecessary.

The cylinder and valve motion embody other novelties. The front head is solid with the cylinder, thus eliminating the complication of side rods. When the coal is missed, the piston

* Spokane, Washington. Reprinted from Mine and Quarry.

will overrun the front port and stop, thus obviating damage to the front head. This overrunning occurs very seldom with a skilled operator, but the delay attendant on feeding the machine against the coal, so that it will start again, is nothing to the delay due to breakage that would occur eventually if the piston could strike the front head. The cylinder is made of steel to resist the heavy shocks and strains received from falling coal.

Fig. 3. Post Puncher Using 80-inch Extension Cutter

Felt packing of a very durable character and having an adjustable gland, is employed. A combined shell adjustment and stop, to prevent feeding too far, and a rotation without springs, provided with five round steel pins or pawls, are other interesting features of the "post puncher" design.

The exhaust cap is so arranged as to throw the air in any desired direction to avoid raising dust. The extension rods or length bars range from 20 to 100 inches in length, and are tapered at both ends, one to fit the chuck and the other to receive the bit holder. This bit holder has tapered holes for from three to seven chisels or bit points, as shown in Fig. 2. A solid bit may be used instead, depending on the nature of the cutting.

The mining may be put in at any desired height in the

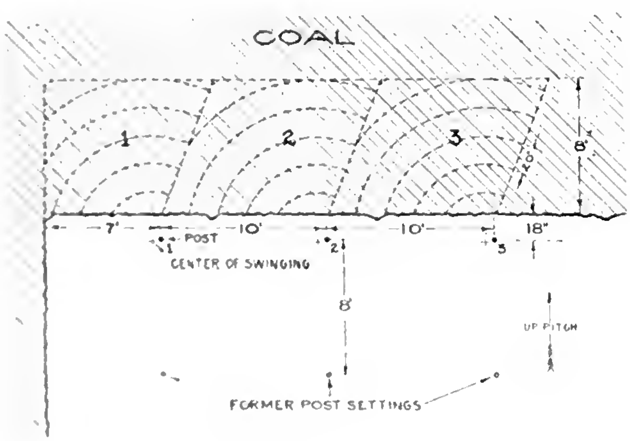


Fig. 4

seam, so as to mine in fireclay, the coal itself, or in a dirt band, by raising or lowering the clamp on the post. In the illustrations the machine is shown mining in a band of bone and clay near the roof, and is accordingly above the segment. In the 6-foot "Muldoon" vein, the impurities are lower in the seam, and the machine is hung below the segment.

Method of Operation.—The rooms are driven 45 to 50 feet wide. The first cut is made from a post set 7 feet from the left rib and about 18 inches from the face, as shown in Fig. 4. A cut 8 feet in depth is put in, using an extension bar 80 inches long. The chuck enters the cut, which accounts for the mining being deeper than the length of the extension. After the 80-inch extension has been swung, a 100-inch bar is used to square up the cut. One man operates the machine, swinging it by means of the worm-crank with one hand, and feeding the cylinder forward two or three turns with the other at each end of the swing.

Although the cuttings fall out of the cut without scraping, due to the pitch, a helper is required to set up, pick down coal, etc. When the operator is completing the left-rib cut, the helper sets up a second post about 10 feet from the first one, or about 17 feet from the left rib, and 18 inches from the face. Upon finishing the rib cut, the machine and swinging attachment are transferred to the second post with only a few minutes delay. While the operator is making the second cut the helper takes the first post down and resets it about 27 feet from the left rib for the third cut, and so on, until the face is crossed. Only a half swing is used, so that the operator is protected by the unmined coal as well as by the machine itself from the coal loosened from the face by the mining.

The machine and posts remain in the room at the face until the room is completed, for there is no shooting of the coal that can injure the machine nor any loading (as in a flat seam) that the machine would interfere with. Hence there is no waste of time due to moving, except from post to post, and little heavy lifting. Two men can set up the machine, see Fig. 5, with ease, as the heaviest parts (the machine and shell) weigh only 225 pounds.

Economic Results.—These machines will average about 300 square feet in an 8-hour shift, or from 45 to 55 tons per machine per shift, according to the height of the coal. While, as stated, the reason for the installation of these machines was solely to increase the proportion of lump coal, even if the cost of mining was increased, the results point strongly toward a material reduction in the cost of mining, after all interest, depreciation, power, pipe line, and maintenance charges have been made against the machines.

In shooting off the solid, the former method, a yardage system of payment was used, the rate being $9.50 for a 50-foot room. Three men worked together, furnishing their own powder. The reason for using a yardage and not a tonnage system was because of the impurities in the seams, and the pitch. The coal and debris go down the chutes together to the slope, there it is loaded in mine cars, Fig. 1, hauled by electric locomotives to the surface and run through a preparing plant of picking tables and washers.

General Mining Features.—The mine where these machines are used is a new one and is worked from a slope 600 feet deep, sunk on the pitch of the seam. Fig. 6 shows the slope bottom of Coal Creek Mine. It adjoins an old mine of the same company, operated through a tunnel. The old mine is nearing exhaustion, but is still furnishing about 600 tons daily.

The new mine is to be worked upon the retreating room-and-pillar system, the entries having nearly reached the boundaries of the property at this writing.

A Sullivan, straight-line, two-stage, steam-driven air compressor with a capacity of 1,843 cubic feet of free air per minute has just been installed at these mines, and a new pipe line laid. The management now plans to put the new mine on one-half capacity (600 tons daily), operating about 12 Sullivan post punchers. Later on, when the old mine is exhausted, the compressor and machine plant will be doubled and the production increased to 1,200 tons.

While the use of Sullivan "post punchers" is particularly adapted to heavily pitching seams, they are also useful in flat seams, where it is desirable to mine in a dirt band above the floor or near the roof. In seams where the cuttings do not fall from the mining by gravity, the helper keeps the cut free by means of a long-handled flat shovel or a scraper.

In some fields these machines have been successfully used for driving entries, and for shearing they can hardly be excelled, especially when both mining and shearing cuts are required, owing to the quickness and ease with which the setting may be altered for either purpose.

From the above article it has been noted that the post puncher was used at Coal Creek with only a half swing, and that the posts were therefore set only 10 feet apart. In stronger coal, or in a flatter seam, where the danger to the operator and machine from falling coal is less serious, the machine is used to cut on both sides of the post, from the same setting, having a maximum capacity of about 18 feet of face from one set-up. To keep the ribs or side walls of an entry or room straight, the bit is swung back and forth across the corner, gradually advancing it with the feed-screw until the angle is square. Or a hole may be drilled at the extreme end of the swing to the full length of the feed-screw, then the bit cranked back and the mining made up to this hole at each swing.

Wedging Down Coal.—The post puncher may also be employed for wedging down coal in mines where the use of explosives is prohibited. After undercutting and shearing, the machine drills a hole about 3 inches in diameter at the top of the seam. An ordinary bit may be used for this purpose. The hole is then scraped out and a compound wedge (plug and feathers) inserted. The machine may be swung around the post if necessary to give access to the hole. When the plug and feathers are placed, the machine is swung back to position opposite the hole and a special hammer inserted in the chuck. The air is gradually turned on, the hammer striking the wedge with increasing force until the coal breaks down.

The writer's thanks for data used in preparing this article are due Mr. J. J. Jones, Superintendent of the Coal Creek Mine, Newcastle, Wash.

THE UNITED STATES BUREAU OF MINES

The act establishing a Bureau of Mines in the Department of the Interior, approved May 16, 1910, became effective July 1. As originally approved, the law contemplated the transfer of the entire Technologic Branch of the United States Geological Survey, the mine accident investigations, fuel investigations, structural materials investigations, the entire personnel, property, and equipment, to the Bureau of Mines, but the Sundry Civil appropriation act, approved June 25, amended the law to such an extent that the structural materials investigations, including the personnel and equipment for these investigations went to the Bureau of Standards, Department of Commerce and Labor.

The Secretary of the Interior has transferred to the Bureau of Mines the investigation of mine accidents and fuels, together with the personnel and equipment of these investigations, and has transferred to the Bureau of Standards the structural materials investigations and the employes of the Technologic Branch of the Survey engaged in these investigations. The Testing Station at Pittsburg also goes to the Bureau of Mines.

The Bureau of Mines therefore includes the mine accidents and fuel investigations for which an appropriation of $410,000 was made. The total appropriations for the Bureau, including salaries, rent, and expenses of removal, amount to $502,002.

The work of the Bureau of Mines for the first year will be a continuation and expansion of the work carried on by the Technologic Branch of the Geological Survey. The law in itself provides for a variety of other problems that properly belong to the Bureau of Mines and which should eventually be undertaken, such as methods of mining and metallurgical processes, but these activities will be deferred for the most part until Congress gives additional authorization in the shape of adequate appropriations. The spirit of the debates in Congress, both on the Bureau of Mines legislation and on the appropriation items, emphasized the desire to regard the mine accident investigations as urgent and this will be the feature of the work.

In all, $310,000 was appropriated for mine accident investigations. Of this sum under the general plans approved by the Secretary of the Interior, $120,000 is to be spent on the rescue stations, $36,000 for housing nine stations; $34,000 for equipping eight new stations; and $10,000 for additional equipment for five existing stations. The allotment for the investigation of explosives is $40,000; for electricity in mining, $14,000; appliances for preventing mine accidents, $8,000; examination and codification of mining laws, $5,000; and other technologic investigations, ore treatment, etc., $10,000. For the analyzing and testing of the coals, lignites, ores, and other mineral fuel substances belonging to or for the use of the United States, $100,000 was appropriated. Of this amount, $35,000 will be spent in the chemical and physical investigation of fuels; $25,000 in the inspection of government fuel purchase; $22,000 in fuel efficiency investigations; $5,000 in lignite and peat investigations; and $4,000 in briquetting investigations.

The publications of the Survey relating to mine and fuel investigations, prepared by the Technologic Branch, will in the future be distributed by the Bureau of Mines and can be obtained by addressing the Director of the Bureau of Mines, Washington, D. C. The publications relating to structural materials will continue to be distributed by the Geological Survey.

Fig. 5. Setting Up a Post Puncher in a 38-Degree Seam

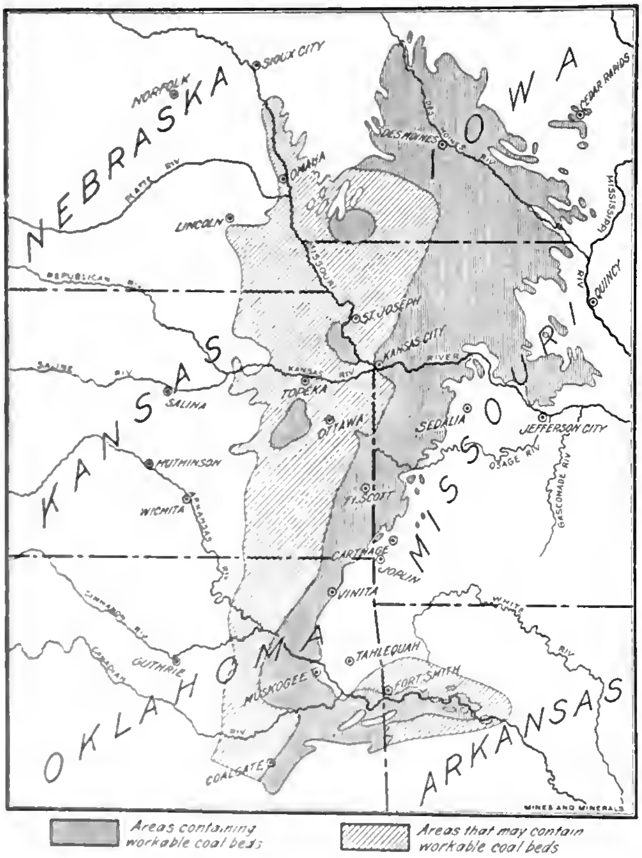
Fig. 6. Bottom of Slope, Coal Creek Mine

COAL FIELDS OF IOWA AND MISSOURI

By Henry Hinds

Extent of the Beds and Quantity of the Coal. Prospecting. Mining Methods

Extending from northern central Iowa south to central Missouri and thence west of south across southeastern Kansas into Oklahoma is the outcrop of Carboniferous strata that constitutes, one of the most important coal reserves of the United States. Of this region, the coal in Iowa and Missouri is essentially a unit as regards its properties and will be briefly described. Estimated on a most conservative basis Iowa contains 12,560, and Missouri 16,700 square miles of coal-bearing strata, making a total of 29,260 square miles; while a large additional area may contain coal of workable thickness. The possibly productive area in the two states may be 40,000 square miles. Deep drilling west of the present producing fields may increase this estimate, for very little is known concerning the westward extension of the coal horizons.



Map of Iowa and Missouri Coal Fields

Structure.—The structural geology of the region is simple in its general outlines: the strata dip at the rate of from 10 to 20 feet per mile to the southwest in Iowa and to the west in Missouri Small gentle folds and normal faults with maximum throws of 8 feet occur, but are of slight importance. Many of the coal beds, however, rise and fall notably, varying in altitude as much as 40 feet within the limits of a single mine. This feature may not be so much structural in character as the result of irregularities of the floor upon which the coal plants grew. The smaller faults, also, are perhaps due more to unequal shrinkage of the coal-forming peat than to earth movements subsequent to deposition. A feature that is in one sense structural is the presence in the coal of highly inclined clay seams, variously known as "troubles," "wants," "clay slips," or "horses." These seams are in general the result of the filling of fissures in the coal by a squeezing of the underlying clay. In places, however, the fissures were apparently made in the drying peat before it was buried beneath later sediments, and were then filled with material washed in from above.

Stratigraphy.—All of the rocks under discussion are classified on lithologic and economic grounds as consisting of an upper group called the Missouri, and a lower known as the Des Moines. The Des Moines group is divided by Iowa geologists into three divisions known as the Pleasanton above, the Appanoose next below, and the Cherokee at the base.* The subdivisions of the Missouri group need not concern us here. Essentially, the same arrangement of strata is present in both Iowa and Missouri, but in the latter state the Pennsylvanian rocks are often known as the "Upper, Middle, and Lower Coal Measures,"† the "Upper, or Barren Measures" being the equivalent of the Missouri group and the "Middle and Lower Measures" representing the Des Moines group. The maximum thickness of the entire Pennsylvanian series is about 1,400 feet in Iowa and 2,000 feet in Missouri and increases toward the southwest in Kansas to 3,000 feet.

The Missouri group is about 950 feet thick‡ and outcrops in the southwestern corner of Iowa and the northwestern of Missouri. It represents deeper sea conditions than does the Des Moines group, containing as it does an abundance of calcareous and sandy shales and a considerable number of limestones, some sufficiently massive as to form conspicuous topographic features. At times during the deposition of this stage shallow water and land conditions prevailed, permitting the formation of thin coal beds. All of the strata, including the coal beds, are persistent and can be traced over large areas. Two of the coal beds, the Nodaway and the Linquist, have been mined in a small way and the Nodaway is of considerable local importance. It is interesting to note that specially favorable conditions make it possible to profitably mine and ship coal from a bed that is only 14 to 22 inches in thickness.

The Des Moines group, the lower division of the Pennsylvanian series, is from 500 to 650 feet in thickness where it outcrops in a broad band east of the Missouri group. It supplies practically the entire coal output of Iowa and Missouri, its lower portion, the Cherokee shale, being by far the best producer in Iowa and an important one in Missouri. The most striking characteristic of the Cherokee is the extreme variability of its constituents, which are chiefly shale with sandstone and coal and a very few thin beds of limestone. When traced horizontally the coal beds or their associated strata thicken, thin, disappear, and reappear with startling rapidity. Coal merges into carbonaceous shale and carbonaceous shale into light-colored clay shale within surprisingly short distances. Many drill records of prospects taken only a few hundred feet apart have so little in common that it is impossible to correlate them, and sections measured on one side of a narrow valley may show little resemblance to those found opposite. The coal lies in lenticular basins of small extent, for the most part underlying, as a single workable bed, less than 500 acres. In many districts, however, the horizon of one basin is continued by other beds at the same stratigraphic level. The Cherokee shale lies for the most part on a very irregular surface of Mississippian limestone and many of the most productive coal basins of Iowa lie in deep valleys that were eroded in the limestone before Pennsylvanian deposition began. In many places Cherokee strata are thus found surrounded on two or more sides by older rocks. The productive basins of this kind lie well within the main outcrop of the Des Moines, but there are

*Lees, J. H., General section of the Des Moines stage of Iowa: Iowa Geological Survey, Vol. XIX, 1909, page 599.

† Bush, B. F., The Coal Fields of Missouri: Transactions American Institute Mining Engineers, Vol. XXXV, 1904, page 905.

‡ Smith, G. L., The Carboniferous Section of Southwestern Iowa: Iowa Geological Survey, Vol. XIX, 1909, page 613.

interesting outliers of probable Cherokee age scattered through both states some distance east of the main coal field and completely surrounded by older formations. Coal found in such situations is of poor quality and very limited lateral extent, though surprising thicknesses are reported from Missouri, in one case as much as 75 feet.*

Between the Cherokee shale and the Missouri group is a series of beds that in every way form a transition from the sediments of marginal type prevailing in the former to those of deeper and more quiet waters found in the latter. Limestone is more common than in the Cherokee, but not so thick or abundant as in the Missouri group. Shales, argillaceous, sandy, and calcareous, are still common, but all the strata are more persistent and regular than in the Cherokee. Fairly persistent coal beds are present at several horizons. These are of economic importance and are becoming more so as the more easily accessible basins of the lower formation approach exhaustion. One remarkable bed deserves special mention because of the persistence of special characters and uniform thickness over an area of approximately 1,500 square miles and because it produces nearly 20 per cent. of Iowa's total output and a notable portion of that of Missouri. This is the Mystic, or Mendota, bed. Throughout the wide extent of territory mentioned it varies but a few inches either way from its normal thickness of 32 inches and almost always bears in its middle part a clay band 2 inches in thickness. A thinner clay band near the base of the coal is not quite so persistent but is usually present. The bed indicates a surprising uniformity of conditions over a large area during the time of its deposition. The persistence in position and extent of three thin beds of limestone above the coal shows that the same uniformity continued over the region for a considerable period.

One of the most important questions before the coal geologists of this country at the present time is the western and northern limit of the workable coal beds of the lower Pennsylvanian in this region. Owing to their slight westward dip the formations do not reach a depth too great for profitable mining very far east of central Kansas and Nebraska, but the expense of prospecting deep coal basins as small as those in the Cherokee shale is almost prohibitive. Not one barren drilling in a district, nor even ten, is sufficient to prove that no coal is present. It is possible that no coal may be found in a number of prospect holes that are sunk in the midst of valuable but small coal basins of the lower Pennsylvanian, as is often proved to be the case in prospecting operations along the productive eastern border. Deep mines in Kansas show that the lower Des Moines group is coal bearing as far west as Leavenworth. Deep drilling in Decatur County, Iowa, described recently by the author,† shows the presence of deep coal farther west than it was previously known to occur in the southern part of the state. Certain geologists have advanced theoretical considerations,‡ however, for doubting the continuation of the coal field toward the west, and it is certain that the shaly facies of the Des Moines does thin in that direction. Under present market conditions southwestern Iowa and northwestern Missouri are not legitimate fields for prospecting, but the distant future may witness large mining operations in these districts. The probability that coal may be found much farther north and northwest than it is known at present is not great. The Carboniferous of northwestern Iowa is effectually concealed by a heavy cover of Cretaceous and glacial material. It is probable that the Pennsylvanian, where present, was largely removed by erosion before the Cretaceous was deposited.

Mining Methods.—The occurrence of the lower coal beds in basins of small extent increases greatly the cost of mining, and more especially that of prospecting. In Iowa and northern Missouri a heavy cover of glacial drift from 50 to 400 feet thick adds still another difficulty, as it must be drilled through before the coal horizons are reached and because it effectually conceals all natural outcrops except along the major streams. The irregularity of the beds renders imperative the thorough prospecting of a field before a shaft can be sunk with safety. The preliminary drilling of a large tract commonly results in the abandoning of about one-third of the mineral options held, and when workable coal is finally located its limits can only be determined by drilling at least every 30 acres. This forms a striking contrast to conditions in the neighboring fields of Illinois, where coal beds are, as a rule, quite persistent.

The coal beds of the Cherokee shale, where mined, are in general 3 to 6 feet in thickness. They have a tendency to "thin to the rise" and "thicken in the swamps," as it is expressively termed, increasing locally to as much as 16 feet, as in Marion County, Iowa. Clay-ironstone concretions are plentiful in these abnormally thick portions of a bed, in places partially destroying its value. Coal beds are almost invariably underlain with clay and overlain with at least a few inches of dark shale known as "slate." Now and then a sandstone comes down to form the immediate roof, or fills shallow channels eroded in the coal swamps in Carboniferous time, forming the so-called "rolls in the roof" and the "faults" of mining phraseology. In some few cases an impure concretionary limestone makes a "boulder" or "niggerhead" roof of exceptional stability. Most of the mines in Cherokee beds find it necessary to employ the room-and-pillar system of mining and to carefully timber the roadways. Coal beds in the upper two formations of the Des Moines group and in the Missouri group are somewhat thinner, commonly ranging from 2 to 4 feet. A limestone cap rock generally lies a few feet above these beds and makes it possible to mine longwall, a system that is growing in favor.

Some mining is done in Missouri by stripping, but the greater part of the coal in both states is taken out through shafts, slopes, and drifts. Shafts are never deep; a depth of 300 feet is exceptional. Owing to the small area to be won from a single shaft, not many mine equipments are expensive, and the life of a mine is seldom greater than 10 years. Of late years more systematic prospecting has made it possible to install larger mines with plants that are modern and complete in every respect. Tail-rope haulage, powerful hoisting engines, self-dumping cages, and other improvements are now the rule rather than the exception. It is to be hoped that the wasteful practice of "shooting off the solid" will decrease in favor.

In quality Iowa and Missouri coal is a low-grade bituminous, with a few occurrences of cannel. The bituminous coal shows a range in calorific value in air-dried samples from 10,000 to 12,000 British thermal units, with an average of about 11,000 British thermal units. Sulphur and ash are extremely variable in amount, but are commonly high, the sulphur averaging about 5 per cent. and the ash 13 per cent. Gas-producer tests have been fairly satisfactory; a test on an Iowa coal required 1.73 pounds of dry coal to produce one electrical horsepower, compared with 4.95 pounds under the steam boiler, a gain of 186 per cent. in efficiency in favor of the producer.* Attempts to coke these coals for commercial purposes have not been very successful; the coke made was not of sufficiently good quality to compete with the better grades from fields farther east, while the high content of sulphur precluded its use in iron furnaces. The amount of sulphur present also renders the coal undesirable for making illuminating gas. For steaming purposes the Iowa and Missouri coal gives good results, and for domestic use some of the beds, notably the block coals of the Mystic bed, compare very favorably with those of Illinois.

M. R. Campbell of the United States Geological Survey estimates that original coal supply of Iowa was 29,160,000,000

* Bush, B. F., Local Cities, page 911.
† Hinds, H., Coal Deposits of Iowa: Iowa Geological Survey, Vol. XIX, pages 247-253.
‡ Winslow, A., The Coal Deposits of Missouri: Missouri Geological Survey, 1891, pages 25-32.

* Wilder, F. A., Fuel Values of Iowa Coals: Iowa Geological Survey, Vol. XIX, page 411.

short tons and that of Missouri 40,000,000,000.† The total for the two states, 69,160,000,000 tons, was 30 per cent. of that for West Virginia. Of this amount there has already been utilized and wasted at the mines only $\frac{4}{10}$ of 1 per cent. The production of Iowa in 1909 was 7,163,000 short tons; that of Missouri 3,787,000 short tons.‡ The total output, 10,950,000 tons, is 23 per cent. of that of West Virginia. Missouri was the first and Iowa the second state west of the Mississippi River to mine coal, the former being credited with nearly 10,000 tons and the latter with 400 in the United States census for 1840. Iowa's output has steadily increased and kept pace with the wonderful growth of the country which it supplies, and it now ranks second only to Colorado among states west of the Mississippi. Missouri has been less fortunate, as the influx of cheap natural gas and fuel oil from Kansas and Oklahoma has made serious inroads on the demand for coal. The output of the state for 1909 was practically the same as that for 1888. A decrease in the supply of oil from the Mid-Continent oil field will witness a corresponding increase in the coal mined in the states of the southern and central parts of the western interior coal region.

Iowa and Missouri coal has made its influence felt but little in markets outside of the states in which it is mined,§ the product is consumed chiefly in districts immediately adjacent to the mines. Coal that leaves the confines of the fields goes west into Nebraska, Kansas, and South Dakota, and north into Minnesota. Toward the west it enters into competition with the large fields of the Rocky Mountains; toward the north it meets coal that comes by water to Duluth from points in Ohio, Pennsylvania, and West Virginia. Very little is shipped as far east as the Mississippi River, because of the better grades supplied by Illinois and eastern fields. Very little goes south of Missouri, though southern Iowa has sent small quantities as far as Oklahoma. Roughly speaking, 35 per cent. of the product is used by the railroads, 45 per cent. for domestic purposes, and 20 per cent. by manufacturing plants. The future growth of the coal industry will be that of the territory which it supplies, augmented by the inception of manufacturing in territory now purely agricultural.

† Mineral Resources, United States, for 1907, United States Geological Survey, 1908, Plate I.
‡ Saward, F. E., The Coal Trade, 1910, pages 68-135.
§ Bain, H. L., The Western Interior Coal Field: Twenty-Second Annual Report, United States Geological Survey, Part III, 1900-1901, page 360.

HOW HE EXPLAINED THE B. T. U.

The increased use of the term "B. T. U." by the buying public has brought out a good story of its introduction in an Indiana town.

It was a Federal institution—and still is. The highbrow in Washington who had drawn up the specification had heard something of B. T. U., and inserted in the specification that each bidder must state the B. T. U. value of the coal proposed to furnish under his bid.

The B. T. U. was a new one on the board of managers of the hospital, whose knowledge was more extensive as to crops, and the best method of keeping the inmates solid for the Congressman who had secured the hospital for that district.

On the day the bids were to be opened the managers gathered about the long table in the board meeting room. The salesmen of 15 bidding concerns were there to see that everything was done according to form, and each determined to bring the contract home for his company.

Each was called upon in turn to tell the merits of his particular coal. Not one failed to discourse learnedly upon the new subject of B. T. U. After it was all over, seemingly, but the actual awarding of the contract, one member of the board waggled his chin whiskers to inquire: "What all this B. T. U. business was about anyhow. I don't know what it means," he admitted, to the relief of the other six members, who were equally ignorant, but not so willing to take the lead in admitting it. "I'd like to have you coal men enlighten me and this board on that subject. You seem to know all about it, but we don't."

The board settled back, relieved of a responsibility and with a hunger for knowledge.

"You, Mr.——," the president said, addressing a salesman at the end of the table. "You tell us what this B. T. U. business is, anyhow."

Not one of the other salesmen in the room envied the honor that had gone to their fellow laborer. He hemmed and hawed and ended by talking all around it, but the rest of the salesmen and the board realized he didn't know what he was talking about. Thus it went down the line. Not one knew the real meaning of B. T. U. until it came the turn of "King Cole"—only that wasn't his real name. He represented a Columbus, Ohio, firm and had a reputation to sustain. His specialty was getting out of embarrassing situations.

He advanced to the center of the group, with a confidence that gave the board assurance that at last a man had been found who would rend the veil.

"Mr. President," he began, "these gentlemen are all wrong. I'll tell you how you can tell the B. T. U. of any car of coal that comes to this institution."

The board was convinced he knew already. The other salesmen were amazed because they were aware that he knew nothing more about it than they did.

"Mr. Secretary, get out your paper and pencil and take down these figures as I give them to you.

"All right, now? When a car comes in here you first look for the serial number. Suppose that is 10,177. Put that down, Mr. Secretary. Then you look on the side of the car for its capacity. That is 60,000 pounds, say. Put that down, Mr. Secretary. Then you multiply those two figures. Have you done that, Mr. Secretary? You got that down?

"Then you go around to the end of the car and look for a number on the brake beam. That's 19,015, say. You divide that into your quotient and that'll give you the B. T. U. of that car of coal. If that doesn't make it any clearer to you I don't know who can."

By that time the room was in an uproar of laughter. The board members realized just at the finish that they were being "jobbed." But it was the psychological moment.

And "King Cole" got the contract!—*E. B. K. in The Coal Trade Journal.*

OBITUARY

WRIGHT S. PRIOR

Wright S. Prior died at Denver, Colo., July 5. He came to Colorado from Worcester, Mass., and had been engaged in the mining business for some years. For the past four years he had been president of the Conqueror Mining Co., at Empire, Colo.

WILLIAM SMURTHWAITE

William Smurthwaite who was engaged in mining for half a century, died in Steubenville, Ohio, on July 12, aged 80 years. Mr. Smurthwaite was born in Philadelphia, Durham County, England, December 19, 1829. At the age of 15 he assisted his father who was deputy foreman. In 1848 he was employed as mine foreman at Quarantine Hill. He emigrated to the United States in 1858, and was employed as mine foreman by the Steubenville Coal and Mining Co. in 1859. A few years later he was appointed superintendent, a position he held for 47 years, retiring at the age of 77. During his entire administration there were but three men killed, and although the mines generate firedamp there was not a miner burned in 40 years. Andrew Roy in his "History of Coal Mining in the United States," pays a tribute to Mr. Smurthwaite's ability as a mine manager.

CORRELLATION THACKER FIELD

Written for Mines and Minerals, by Audley H. Stow

Thacker, Grapevine, Mate Creek, Williamson, and Naugatuck Coal Seams

The "Thacker" seam derives its name from the town of Thacker in the vicinity of which it was first mined. The town in question is located on Tug River, on the Norfolk & Western Railroad, about 14 miles (by rail) east (toward Norfolk, Va.) from Williamson, the county seat of Mingo County, W. Va.

The Thacker seam, as it is originally known, consists of two seams of coal, the interval between which, in the vicinity of Thacker, is so slight that a considerable tonnage has been derived from the mining of the two seams as one.

Fig. 1 shows the Thacker seam, on Thacker and Grapevine creeks, the latter being about ½ mile east of the former. On only a part of the watershed of these two very small streams, however, is it practicable to mine the two seams as one; while going in any direction from the locality mentioned, the interval between the two increases rapidly in thickness, being more usually from 30 to 80 feet.

At Thacker, it is the upper bench, or the upper seam, that gives Thacker coal its high market rating. It is characterized by a middle band of gray splint, and a tough blue slate roof.

On Mate Creek, 4 miles west (toward Columbus) by rail from Thacker, the upper bench of the Thacker seam is mined exclusively (unless perhaps quite recently); on Mate Creek, however, where the holdings of the Red Jacket Consolidated Coal Co. are largely located, the development of the lower bench, is generally fully equal to that of the upper bench, and of equal purity, both being entirely devoid of slate bands or partings. The lower bench is characterized by a sandstone roof.

Owing to the large tonnage now coming from the upper bench of the Thacker seam, as compared with that, if any, from the lower bench, the former is now more generally known as, and will hereinafter be understood to be, the Thacker seam; some years ago, I suggested the Mate Creek seam as an appropriate local name for the lower bench, which has, as far as noticed, been generally accepted.

Fig. 2 illustrates the transition from the sections shown on Thacker Creek, to the two separate seams on Mate Creek. These sections were taken near the mouth of Mate Creek, on the holdings of the Logan Consolidated Coal Co. Farther up Mate Creek, the lower seam becomes clean; the United Thacker Co. having in this territory a large acreage over which the two seams referred to rival each other in thickness and freedom from impurities. Throughout Mingo and Logan counties, W. Va., and that portion of Pike County, Ky., that is drained by the waters of Tug River, the Thacker seam (upper bench or seam), may be said to be the lower limit of the splint coals; that is, the proportion of splint in the Thacker seam, and those overlying it, is much greater than in those underlying the Thacker seam. The seams above are not infrequently solid splint, while much the larger portion of the coal in the lower seams has a bright glassy luster and conchoidal fracture.

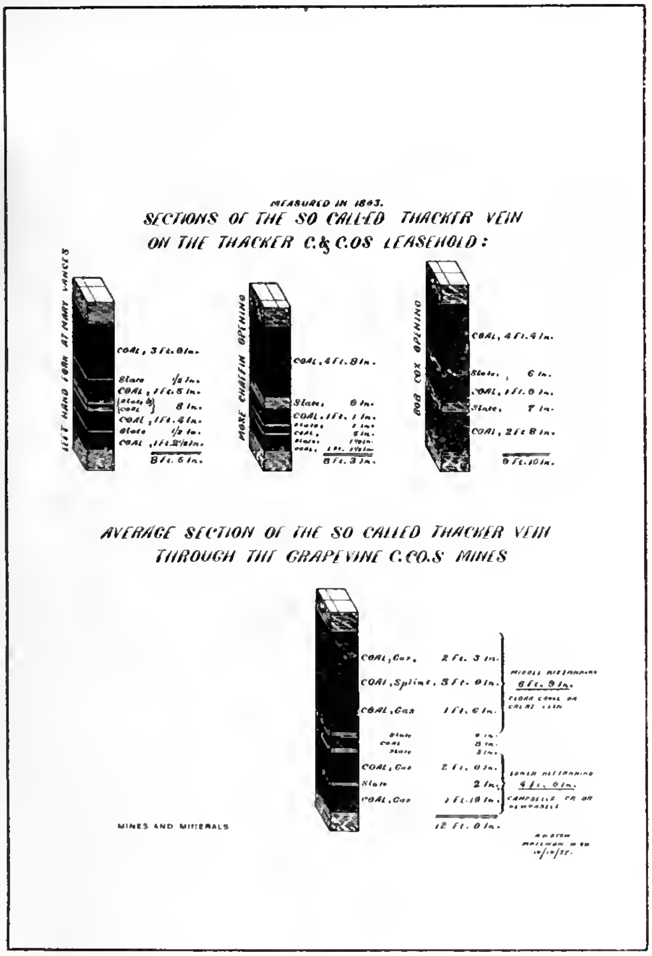


FIG. 1

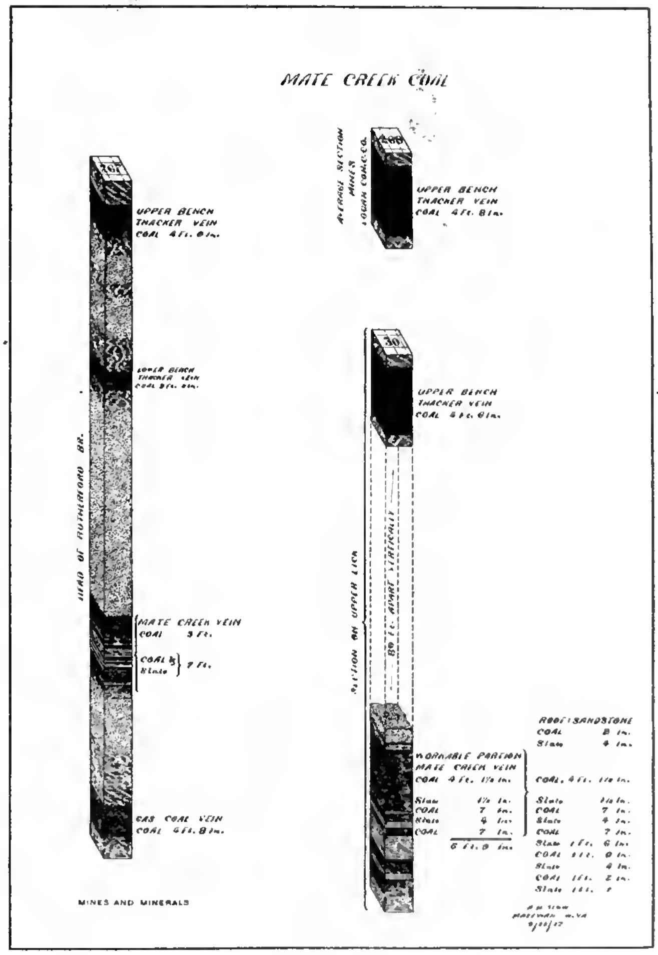


FIG. 2

At an interval of usually 90 feet above the Thacker seam, is one that is of interest, owing to its great persistence and comparatively regular thickness, the roof also being a blue slate. The tonnage in this seam must closely approximate the Thacker seam, and has not as yet been noticed to contain any slate bands or partings whatever. As it is only 3 feet in thickness, it is not,

in comparison with the Thacker seam below, now considered workable. As only too often happens, it is most unfortunate that the relative positions were not reversed, at least in so far as the conservation of natural resources is concerned.

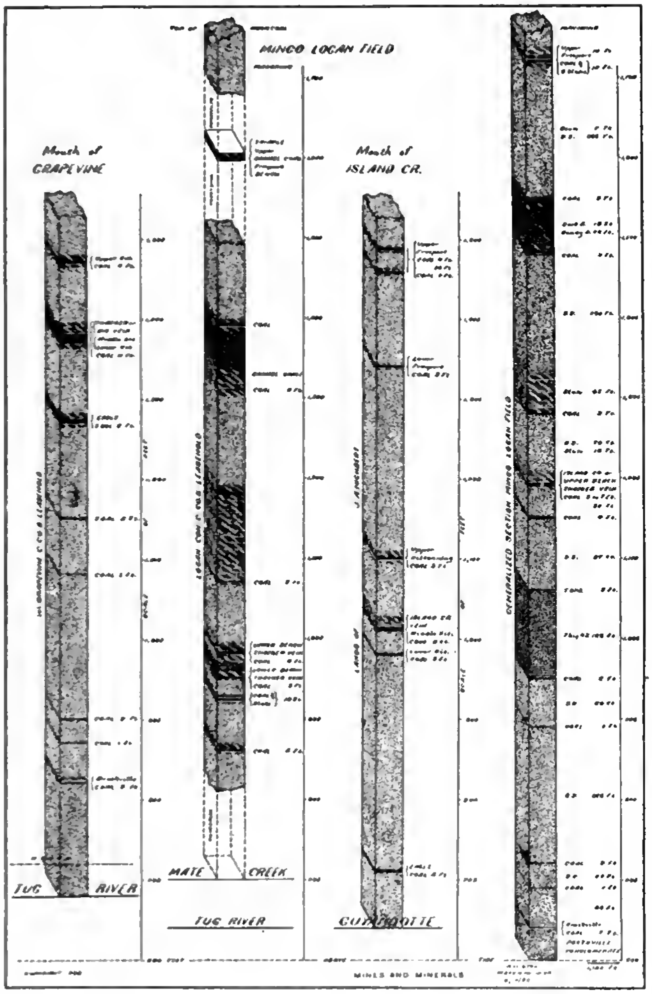


Fig. 3

At the mouth of Grapevine Creek, the Thacker seam is 700 feet vertically above the railroad, whereas at the mouth of Mate Creek, about 5½ miles west by rail, it is only 450 feet, the strata thus dropping, in the distance stated, 250 feet faster than Tug River.

Two miles west of the mouth of Mate Creek, an effort was made, some years ago, to locate the Thacker seam. As the dip of the strata had been determined beyond question, and as the seams above and below were well understood, the problem was easy. The "bench" on which the Thacker seam was due to be found, at an elevation of 350 feet above the railroad, shows only a dirty seam having a band of cannel shale, thus illustrating the relation between splint and cannel coal.

At Williamson, 250 feet above the railroad, the Thacker seam, locally called the Garner seam, shows a total section of 4 feet 3 inches, 6 inches of which is slate, in three separate bands or partings; the roof still being a tough blue slate, while the splint is also still somewhat in evidence. At Nolan, 9 miles west of Williamson by rail, the Thacker seam, at an elevation of 140 feet above the railroad, shows a total section 4 feet 10 inches, 2 inches of which are slate, as before, in three bands.

At Naugatuck, the mouth of Pigeon Creek, 18 miles west by rail of Williamson, is found, at an elevation of 300 feet above the railroad, the upper Freeport or Stockton seam, which has been mistaken for the Thacker seam. The following, although not a detail section, will give an idea of this seam, at the point under consideration:

Roof, Sandstone			
Coal	1′ 10″		
Slate		4″	
Coal	1′		
Slate		1″	
Coal	1′ 7″		
Slate		1′ 10″	
Coal	1′ 6″		
	5′ 11″ +	2′ 3″	= 8′ 2″

The bottom bench, given as 18 inches, more usually appears as 4 to 5 feet total section, somewhat cut up, however, with slate, the total section being more usually 10 to 12 feet. This seam is easily recognized over a considerable extent of country, and is a most convenient horizon for correlation. One of its characteristic features is a considerable band of extremely fine black slates interleaved with thin laminæ of coal which break up into minute flat square blocks, somewhat resembling splint, on fractures at right angles to the seam.

The Freeport, as it is often called even locally, in the vicinity of Dingess, on the Norfolk & Western, at which point the tunnel to the head of Twelve Pole, is located, was measured as follows:

Roof, Sandstone			
Coal, gas, good	1′ 1″		
Slate, tough		4″	3′ 5″
Slate, shelly		5″	
Coal, splinty, good	1′ 7″		
Splint, slaty	1		
Splint, and slates		8″	1′ 8″
Coal, splinty, good	2:		
Slate, shelly		7″	
Blast and splint or bone		4″	5′ 8″
Coal, gas, 3 small bones	2	9″	
	7′ 8″	3′ 1′	10′ 9″

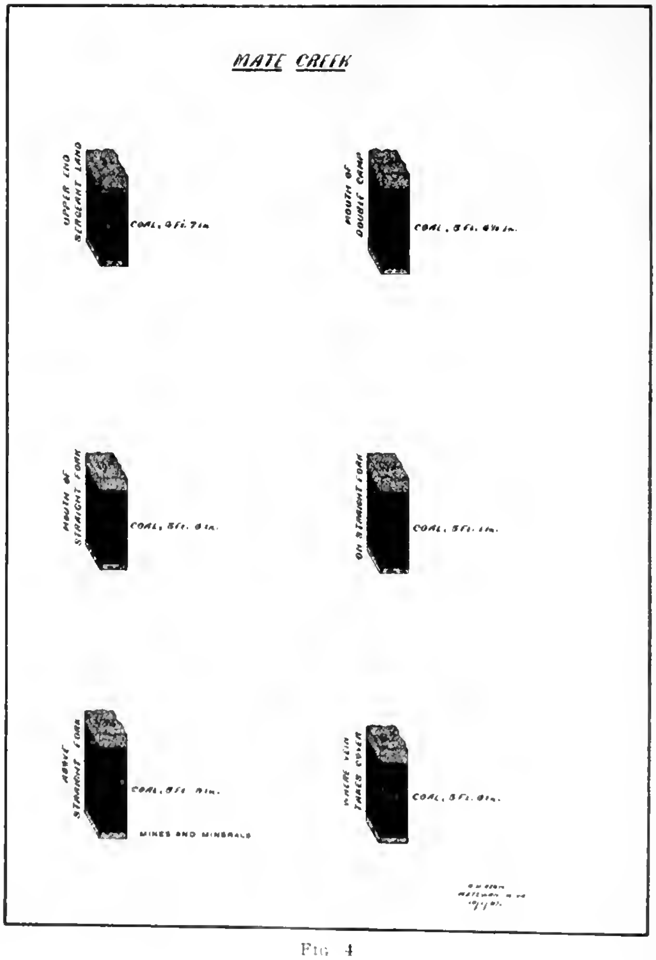


Fig. 4

As will be evident, this seam consists of two main benches, each of which in turn also consists of two benches, making four in all. The 10-foot 9-inch section just given may be said to be a fairly representative section of the seam under consideration.

The interval between Nolan and Naugatuck has not as yet been prospected by myself, so the above chain of evidence may appear defective. As the Warfield anticline is being approached, it might be argued that the Thacker seam rises from an elevation at the former point, of 140 feet above the railroad, to 300 feet at Naugatuck. Yet that such is not the case would appear from a partial exposure, 4 miles west of Williamson, at an elevation of between 600 and 700 feet above the railroad, of what was fairly clearly the Naugatuck, or Upper Freeport, seam given above.

The accompanying generalized sections, Fig. 3, were made in 1897, from aneroid measurements, the exposures being seldom vertically one above the other, the dip of the strata being, however, allowed for. The intervals between the seams are probably not sandstones and shales, exactly as shown, but were so, as nearly as at the time could be determined from natural exposures in the immediate vicinity. The interesting area of Winifrede coal west of Williamson had not at that time been prospected, although the Winifrede and Coalburg seams at the latter point are probably the two seams below that designated as the Upper Freeport, in the generalized section of the Mingo-Logan field. The section on the lands of J. A. Neghtert, is on what is now the holdings of the United States Coal and Oil Co., at Holden, W. Va. The two remaining sections were taken in the holdings of the Red Jacket Consolidated Coal Co.

The intervals shown apply to the territory east of Williamson, to Grapevine Creek of Tug River, and across to Logan, the county seat of Logan County, on the Guyandotte River. From a point 9 miles west of Williamson, going toward the Ohio River, it appears probable that there is a rapid thinning of the measures, the intervals at Naugatuck being probably much less than shown. At the time these sections were made, it appeared preferable to refer them to the Pennsylvania series.

AUTOMATIC COAL SAMPLER

Quite a number of letters have been received by Mines and Minerals, requesting information on automatic coal samplers. After diligent search the one illustrated in Figs. 1 and 2 was furnished by Mr. H. G. Statt, superintendent of motive power, Interborough Rapid Transit Co.

It consists of a spout *a*, attached to the sloping side of a scale hopper *b*. In the spout are two sliding plates *c* and *d* actuated by the levers *e, f, g, h, i* in such a manner as to move one in as the other is moved out, thus cutting out a sample of coal in the spout *a*. There is a reach rod *j* connected with lever *e* and a hand wheel *k*, for the purpose of causing the sliding doors to reciprocate as the wheel revolves.

The metallurgist will see at a glance that the principle on which the sampler is constructed is wrong, and that even a fair average sample of run-of-mine coal could not be taken with it, let alone prepared sizes. If coal is purchased on analysis from this kind of a sampler, it is certainly proof positive that the producer is furnishing an excellently prepared coal. As previously stated, nothing is to be gained by purchasing coal on analysis. Coal producers prepare their coal for market to the best of their ability whatever way it is sold, and in addition have coal inspectors on each car looking for impurities as the car is loaded. Market conditions are such that if the producer does not ship good clean coal he will lose his customers once and for all, because consumers do not care to change coal dealers if it can be avoided.

It may be added that experience has shown that inter-

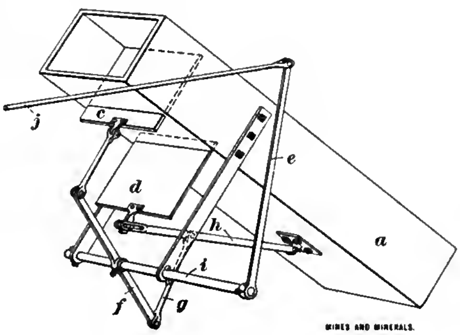


Fig. 1

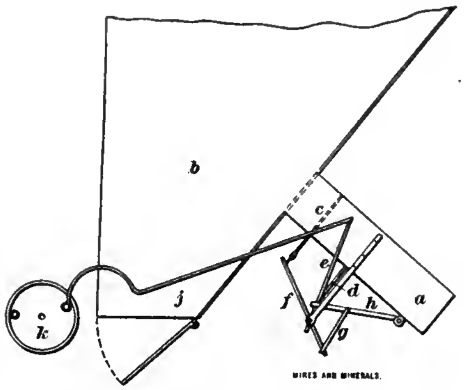


Fig. 2

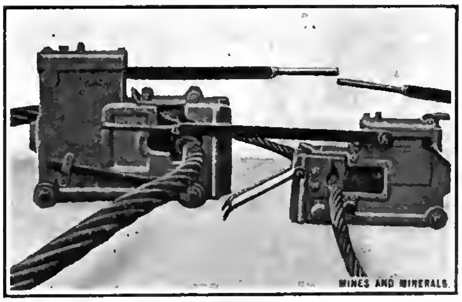
mittent samplers are failures in furnishing fair average samples and since the sampler illustrated is in this class it too must prove a failure.

Most consumers think miners work by the day and are under the eye of a foreman who can inspect their work. For their benefit it is stated that coal miners are contractors who mine by the ton, and while most of them undoubtedly try to clean the coal, some in the hope of its passing inspection are not so particular. The value of a first class coal sampler under these conditions becomes apparent, because if the consumer notices impurities in a single case he is likely to think they extend through the whole shipment. With a good sampler he would know.

A WIRE ROPE CUTTING MACHINE

Written for Mines and Minerals, by Frank C. Perkins

The accompanying illustration shows the construction and method of operation of a novel English mining wire rope cutting machine of the Selby type designed in London for use in collieries and other mines. It is a simple portable device and rapid in its operation, requiring only one man to actuate it in cutting wire ropes up to 9 inches in circumference.



Wire-Rope Cutter

In charging the cutter the ram is moved back and the reservoir filled with clean water with a small quantity of glycerine added, the air screw in the device being opened and the relief valve screwed up when ready to cut the rope. The cover is opened and the rope inserted when the former is replaced and locked. The pump is then operated by means of the lever until the rope is cut by the shears. The relief valve then has its screw slackened in order to return the shears by sliding the lever on to the hexagon spindle and pressing back the same steadily. The operation of the shear by the ram in cutting the wire rope is clearly indicated by the accompanying illustration.

POWER USED IN MINING*

By Edward O'Toole†

When mining first began human beings furnished the power. They broke the coal and loosened it with rude tools fashioned by hand from wood, stone, or iron, often carrying it on their backs, in baskets made from willows or bark from trees, to places where it was used, sometimes it being necessary to carry it up ladders for considerable distances.

Men, Live Stock, Mechanical Power, and Explosives. Payments in Case of Accident

When there became what was considered in those early days a scarcity of labor, animals, such as horses, donkeys, oxen, etc., were brought into use for the purpose of raising and transporting the material mined.

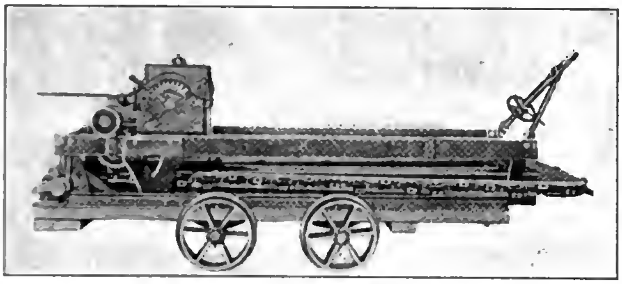
Afterwards explosives were introduced to help break down the materials, hydraulic, steam and electric machinery coming later for various purposes.

This gave six sources of power; namely, labor, animal, explosive, hydraulic, steam, and electricity, the five latter being introduced by the miner to lighten his labor.

This will bring us to the present, when the percentages of power cost, as used by the United States Coal and Coke Co., and the percentages of energy produced, are as follows:

	Cost Per Cent.	Energy Per Cent.
Labor cost per ton	87.6	38
Total live stock expense	4.0	12
Mechanical power	4.7	35
Explosives	3.7	15
	100.0	100

The above shows that while our mines are well equipped with machinery, we are still large employers of labor. This labor is divided into executive, engineering, supervising, laborers, and clerical. The engineers directing what is to be done and how to do it, the supervising department carrying out the ideas of the engineers by directing the laborers and guiding them in the most practical way to perform their task, the laborers performing the task, and the clerk in the accounting department keeping the accounts and distributing the allotted compensations.



Morgan-Gardner Coal-Cutter, Mounted on Truck

Of the various departments, the supervising is the most important, for the reason that the labor is the most expensive item, and therefore, the most important; and a good executive is considered of more importance than a good engineer. As the mechanical appliances around a mine increase, the engineer becomes of more value, particularly is this the case where the property being developed is large and the machinery diversified. When the engineer is thoroughly competent, does the advance thinking, and gives his property the attention it should have, his value is second to none. We appreciate the value of the engineer and employ him possibly more than he is employed by the mining industry generally.

Our supervising department follows the plans and the instructions given by our engineers, but they are encouraged in reporting what, in their minds, will be desirable changes; and they are often able to assist the engineering department with suggestions, otherwise they simply supervise and direct.

The general labor around coal mines is diversified. Our laborers represent 12 nationalities, and as the Americans are divided between whites and negroes, they actually represent 13, which are as follows:

Nationality	Per Cent.
American	17.80
Hungarian	22.20
Slavs	5.80
Russians	3.00
Poles	4.50
German	.50
Italian	13.80
Roumanian	5.50
French	.01
Greek	.06
Spanish	.01
Swede	.02
Negroes	26.80
	100.00

You will notice the absence of the English, Scotch, and Irish, which nationalities in former years were numerous among the rank and file of mine laborers. Where mining has been carried on for a number of years, they or their children are still largely employed, but with us they are only represented in the official classes. Some of the above nationalities are also represented among the officials, particularly the German, Italian, and Hungarian. These people all make very good mine employes, and, if properly instructed, soon recognize the necessity of care and readily adapt themselves to any safety measures introduced for their benefit. Some of the negroes are reckless and are prone to take chances, but this applies to some of the younger ones only, and usually the matured men are ideal miners, capable and trustworthy. They mostly follow the transportation and mechanical departments, run mining machines, motors, etc., and where premiums are offered for the care of the machinery they are often successful in earning them.

Our miners and other laborers around the mines are required to obey the mining law and the rules of the company, which they do fairly well; and while some 2 or 3 years ago, when breaking up the blasting from the solid, it was necessary to have some of them prosecuted, I have heard of no breach of the law or rules since that time. These people are also law abiding and peaceful at their homes or on the streets, and with a population of 10,000 to 12,000 in this mountainous district, we have very little trouble. We employ no police officers except the regularly elected constables and justices of the peace, and they do not have sufficient business to make a living at their office. This, I believe, will compare favorably with any community where any class of people are employed in the same numbers.

The other, or accounting department, needs no comment, it being the same as the clerical department of any mining company.

Source of Power.—The introduction of explosives in coal mining was very unfortunate, as there is hardly a mining community that has not suffered from a disaster which is directly traceable to them, and of late years, as their use has increased, the disasters have become more frequent; and in communities where their use has been unrestricted, it being left to the judgment of the individual miner, the results have been appalling. Fortunately for those engaged in coal mining, the government is now testing explosives and giving their approval to those that will justify it, after a public demonstration. The powder they approve they term a "permitted explosive," and if the people will use this permitted explosive in limited quantities at each blast we will surely have fewer disasters, and the fatalities and injuries to employes will be greatly reduced. All of the mining countries in Europe have what is known as a charge limit, and in Belgium the use of explosives is not permitted in gaseous mines under any circumstances or for any purpose.

*Paper prepared for Summer Meeting of West Virginia Coal Mining Institute.

†General Superintendent United States Coal and Coke Co., Gary, W. Va.

Explosives are undoubtedly of great service to the miner and assist him materially in getting his coal, but their use would better be stopped entirely than to go on at its present cost to human life.

We use only permitted explosives in our mines, or powder that has passed the government testing station. The miners seem to prefer those listed under the name of Colliers or Monobel, which are recommended and sold to us as a powder that will not deteriorate and can be bought in carload lots, which is quite an item to us, situated as we are, so far from the source of supply. The only requirements we make of the powder people is that the powder will sell to the miner at 5 cents per stick. Our charge limit for these powders is 1½ pounds.

Baldwin-Westinghouse Electric Gathering Locomotive With Electric Cable Reel

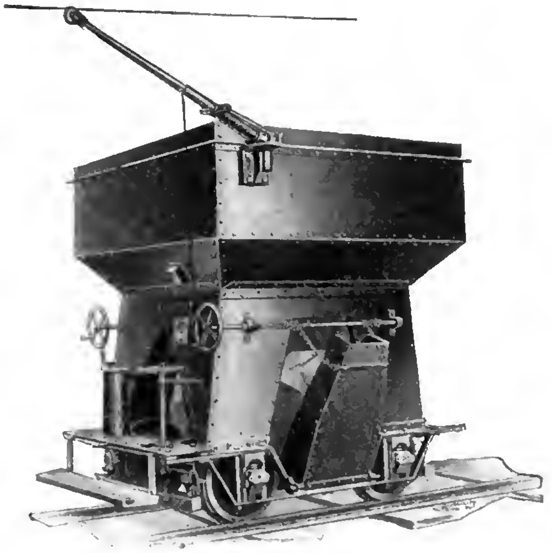
Animal power is also largely used in our mines, equaling 4 per cent. of the total power cost, while the energy produced is about 12 per cent. We have made some efforts to introduce mechanical power to displace the mule, but so far, for certain services, he has proved indispensable. The mule is the principal animal power, but horses are used in some of our mines.

Mechanical power will undoubtedly be the power of the future; and while the cost of the mechanical power used only represents 4.7 per cent. of the total cost of mining coal, we supply mechanically more than 35 per cent. of the total energy required. This would certainly justify us in endeavoring to increase the use of mechanical power at the expense of power of any other kind. The introduction of machinery into coal mining has come by numerous ways. It was first used to raise material out of shaft workings, then water, then as a transportation and ventilating power, and last, to undermine the coal. The machines used for the different purposes have been subjected to a series of modifications; and while some have been simplified, some have been made more complicated as their uses have been extended, until at present, we have the self-propelling under- and over-cutting mining machine, the electric and compressed-air haulage motors, which operate with equal facility both on the surface and in the mine, rotary pumps, and electric-driven, self-starting, ventilating fans. We also have extended systems of underground railroads or haulageways, in some instances laid with rails which weigh 90 pounds to the yard; vast numbers of mine cars, and other equipment too numerous to mention, which cost money to install and operate; but the cost is insignificant when compared with the cost of labor, and it would be utterly impossible to find labor to produce half our present tonnage should we dispense with the use of mining machines for undercutting the coal and the explosives for blasting it down. In fact, with all our machinery, there never was a time in the history of mining when labor was as scarce as it is today. This being the case, it behooves us who wish to stay in the mining business to find more ways and means of using machinery. We must also find ways of improving the machinery we now have to make it more effective. We must find more direct methods of applying our power. We must eliminate all useless operations and roundabout ways; go to the portions of the mining operations which now require the manual labor and apply mechanical power to the same in our efforts to reduce the cost of labor per ton.

At the present time we are developing, and have partly developed, a system of mining and handling coal, which we think will eliminate a large portion of labor now used, particularly where the coal mined is to be used for the manufacturing of coke or the raising of steam, where the firing can be done by means of mechanical stokers. We have accomplished this by the use of a machine which cuts out the total thickness of the seam, removing the coal as it is cut by the machine to the outside by means of currents of air at high velocities, on the order of the pneumatic carpet cleaner. We have termed this method of handling coal "The Pneumatic Transportation." We have had an experimental plant in operation at our No. 9 mine for a period of about 6 months, and are now through with our experiments, having gathered all the data we consider it necessary to have for the establishing of a commercial operation. We will operate this plant for the benefit of the Mining Institute, and after said operation the experimental plant will be partly dismantled.

In the changing of mining from hand power to mechanical power, both for mining and transportation purposes, we have largely increased the number of accidents, not only per 1,000 men employed, but per 1,000 tons of coal mined as well. This shows clearly that the introduction of machinery so far has not paid when calculated on the basis of accidents. This increase of accidents applies to the mines in which no machinery is used, as well as to the machine mines, and it is undoubtedly caused by the changes in the occupation of the men in the machine mine, and to the increased amount of coal they endeavor to get in the hand mines to compete with the machine mines.

This increase of accidents cannot go on, as it is now much higher in this country than in Europe, I will not use the general term, "Where the mining is more hazardous," but say instead, where the conditions of labor are more severe and exacting; neither will I attribute it to our less skilled labor, but rather blame our less rigid discipline, freedom, and variable methods of operation. There is an awakening; the public is being aroused to a sense of responsibility in the matter of industrial accidents, and to the amount of property and distress occasioned by them, not only in coal mines, but in every other industry, and will gladly pay the few additional cents per ton necessary to permit the business to be conducted in a more safe and humane manner, and whatever additional cost is necessary to alleviate the suffering and distress caused by unavoidable accidents. I wish to assure you and the public at large that


Scottdale Electric Coke-Oven Charging Larry

there is no coal operation but what will gladly sacrifice its dividends, if by so doing it can reduce the number of accidents in and around the mine.

To reduce these accidents it will be necessary for the miners to become more thoroughly familiar with the machinery and with the conditions created by its use, and to protect the machines with guards of different kinds so that accidents from the same will be impossible. Where coal is still mined by hand, it will be necessary to revert to the old method of mining; namely, the undercutting of coal must be done as formerly, so that large charges of powder will be unnecessary. In both cases the supervision will have to be more thorough and vigilant than it has been heretofore.

H. K. Porter Co. Compressed-Air Mine Locomotive

I would suggest increasing the number of district mine foremen, so that the working places can be visited three to four times each day, and that some means be adopted to insure his attention to business, particularly in regard to accidents. In addition, I would suggest each company putting accidents on a financial basis, which would take the nature of an insurance or assurance to the victim, and that in case of accidents, whether fatal, serious, or minor, for that matter, a stipulated amount will be received by him, sufficient at least to furnish the necessaries of life during the disablement period, and in cases of permanent or fatal accidents a stipulated sum.

The following is the basis on which the United States Coal and Coke Co., as well as all other constituent companies of the steel corporation, award payments in case of accidents:

TEMPORARY DISABLEMENT

Single Men.—Single men who have been 5 years or less in the service of the company shall receive 35 per cent. of the daily wages they were receiving at the time of the accident. Single men of more than 5 years service shall receive an additional 2 per cent. for each year of service over 5 years. But in no case shall single men receive more than $1.50 per day.

Married Men.—Married men living with their families, who have been in the service of the company 5 years or less, shall receive 50 per cent. of the daily wages they are receiving at the time of the accident. For each additional year of service above 5 years 2 per cent. shall be added to the relief. For each child under 16 years 5 per cent. shall be added to the relief. But in no case shall this relief exceed $2 per day for married men.

PERMANENT DISABLEMENT

For the loss of a hand, 12 months wages.
For the loss of an arm, 18 months wages.
For the loss of a foot 9 months wages.
For the loss of a leg, 12 months wages.
For the loss of one eye, 6 months wages.

FATAL ACCIDENTS

In the case of married men living with their families, who have been in the service of the company 5 years or less, and leave widows, or children under 16 years of age, the company will pay relief to the amount equal to 18 months wages of the deceased employe. For each additional year of service above 5 years, 3 per cent. shall be added to this relief. For each child under 16 years, 10 per cent. shall be added to this relief.

In addition to the above, we have adopted a system of paying premiums for the elimination of accidents.

The ratio of accidents for the first 4 months of the year 1910, in our mines, is as follows:

Fatal

	Per Cent.
Falls of slate and coal	83.3
Mine cars and motors	16.7
	100.0

Injuries

	Per Cent.
Falls of slate and coal	47.2
Mine cars and motors	38.9
Blasting coal	2.8
Mining machines	2.8
Miscellaneous	8.3
	100.0

This shows very plainly, under our present method of mining, that the large percentage of accidents come immediately under the supervision of the minor officials of the mine, and can be materially reduced if these officials give slate and loose coal their personal attention; and I am glad to inform you that I believe we are now on the right road to eliminate a large number of the accidents at our mine. Our officials are beginning to understand that there is something expected of them besides getting cheap coal.

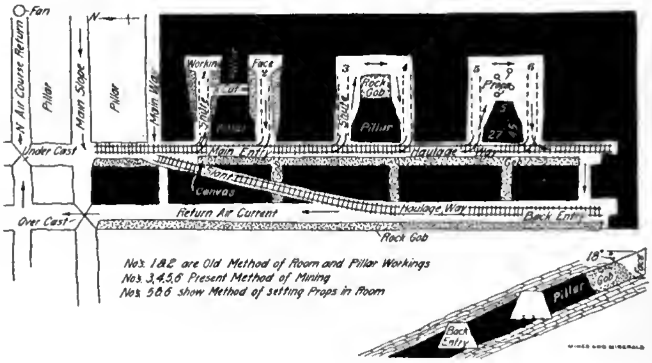
I thank you, gentlemen, for your kind attention, and I hope that something I have said may assist you in not only producing your coal more cheaply in dollars and cents, but that you will use your best efforts to bring the mining accident record of West Virginia, not only up to the record of any state in the Union, but the equal of any in the world. So be it.

A ROOM-AND-PILLAR METHOD

Written for Mines and Minerals, by A. E. Robinson

The No. 4 mine of the Kemmerer Coal Co. has three openings, and is ventilated by a small exhaust fan capable of 33,000 cubic feet of intake per minute. The mine has 5 feet 6 inches of coal on an 18-inch pitch, with a soapstone covering averaging 15 inches thick, which at present has to be taken down and gobbed.

The roof above this covering is good, and the system of working the mine is by room-and-pillar method. At first, as indicated by sketch, it was worked by single rooms with 55-foot centers, with chutes placed in the centers of rooms, and the rock covering gobbed on both sides of the chutes. At present it is being worked with much better success by a double-room system, the two rooms being broken away from the entry by two openings at a distance of 45 feet from the outby rib to the inby rib At a distance of 45 feet up the rooms, the two rooms are joined together, making a 45-foot working face with two chutes, and the rock is gobbed in the center of the double room, tight up to the roof, thus forming an air-course or a "wind pack," so that fresh air sweeps the working face of each set of rooms. No crosscuts are needed as each set of rooms has its own circuit of air, which is obtained by throwing a brattice cloth across the entry at each three sets of rooms, affording ample ventilation for the miners.



Room-and-Pillar Method

The entries are worked on the butt-entry system, driven 9 feet wide; the rock taken down over the coal is gobbed in the lower side of the entry after a butt shot has been taken out to make place for the rock.

COAL AND COKE SAMPLING

*By E. G. Bailey**

Much money is wasted by having non-representative samples of coal analyzed; unwarranted complaints are often based upon analyses made from such samples; and sometimes unjust premiums are paid or penalties exacted, due to the samples not truly representing the coal from which they were taken.

Taking the Sample. Quantity and Method of Reduction. Sampling Mine and Railroad Cars

Properly taken samples of coal are of great value, for the correct analyses of them assist the coal operator to intelligently direct the working of his mines and the preparation of coal or coke being produced; they aid the sales department in placing coal where it will give best satisfaction; and they make it possible for the consumer of coal to buy intelligently and secure the fuel best adapted to his needs, but non-representative samples are worse than valueless under all circumstances.

The variety of conditions under which coal and coke must be sampled makes it quite impossible to follow any one method without some deviation; however, there are a few general principles, which, if understood and practiced, will greatly increase the value of samples taken.

Coal should be sampled:

1. By taking equal increments from a great many equal parts of the original quantity.
2. By taking increments of such size that the largest lumps may be included.
3. In such a manner that a true proportion of all sizes will be secured.
4. Blindly, so far as slate and impurities are concerned.
5. By taking such a quantity that the largest pieces of slate or impurities will be insignificant in proportion to it.

The original sample should be reduced in such a manner that:

1. The part discarded is exactly like that retained.
2. There will be no loss or gain in moisture.
3. No dust or other material will be blown away or lost through cracks in the floor.
4. No foreign matter will get into it.

The above fundamental principles should be followed whether the coal is sampled in the mine, as it is being burned, or during any intermediate stage of its transportation, but the detailed method of applying them will naturally vary with conditions. Some of the principles are self-evidently essential, while others may at first appear to be unnecessary precautions.

Taking Original Sample.—There may be a great difference in the quality of coal in different parts of a car, cargo, or pile, therefore if a sample is to represent any of these quantities of coal it should be accumulated from enough different parts that it will represent the average, rather than merely the coal which happens to be most available. In many cases it is impossible to comply with this condition, except where the coal is being loaded, discharged, or rehandled. Coal becomes better mixed each time it is rehandled, thus reducing errors arising from failure to fully comply with this stipulation.

A small shovel or scoop of 2 or 3 pounds capacity is well enough for accumulating samples of slack or small sizes of anthracite, but it would evidently be wrong to take such small increments from run-of-mine or prepared sizes where large lumps are present. When coal is extremely lumpy it is best to break a proportional amount of the lumps before taking the various portions of the sample. As a rule, 10- to 12-pound increments are satisfactory for the ordinary sizes of coal.

There is a natural tendency for a person to take a larger amount of fine coal than is proportional to the total, when taking a sample by shovel. In other cases the coarser coal may be more available, and with a mechanical sampler it is possible for a constant error to exist in obtaining an undue proportion of either the fine or coarse coal. Such conditions should be carefully avoided to prevent serious errors. In some coal the fine material is much higher in ash and sulphur, while in others the reverse is true.

A mechanical sampler cannot discriminate for or against slate and other impurities, and if the sample is taken by shovel, the person doing so should be as blind to such impurities as is the machine. There is no one who can possibly be capable of judging whether any certain impurities should be included or rejected, as he cannot view the entire quantity of coal as carefully as he can the sample; but even if he could, it would be useless for him to attempt to accurately estimate the proportional amount of impurities. The only safe way is to accumulate portions of the sample from equal parts of the whole, and leave the amount of impurities entirely to the law of chance.

Quantity of Original Sample.—The total weight of the original sample accumulated is of the greatest importance. For illustration, take the problem of sampling 100 tons of coal containing 200 pieces of slate which weigh 1 pound each. This corresponds to a 1-pound piece of slate to each 1,000 pounds of coal, or only .1 per cent. ash over and above the other ash contained in the coal, which, we will say, is 6 per cent., making the total correct ash 6.1 per cent. If a person took a 10-pound sample, which is as large as many people do take, it is possible for one of these 200 pieces of slate to be contained in it, which would amount to 10 per cent. in addition to the 6 per cent. intrinsic ash, making 16 per cent. total ash in the sample. Even a 100-pound sample containing one piece of this slate would show 7 per cent. ash, instead of the true 6.1 per cent., and unless a person has actually counted the pieces of slate in the entire 100 tons he does not know whether or not he should have one or more such pieces of slate to every 100 pounds he might happen to take as a sample; but if a 1,000-pound original sample were taken, and it had even two or three pieces of slate more or less

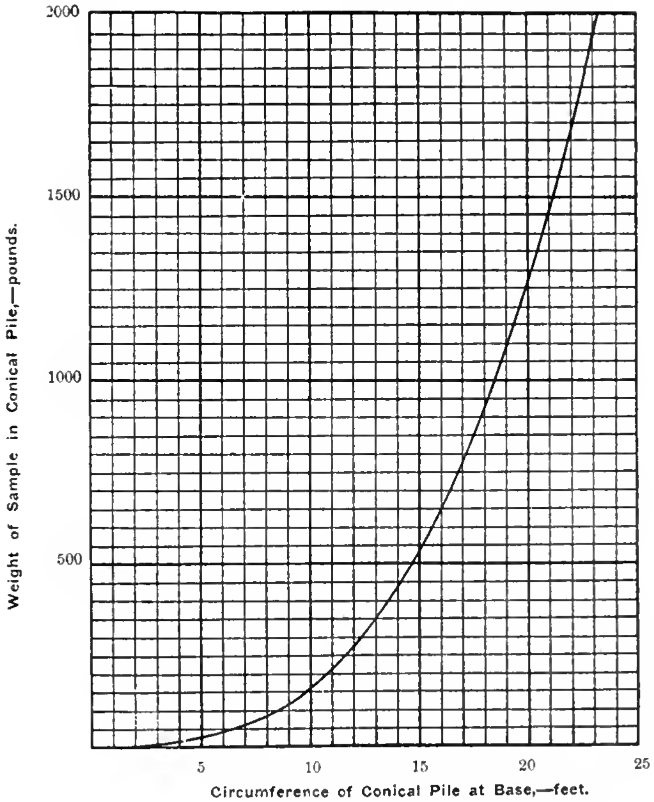


FIG. 1. CURVE SHOWING APPROXIMATE WEIGHT OF SAMPLE WHEN FORMED INTO A CONICAL PILE

*Mechanical Engineer, 220 Devonshire Street, Boston, Mass.

than the true proportion, it would cause an error of only a few tenths of a per cent. ash, which is allowable.

If this 100 tons of coal contained 2,000 pieces of slate, weighing 1 pound each, the true ash would be 7 per cent., and while there is one piece for each 100 pounds of coal, a sample of this weight might happen to contain several, and each one would cause an error of 1 per cent. Even a 1,000-pound sample, which should contain 10 such pieces of slate, might happen to contain only five, or as many as 15, which would correspond to a possible error of .5 per cent. below or above the average, as compared with less than half this possible error with the smaller amount of free slate.

In order to secure an accurate sample of coal it is absolutely necessary to take such a large original sample that several of the largest pieces of slate or impurities more or less will cause no material error. The larger the size of the pieces of impurities, the larger should be the original sample to prevent excessive error, and the same is true with respect to the quantity of impurities. In a similar manner the quantity of impurities and the size of sample to be divided should govern the fineness of crushing the sample throughout the various stages of dividing and quartering, from the original sample to that which is pulverized in the laboratory for analysis. The total percentage of ash or impurities does not indicate the size of sample which should be taken, but it is the impurities existing as slate, bone, and extraneous matter which should be considered, regardless of the amount of impurities distributed uniformly through the coal itself

On account of the variation in the size and amount of the impurities in different coals, it is impossible to state how large a sample should be taken in each case. Many coals contain between 1 and 2 per cent. of ash in the form of slate and free impurities, and when the larger pieces of slate do not weigh more than 1 pound, a 2,000-pound original sample would be sufficient for reasonable accuracy. If there is any doubt about the quantity of sample that should be taken, it is best to pick out all slate and bone larger than ½ inch from the sample as it is being taken and weigh it, and from the weight of sample from which it was taken then determine approximate percentage of ash existing in this form. With these data and the size of the largest piece of slate, one can determine the quantity of sample which should be taken by referring to Table 1. All of this slate and bone should then be broken and thoroughly mixed with the coal before quartering it.

Table 1. Weight of Original Sample to Be Taken

Largest Size of Slate Existing in Coal	½ Per Cent. Ash in Free Impurities	¾ Per Cent. Ash in Free Impurities
2-inch screen	2,000 pounds	4,000 pounds
1½-inch screen	1,000 pounds	2,000 pounds
1¼-inch screen	600 pounds	1,200 pounds
1-inch screen	300 pounds	600 pounds
¾-inch screen	120 pounds	250 pounds
½-inch screen	50 pounds	100 pounds

Samples of the above size will also insure sufficient accuracy in the sulphur determination unless there is a large percentage of sulphur balls present, in which case the sulphur would be much higher than exists in coal used in iron, steel, and gas industries, and extreme accuracy would not then be required.

The most satisfactory method of determining the approximate weight of samples accumulated is to measure the conical pile. The circumference at the base and the slant height are the dimensions most readily determined. The slant height is very nearly one-fifth of the circumference, and whatever variation there may be from this figure causes only a slight error, so the weight of coal in a sample can be expressed in terms of the circumference of the base of the conical pile. This can easily be measured by means of a 25-foot tape measure, and from the curve in Fig. 1 the weight of coal can be determined accurately enough for all practical purposes.

The size and percentage of the impurities have more influence on the amount of original sample to be taken than has the quantity of coal to be sampled. It has often been stated that a certain percentage, say .1 per cent., of the original quantity of coal should be taken as a sample, which would mean about 15,000 pounds from a 7,000-ton cargo, and 50 to 100 pounds from a railroad car. As a matter of fact, the same size sample should be taken from each, if they contain the same kind of coal and the same accuracy is desired in each case.

Reduction of Sample.—The original sample is larger than can be conveniently sent to the laboratory, and the manner by which it is reduced to a smaller quantity is equal in importance to the method of accumulating the original sample. It is usually broken, mixed, and divided repeatedly until any desired quantity remains. In doing this it is essential that the part discarded be exactly like that retained, and so long as this is true the final sample represents the original. In order to meet these conditions it is necessary to break all lumps of coal, slate, and impurities very fine, as compared with the quantity of sample before every division, in order that the slate and impurities may be equally distributed throughout the whole by thorough mixing. Table 2 will be found of assistance in determining how fine the sample should be crushed before mixing and dividing.

Table 2

Weight of Sample to Be Divided	Size to Which Slate and Impurities Should be Broken
4,000 pounds	1½ inch
1,200 pounds	1 inch
500 pounds	¾ inch
200 pounds	½ inch
50 pounds	2 mesh
5 pounds	4 mesh

It is not necessary for the lumps of coal to be broken as fine as should be the slate and impurities.

When taking large samples of coal a person should have a smooth solid floor upon which the sample can be broken and mixed, also an easy rapid method of crushing the coal. Where a great deal of sampling is to be done at one place the necessary arrangements can be made and a hand or power crusher installed, but when samples are to be taken from miscellaneous places a light portable outfit is necessary. In the absence of any crusher, an iron tamping bar is the easiest and most rapid method of crushing the sample, but for portable work a flat-sided hammer is about the best tool that can be used. Meat choppers work fairly well for the finer crushing, but are rather slow and difficult to feed.

Whenever the different shovelfuls or increments of samples are taken throughout some considerable period of time, each shovelful should be crushed as soon as taken, and the impurities can often be broken sufficiently small to permit the total accumulated sample to be quartered three or four times before further crushing is necessary.

In the absence of a smooth tight floor it is necessary to use a heavy canvas sampling cloth. The coal can be broken on a solid piece of iron or block by means of a hammer, and after the sample has been accumulated and broken it can be mixed by raising first one end of the canvas, then the other, rolling the sample back and forth. A sample of 300 or 400 pounds on a canvas 6 ft. × 8 ft. is about all one man can handle, and in cases where a larger sample is necessary it can be taken by instalments, each 300 or 400 pounds being broken and quartered three or four times and then reserved till the entire sample required has been so reduced, then they can be mixed and worked down as one sample.

After a sample has been broken to the size corresponding to its quantity, as given in Table 2, it should be thoroughly mixed by shoveling it over and over several times, or by rolling back and forth on a canvas, before it is divided.

Original samples larger than 300 pounds can best be divided

by discarding alternate shovelfuls, while samples of about this size or smaller can be formed into a conical pile and divided through the center into four equal parts and the two opposite quarters discarded. The part of the sample remaining should be crushed smaller, again mixed and divided, or quartered, until it is finally reduced to not less than 25 pounds, all of which will pass through a 2-mesh sieve; or 3 pounds, finer than 4 mesh; which may then be sent to the laboratory.

A man can take and properly work down a sample of 1,500 pounds of soft semibituminous coal in a day, doing all crushing by hammer. A correspondingly less amount of the harder gas and anthracite coal can be taken and reduced, much depending upon the average size of the original coal.

Special Moisture Sample.—The percentage of moisture contained in the coal sampled should be determined as accurately as is the ash or any other constituent, but this is impossible if it is taken in the manner just described, even though it is sent to the laboratory in hermetically sealed jars or cans. In other words, it is impossible to obtain a sample which will correctly represent the ash and sulphur and still retain the original moisture, if it is worked up in the open air. The necessary crushing, mixing, and dividing cause a loss of 2 to 3 per cent. of moisture under average weather conditions. The only method by which the true moisture can be determined under normal conditions is by taking a separate special moisture sample. This is accumulated by placing small quantities from the freshly taken portions of the original sample into a hermetically sealed receptacle, which is not again opened until it reaches the laboratory. This special moisture sample is used for no other purpose than to obtain the total moisture in the coal sampled. The analysis of the other sample, which is practically air dried by the time it reaches the laboratory, should be calculated to the total moisture, as determined from the special moisture sample, in order to obtain the true analysis of the coal sampled.

Loss of Sample.—It might seem unnecessary to caution against losing any dust by the wind, or fine coal sifting through cracks in the floor, but errors due to failure to realize the importance of such precautions are often encountered. Another error is sometimes caused by hard pieces of bone or slate flying out of the sample when struck a glancing blow with a hammer.

SAMPLING AT COAL MINES

Samples From Seam of Coal.—Samples of coal cut from the working face in a mine are of little value for determining the quality of coal shipped to market, due to failure on the part of the miner to eliminate all roof, floor, and parting slate, and impurities in the coal loaded, as is usually done in taking section samples. However, mine samples are of value in determining the character and approximate quality of coal from a new or idle property. They are also of value when taken from strata of the seam separately, in locating the part of the seam which contains the higher or lower percentages of ash, sulphur, phosphorus, or clinker-forming impurities. If consistent with economical mining, parts of the seam may be left in, or gobbed, in order to produce a coal of desired purity to meet certain market demands.

Mine samples should be taken as a standard in representing the best coal that could possibly be produced with perfect preparation, and the quality of coal being shipped should be made to approach this standard as closely as is practicable.

In some mining districts the sulphur varies widely in different parts of a mine, and often certain headings will be found to produce coal low enough in sulphur to meet certain specifications while the average of the mine would be prohibited.

Very few coal operators open up a new property without first diamond drilling it to learn the dip and uniformity of the seam. After expending so much money to obtain these cores, they should be analyzed and the quality of coal from different parts of the field determined. It even pays to make a separate sample from each 6 to 12 inches of the core, whether divided according to natural partings or not.

The method of taking mine samples is to select or make a freshly exposed full section of the seam and spread a canvas (4 ft. × 6 ft.) on the floor beneath it, then cut a channel about 4 inches wide and 4 inches deep from the bottom to the top of the seam, leaving any partings or streak of impurities which are to be discarded in mining. Care should be taken to see that all coal cut from the upper part of the seam falls on to the canvas, as it has a tendency to spread when it falls. It is also essential that the groove be uniform in size from top to bottom and not smaller where the coal is hardest and larger where it is softest. When the coal has well-defined butts and faces it is easier to cut a uniform groove from the butts. After the sample has been dug from the seam it should be broken and reduced in accordance with the method already described and placed in a hermetically sealed jar or can, if the actual moisture is desired, otherwise a coin sack will be more convenient. It is unnecessary to take a special moisture sample before crushing and mixing mine samples, provided the work is done immediately and before the sample is taken out of the damp atmosphere of the mine.

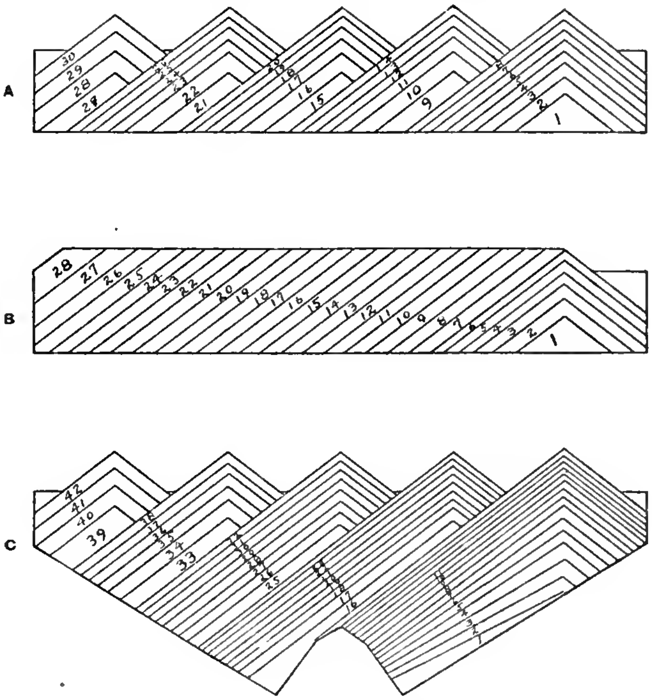


FIG. 2. DIAGRAMS REPRESENTING THE DISTRIBUTION OF COAL AS DUMPED FROM MINE CARS INTO RAILROAD CARS
A—Section representing car moved to five positions as loaded at the mines
B—Section representing car moved a short distance as each mine car of coal was dumped. C—Section representing steel-hopper car moved to five positions as loaded at the mine

If detailed information of the quality of coal in different parts of a mine is desired, and the cost of having a large number of samples analyzed is not too great, such samples should be taken from every three or four working places. Where only a few samples can be analyzed it is not best to depend upon single sections for each sample, but three or more such grooves should be cut and the coal from them mixed as one sample to represent the heading or part of the mine from which they were taken.

It is frequently desired to know the quality of coal coming from different benches or strata of a seam, in which case the groove should be cut across such parts of the seam separately.

Samples From Mine Cars, Tipples, and Loading Railroad Cars.—The comparison of the analyses of samples taken from mine cars with those taken from sections in the working places indicates the care used by the miners in cleaning the coal. A

similar comparison between the results of mine-car and railroad-car samples shows the efficiency of the tipplemen and car trimmers with respect to preparation. Railroad car samples represent the quality of coal shipped to market, as well as a comparison of the different grades, where the coal is screened or washed.

Mine-car samples should not be taken from the tops of loaded cars, as they are likely to be trimmed with lumps, thus preventing the sampler from securing a true proportion of the lumps and fine. There is also a chance of pieces of roof or draw slate having fallen on to the car, which might cause considerable error in a sample taken from the tops of the cars. It is better to take mine-car samples from the open end of the car as it is being dumped, or from a chute as the coal passes. A large shovelful (15 or 20 pounds) should be so taken from each mine car at the smaller mines up to every tenth or more cars at the larger producers, and the sample accumulated throughout at least one entire day's loading. It is often desired to know the quality of coal being loaded from different headings separately, which can readily be done by throwing the shovelfuls taken from cars coming from different parts of the mine into different piles. Each individual sample should be about as large as if

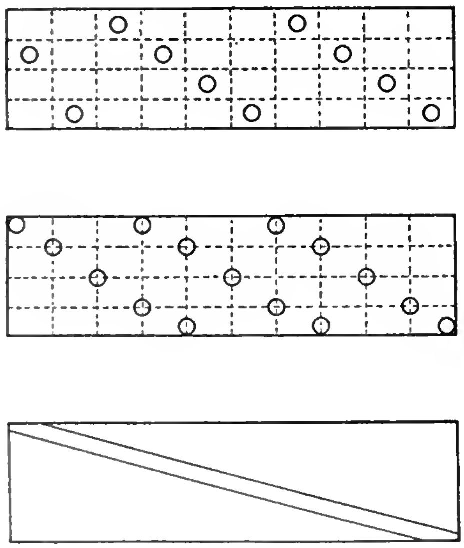


Fig. 3. Diagrams Showing Different Methods of Locating Holes and Trench on the Surface of Railroad Cars from Which Samples are to be Taken

one large sample were taken from all mine cars as a whole. The total weight of sample taken in either case should be governed by the size and amount of impurities contained in the coal. Size of sample and method of reducing it should be as previously described.

If there is any attempt made to throw out impurities as the car is being loaded, the samples representing the coal shipped should be taken from the railroad cars throughout their loading. Shovelfuls should be taken from the slanting surface just before a new mine car or basketful is dumped, and after the slate pickers have finished cleaning the one previously dumped. There is a tendency for the lumps and heavier impurities to go to the bottom of the slanting surface, hence shovelfuls should be taken systematically from various parts of this surface. The shovel should be pushed well into the coal in order to avoid getting only that on the surface from which the impurities have been picked. Occasionally it is desired to take such a sample from a single car, in which case it would be necessary to take several shovelfuls from each mine car dumped, but if a whole day's loading is to be represented, a shovelful need be taken from only every fifth, tenth, or whatever number of mine cars is necessary to secure the required total sample.

An operator often investigates the quality of coal actually being shipped to the market for the purpose of seeing whether or not the coal can meet certain specifications for ash, sulphur, or British thermal units, and if not, how much better preparation would be required to meet such specifications. Very valuable information is gained by taking duplicate samples from the cars and picking out all slate and sulphur from one and comparing the result with the regular representative sample. This will indicate what reduction can be made in the different impurities, and by weighing or measuring the sample, and weighing and noting the size of impurities picked out, one can determine if this better preparation could best be accomplished through inside or tipple inspection, or whether a picker belt would be required. If a washery is being considered, laboratory washing tests can be made to determine the quality of washed coal and the loss resulting from washing to any desired quality.

Sampling Railroad Cars—Loaded or Unloading.—There is great difficulty in obtaining a representative sample of coal from a railroad car that was loaded from individual mine cars, unless the coal can be sampled while it is being unloaded. The usual method of loading cars at the mines makes it almost impossible to obtain a sample from a loaded car which will contain a true proportion of whatever slate and impurities remained in the car. The slate and impurities are usually picked from the surface of a car as it is loading. Fig. 2 represents some of the conditions frequently encountered; the areas containing numbers representing the quantity of coal from each mine car and the numbers give the order in which quantities were dumped. "A" represents a car that was moved to five different positions while it was being loaded. It is evident that any scheme of laying out points from which to take shovelfuls for a sample, or digging trenches, will not include any coal from several mine cars which lie near the bottom and will include an undue proportion of coal from the mine cars emptied on top. "B" illustrates a flat-bottom car which was moved a short distance each time a mine car was dumped. When coal is loaded in this manner, the digging of a trench or holes should secure almost an equal proportion of sample from each mine car, but there is another error of even more importance that is likely to be greater from cars loaded in this manner than those like "A." This error is due to the tendency of the bone, slate, and heavier impurities, as well as the lumps of coal, to roll toward the bottom of the car while it is being loaded. Some authorities who have taken separate samples from the top and bottom of a car upon its arrival at destination, have claimed that the higher percentages of ash and sulphur found in the samples taken from the bottom of the car are due to the heavier impurities settling to the bottom during transit. While these impurities may tend to settle some in transit, the most plausible explanation of their being at the bottom of a car is found in the manner of loading the cars at the mines. Car "C," representing a steel hopper, illustrates the tendency for slate to slide to the bottom while being loaded, and the increased difficulty of obtaining a representative sample at destination without unloading the car.

If there is no alternative but to sample a loaded railroad car, points should be located systematically on the surface and 10 to 15 holes dug at least 2 or 3 feet deep. The upright braces on the side of the car can conveniently be used as guides in locating the position of the holes according to some fixed system, similar to those illustrated in Fig. 3. Directions regarding the amount of sample to be taken and the method of its reduction, as already given, should be followed.

If a sample is to be taken from a car being unloaded by hand, it is best for the sampler to have one shovelful out of every so many placed into the sample. As usually shoveled, there are about 125 to 150 shovelfuls per ton, and from the amount of coal in the car and the size of sample required, one can quickly estimate about how frequently a shovelful should be thrown into the sample.

If the car is unloaded through hopper bottoms to a pile,

the sample should be taken as the coal is dropping out of the car, and if one car is to be sampled as an individual it may be necessary to retard the dumping in order to have enough time to accumulate a sample of sufficient size from all parts of the car. Coal should never be sampled from the pile if it is possible to do it otherwise.

If a car of coal is unloaded into a mechanical conveyer, either by hand or drop bottom, the sample should be taken from the conveyer. If a clamshell or grab bucket is used to take the coal from a car, the sample may be taken from the bucket itself or from the newly exposed surface in the car throughout the unloading, unless it is discharged into a mechanical conveyer or industrial railway from which the sample could be more easily taken.

A composite sample is often desired from several cars of coal loaded into or from a single vessel or else where shipped as a special consignment, in which case a proportional amount of the required sample should be taken from each car. In the case of coal shipped all rail to the consumer it is best to sample a car individually, even if only a part of the total number of cars received can be tested. It is then possible to supply definite information regarding that particular shipment, while if a composite sample were made from two or more cars there could be no definite knowledge gained in case they were loaded at different mines.

COAL MINING NEWS

A German mining engineer and mine director advises an American consular officer that he has made a special study of the possibilities for the profitable importation of American coal into Germany and various parts of Europe, and that he is convinced a successful business could be developed providing the coal could be sent over in shipments of 8,000 to 10,000 tons. He therefore desires to enter into correspondence with parties in the United States who might be interested in exporting to Europe coal, anthracite and steam coal of all kinds and sizes, as well as coke, with Hamburg, Amsterdam, and Genoa as European ports of arrival.

One of the largest single sales of transformers made by Allis-Chalmers Co. during the month was to the Delaware, Lackawanna & Western Railroad Co. The order includes six 200-kilowatt three-phase; three 200-kilowatt single-phase; two 150-kilowatt three-phase, and one 100-kilowatt single-phase, oil-filled, self-cooled transformers. These will be installed in collieries and washeries of the company at Scranton, Nanticoke, and Taylor, Pa.

A good gas coal will furnish 5 cubic feet of gas per pound of coal, or about 11,200 cubic feet of gas per ton of 2,240 pounds.

The imports of American coal into Brazil in 1909 were larger than ever before, owing to a threatened strike among the coal miners of Wales. Consul-General Anderson says: The imports of coal into Brazil were valued at $8,689,482, of which the United States furnished $238,645. In the previous year the value of the coal imported was $9,719,334, and the share of the United States was $19,015. The registration and port dues taxes amount to 4 per cent. of the gold value of coal, or about 95 cents United States money. Coal imported by steamship companies for their own use pays a tax of 40 cents per ton. The contract price for Welsh fuel is at Rio de Janerio 32 shillings, or $8 per ton. The coal trade of Brazil has been and is practically controlled by British interests, which first developed it. Railways first constructed in Brazil were generally built by British capital and have preferred British coal; steamships touching at Brazilian ports have been preponderatingly British, with British engineers who prefer British coal, and the firms handling British coal have been British themselves. The concern now handling most of the coal sold along the east coast of South America is not only British, but is directly interested in the mines in Wales from which most of the coal comes.

The value of a coal mine decreases with every ton of coal removed; the cost of mining increases as the excavation expands; the machinery deteriorates at coal mines so that if 10 per cent. is knocked off yearly for depreciation it will not always cover the loss; the equipment deteriorates at a much greater rate, in fact timbers, ties, cars, tools, etc., sometimes have to be replaced in a year's time. The money for the renewal of equipment and machinery must come from the mine which is gradually becoming bankrupt.

To overcome the fixed charges due to the expansion of a mine, it is necessary to mine a certain number of tons of coal on which there is no profit. This number will depend on the size of the operation and the selling price of the product. There are instances where it is necessary to drive 1 mile of entry in order to obtain sufficient coal from rooms to pay fixed charges. Bearing in mind that the cost of mining increases with development, and that machinery and tools deteriorate and must be paid for in coal, it becomes evident that in due time the mine will become bankrupt unless the selling price of coal is increased to insure a profit.

It is one of the laws of trade that if one man has a thing to sell which another man wants to buy, a sale can be brought about; nevertheless there have been rumors that competition has been so severe in bituminous coal that operators have not made money, and have actually sold coal to railroads below the cost of production, with the hope of obtaining a price from legitimate customers that would make up the deficit.

In 1905 the Chesapeake & Ohio Railway officials informed Abram S. Hewitt that he would have to give them his product from the Gauley mines for considerably less than formerly. Mr. Hewitt said he would not and would hold them to the former price. The officials stated they would not take the coal, and wanted to know what he was going to do about it. Mr. Hewitt replied, "Shut down the mines." "What? Shut down your mines and force us to lose all that coal freight and involve us in contracts we have made?" "Yes, sir; we are not running our coal mines to lose money." Mr. Hewitt was not cut in prices, although other Chesapeake & Ohio operators were.

Recently the coal operators in the Eighth Ohio District determined to stand together for a price that would yield them a profit. It appears that they had been maintaining coaling stations to supply railroad locomotives and selling the coal below the actual cost of production, and had not been making any money. They boosted the price to the railroads 10 cents, whereupon the railway purchasing agents declared they never would pay it, as they could buy coal cheaper elsewhere. To use a Congressional expression the operators were "stand patters" and said, do so. The railroad officials interviewed the operators, and were told pertinent facts concerning the coal business, also that the railroads would have to haul coal to their coaling stations, also store it, employ men to handle and watch it day and night, and dump it into the tenders. The railroads opened negotiations for a compromise, but the "stand patters" would have none of it, and won.

Satisfactory progress is being made at the new operations of the Four States Coal and Coke Co. (Jones interests) in northern West Virginia. The town of Annabelle is being built in connection with the development work, and work upon the openings is also being pushed. When the plan is fully developed it will form one of the largest mining operations in the country, rivaling in size the new operations of the same interests in Southwestern Pennsylvania.

Some people are getting apprehensive in regard to the future value of Rhode Island coal stock, and a Boston publication comments upon a drop to $6 per share for the stock in the curb market. President Whitney is still optimistic and states that developments have been very favorable and all experiments have proven very satisfactory. Between 35 and 40 tons a day are now being produced, and a large increase is expected in August, when the permanent breaker will be in operation.

It is maintained that the coal can be sold for $6.50 a ton, which, with $1.50 for delivery expenses and $2 for mining, assures a profit, according to the calculations of the head of the enterprise, amounting to $3 per ton.

The proved coal fields in Mexico are all in the Sabinas coal field, state of Coahuila, although coal should be found to the west as well as oil. The known coal covers 300,000 acres, from which the minimum average production is fixed at 3,000 tons per acre. The operators in Mexico have invested $20,000,000, and believe they are entitled to some protection, as they are unable to ship into the United States owing to the tariff wall, while on the other hand they are obliged to compete with coal and coke that is sold as a surplus from the mines of the United States and shipped as ballast to Mexican seaports. It would not be advantageous for Mexico to impose any duty on coal or coke, as it would seriously cripple consumers who operate plants on the west coast as well as other points in Mexico impossible to be reached by the native products, but the railroads should protect the territory in the immediate vicinity of Mexican mines from foreign coal and coke by making the rates at least as high per kilometer as is charged from native mines to such points. The railroads claim that it is impossible to make any reduction on the rates on coal and coke from the mines to the different points of consumption in Mexico, and that the rates on foreign coal and coke from Vera Cruz and Tampico to interior points were originally too low. It is therefore asked by the Mexican mine owners that the rates on the foreign products be increased.

The *Black Diamond* has this to say about convict labor in mines: Should the measure pending in Congress ever become a law and operative, whereby convict-mined coal could not be shipped outside of a state, the state coal mining business of Tennessee would be doomed. The coal operators of Tennessee and Kentucky are laboring on their part to obtain legislation whereby the state will cease coal mining altogether.

Appropos, Alabama is a peach. The law in Alabama says the miner shall be paid $1 per ton for coal. The convicts are let out to contractors who feed and work them in the mines of the large operations. Greenhorns from the farms are put in gassy mines to work, and if they do not produce the required amount of coal are flogged. No person talked to in Alabama likes the system, which is graft from top to bottom, and it is not believed the Steel Corporation will perpetuate the present disgrace. It would be a very easy matter to trump up a charge to keep men indefinitely or send new men into the mines of Alabama.

A widespread interest is being taken in opening the Mullaghmore colliery, in Queens County, Ireland. It is thought that if the operation of this mine is successful, others will again be put in operation in different parts of Ireland. Coal it is said was extensively mined in this county some 30 years ago, but the product is not suitable for all purposes, and the mines were abandoned. For certain purposes it can be used to good advantage and will prove vastly more economical than imported coal.—*United States Consular Report.*

According to the *Iron and Coal Trade Review*, of London, England, Welsh coal operators are having a serious time through the loss of foreign contracts. The most important cause leading up to this crisis are the numerous isolated strikes which close the mines for a few days. This sort of thing has created abroad an uncertainty and fear among large consumers in regard to contracts being carried out. After the great strike of 1898 the miners worked with their utmost energy, and the supply of coal could be relied on; now, however, they throw down their tools and leave the mine in consequence of some trivial dispute. The falling off in the demand for coal is making itself felt among the miners, as work is not continuous, and distress is being felt, more especially in Monmouthshire, where children are being fed by the public. Another cause for the lessened demand for Welsh coal is the difference in price between it and the North of England coal. The superiority of the South Wales coal naturally gives it a higher market price, but there is a limit to this difference and the best North of England coal is selling from 75 cents to $1 below the Welsh coal. Coal operators in Wales realize the effect this difference in price has upon their trade, but they declare that since the passing of the Eight-Hours Act the cost of working the mines has so increased they cannot decrease the price of coal except by a reduction of miners' wages and the introduction of such measures as will increase the output. All the great foot-ball matches are played on Saturdays and the younger miners desire to leave work early to attend them or other amusements. They have no objection to commencing work at an earlier hour, but there is an Act which determines the interval that must elapse between one shift and the next at coal mines. The miners thought that their Federation would support them in regard to this matter of overlapping shift, but as this was not done, the men are not supporting the labor union. It is said that many miners have never paid one penny to the Federation since the agreement was signed, and this, with the secession of special mine laborers, is a serious blow to the association.

In 1909 the number of men employed in the Yorkshire and Lincolnshire districts, in 399 mines, was 141,102. Of this number 111,404 worked below ground, and 29,698 above ground. There were 35,894,776 tons of coal mined, and the output of mineral per person below ground was 327 tons, and for those above and below ground 258 tons. In this district there were 155 fatal accidents inside and 27 outside the mines. The death rate in 1909 was 1.29 per thousand persons employed; 63.2 per cent. of the fatalities came from roof falls; 6.4 per cent. from accidents in shafts; 1.9 per cent. from explosions of gas; and 28.5 per cent. from miscellaneous causes. There were 22,002 non-fatal accidents which disabled the injured more than 7 days. Legal action was taken against one coal operator, and 249 workmen. Coal operators show a disposition to abandon the system of fining in favor of prosecutions. In the 249 charges preferred against workmen, 244 convictions were obtained, and fines and costs inflicted; in three cases costs only were imposed; and two cases were dismissed.

During the fiscal year that ended June 30, 1910, the value of the bituminous coal exported from the United States amounted to $26,000,000. Both anthracite and bituminous coal exports had greater value than in any earlier year.

The Pocahontas semibituminous coal is the best all-around coal in the country. The mines for the most part are worked by lessees who pay royalty to the land owners. For years the operators have been working on a very narrow margin of profit and selling their product in competition with poorer coal. Apparently they have awakened to the fact that their coal is in a class by itself and as such can command a price which will return a reasonable profit. The operators last year received $1 per ton for run-of-mine coal on board cars at the tipple, while in Alabama the miners are paid $1 per ton for mining. This year the operators split up into several selling agencies, although previously, with the few exceptions of independent operators, they had been selling through one agency. In April of this year the price of Pocahontas coal was advanced to $1.10 at the tipple, and consumers who had been paying $1 refused without good reason to pay the 10 cents increase. In taking this stand the consumers are the losers, for the intrinsic value of Pocahontas smokeless coal is more than most other coals. While perhaps it was policy to at first sell smokeless coal cheap in order to educate consumers to its value, that time passed long ago, and had selling conditions been different the price of this coal would have stiffened and operators would not have been working on a hand-to-mouth basis.

In the past large corporations told the Pocahontas operators at what price they would allow them to mine and ship coal. Because the operators refuse to be dominated, the large corporations have gone on a strike.

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

(Continued from August)

Mine Inspectors' Examination Held at Scranton, Pa., May 2-5, 1910

QUES. 16.—What determines the size of a barrier pillar in the anthracite field?

ANS.—The rule adopted by the leading coal companies, and the mine inspectors of the anthracite district for determining the necessary width of barrier to be left between adjoining workings is as follows: *Add 5 feet to 1 per cent. of the depth of the seam below the surface, and multiply the sum by the thickness of the seam in feet. The product obtained will be the required width of barrier pillar, in feet.* Thus, for a 6-foot seam lying 800 feet below the surface the required width of barrier pillar would be $6(.01 \times 800 + 5) = 78$ feet.

QUES. 17.—When extracting a range of pillars of an average thickness of 10 feet, how would you proceed with the work to insure the safety of the workmen and the mine? What should be carefully considered before the work of drawing pillars is begun?

ANS.—The details of this work will depend much on the nature of the roof, hardness of the coal, thickness of the seam, and depth below the surface; but, in general, the work on the several pillars should be kept in line and proceed regularly without cessation. Experienced miners should be employed in pillar drawing; no timber should be left standing in the waste; a close watch should be kept for slips and roof cracks. Only safety lamps should be used in pillar work where gas may be expected in the roof. Before beginning to draw back pillars, the probable effect on adjoining workings and the surface or overlying seams should be carefully studied; also, whether their removal will cause the possible loss of certain coal.

QUES. 18.—How should you determine when a hoisting rope is unsafe, and what portion of the rope would you consider the weakest part, or the most liable to give out first?

ANS.—Hoisting ropes should be tested at regular short periods, say once in from 3 to 6 months, by lifting a load off the shaft bottom at least double the weight that may be hoisted at one time, in the regular operation of the mine. This load should be raised a few feet above the landing and held there a few minutes and then lowered again to the landing; this should be repeated two or three times, while the effect on the rope is closely observed. Again, with the load ordinarily hoisted, the engine should be started suddenly, with a few inches of slack rope to test the resistance of the rope and end fastenings to such starting strains. All hoisting ropes used for hoisting men should be examined closely each day by a competent man allowing the rope to pass through a bunch of waste held in his hand, as the cage is hoisted and lowered slowly, the purpose being to detect by this means any loose or protruding wires. The rope for a few feet above the cage fastening is subject to more severe usage than any other portion of its length, owing to sharp bends when rope is slack, severe shock when engine is started suddenly, exposure to injury from immersion in mine water, abrasion, and accidental blows incident to handling of timbers and other material in the shaft, and last but not least, the effect of the swaying back and forth of the rope hanging in the shaft, which effect is most felt at the cage. For these reasons, this portion of a hoisting rope is liable to show the first signs of weakening.

QUES. 19.—(*a*) What are the distinguishing features of, first, a mine post properly stood; and second, a set of mine timbers, or a timber set? (*b*) What is your opinion of the timbering done in the anthracite mines, based upon your own observation? (*c*) What would you suggest to obtain the best results in timbering, from a standpoint of safety and economy?

ANS.—(*a*) When properly stood a mine post is vertical, in a flat seam, or *underset* (inclined a few inches up the pitch from a normal to the floor), in an inclined seam. The amount a post should be underset depends on the inclination of the seam. All mine posts should be squared at each end and have a firm bearing at the roof and the floor. To distribute the thrust of the post over a greater area of roof, give a more uniform bearing on the timber, and protect the head of the post, a cap piece of soft pine, 18 inches long and 1½ or 2 inches thick, should be used above the post. The chief features of a good timber set are the correct notching and even bearing of the two upright legs and the cross-bar; the slight inward inclination of the legs; the solid foot-hold cut in the floor or sides of the entry; and the proper wedging of the frame against the roof so that the roof pressure will be transmitted to the legs, and not be thrown on the center of the cross-bar. (*b*) This calls for the opinion of candidate based on his personal observation. (*c*) To obtain the best results with respect to safety and economy, timbering should be done promptly, as required. There should always be kept on hand a good supply of props, caps, and other timbers of the sizes demanded in the mine. Entry timbering should be done by experienced timber men, and all post timbering done by the miners at the working face should be inspected by the mine foreman or his assistant to see that the work has been properly done.

QUES. 20.—(*a*) To what principal causes can the loss of life occurring in the anthracite mines be attributed, and what is the percentage for each cause? (*b*) Give briefly that portion of the Anthracite Mine Law designed to prevent fatalities from each of the above causes. (*c*) What are your ideas as a candidate for mine inspector, for the prevention of mine accidents?

ANS.—(*a*) The chief causes of mine accidents below ground and the average percentage for each cause are as follows: Falls of coal and slate, 50 per cent.; blasting and powder, 15 per cent.; gas, 10 per cent.; mine cars, 15 per cent.; falling down shafts or slopes, 5 per cent.; miscellaneous, 5 per cent. (*b*) Art. 11, requires the mine operator to furnish the miner with all necessary props and timbers as required and ordered by him. Art. 12, rule 14, requires each miner to keep the roof and sides of his place securely timbered and to do no work under loose top; and rule 55 prohibits the cutting of props or timbers in position to support roof or sides. Rules 11, 26–36 inclusive, and 38, relate to the blasting of coal and rock in mines, the storage and handling of powder, the extinction of gas (feeders) ignited by a blast, and the examination of a working place before and after firing. Art. 10 specifies the methods and means to be used in the ventilation of a mine, of which work (Art. 12, rule 3) the mine foreman is to have the full charge. Rules 4–10, inclusive, specify the manner and time of examining the mine for gas, and provide for the withdrawal of the men in case of danger, and the use and custody of safety lamps. Rule 37 forbids the removal of gas by brushing. Rules 16, 21, and 40–52, inclusive, relate to the safeguarding of the lives of workmen in mines in connection with the movement and handling of mine cars. Art. 4 relates to shaft and slope openings and provides for various appliances designed to minimize the danger to men working in and around shafts and slopes. (*c*) This question calls for the personal ideas of each candidate, in respect to the prevention of mine accidents. A good answer can be expressed briefly as follows: Simplify and enforce mine regulations and laws adapted to the safe operation of each mine, with suitable penalties for infraction thereof. Restrict mine labor, inside and out, to those of lawful age who can speak, read, and understand English, and hold the certificate of a properly authorized examining board clearly stating their qualifications. Require the suitable certification of all classes of mine workers and mine officials charged with the direct operation of the mine and allow no interference with their authority. Educate all miners and mine officials to a knowledge of the principles of mining and the dangers common thereto. Watch closely the operation of each mine.

Ques. 21.—Name the safety appliances required by law to be used in and about the anthracite mines and state what special care should be taken of them.

Ans.—The escape shaft (Art. 4, Sec. 3) must be fitted with safe and available appliances for escape from the mine. The tops of all shafts and slopes, if dangerous (Sec. 6) must be fenced or provided with vertical or flat gates; likewise (Secs. 7, 8) every abandoned opening must be securely fenced. All working shafts and slopes (Sec. 9) must be provided with suitable speaking tubes and signal apparatus. Cages for hoisting or lowering men (Secs. 10, 11) must be provided with hand rails, efficient safety catches, and overhead covers; and (Sec. 12) bridle chains. Winding drums of engines for hoisting men (Secs. 14, 15) must be provided with an efficient brake and side flanges to prevent the rope slipping off the drum, also an indicator to show position of cage, car, or gunboat in shaft or slope. The top of any shaft being sunk (Secs. 16, 18) must be so arranged that no material can fall back into the shaft when the bucket is being dumped; and (Sec. 19) a safety hook or clevis or other safe attachment must be used to attach bucket to chain or rope; and (Sec. 20) guides must be used in shaft to keep bucket from swinging. (Sec. 22) All blasts in sinking a shaft must be exploded by an electric battery. Art. 5 provides for (Sec. 3) safety valves on all boilers; (Sec. 4) a steam gauge in the boiler house; (Sec. 5) guard rails about all machinery. Art. 12 requires (rule 49) safety holes at bottom of all slopes and planes, and (rule 43) on haulage roads of a width not sufficient to allow persons to pass cars with safety; (rule 50) safety blocks at head of all slopes and planes and at top of all shafts.

Ques. 22.—A fan is exhausting 180,000 cubic feet of air per minute from a mine; the area of the port of intake is 60 square feet; what is the pressure necessary to blow air into the fan?

Ans.—Theoretically, assuming there is a perfect vacuum within the fan, the pressure necessary to blow air through the intake orifice of the fan, making the usual allowance for *vena contracta*, is calculated by the formula

$$p = \left(\frac{v}{18}\right)^2 = \left(\frac{180,000}{60 \times 60} \times \frac{1}{18}\right)^2 = 7.7 + \text{ lb. per sq. ft.}$$

This formula, as applied to fans, and the blowing-in or blowing-out pressure of a fan as calculated, have no practical value, however, for the reason that there is not a vacuum within the fan; and further, the relatively large area of the intake orifice, as compared with the area of the passage in fan, and the shortness of the fan ducts greatly modifies the *vena contracta*, and invalidates its application in the formula. The question is not therefore a practical question.

Ques. 23.—How can the dangers attending the firing of blasts in a mine be reduced to a minimum?

Ans.—By faithfully complying with all the requirements of the anthracite mine law, in respect to blasting; and further, by observing due caution when returning to the face after a blast has been fired, or when the explosion of a shot is delayed. When fuse is employed the miner should not return to the face when a shot fails to explode. Fuse has been known to hang fire in some instances for several hours and then explode. This may occur when the fuse is injured in tamping or by some other means so that the trail of powder is broken, and the tape is left to smoulder till it ignites the powder beyond the break.

Ques. 24.—Name the several ages of boys and men mentioned in the Anthracite Mine Law, and the requirements attaching thereto; also, distances specified in the law.

Ans.—The ages mentioned in the law and their respective references are as follows: Age of candidates for office of mine inspector, 30 years or upward (Sec. 9, amendment to Art. 2, approved June 8, 1901). The same requirement applies alike to the Chief of the Department of Mines (Sec. 3, act approved April 14, 1903). Breaker engineers must be at least 18 years of age (Art. 5, Sec. 6); and oilers of machinery at least 15 years (Sec. 8). No boys under 16 years of age may be employed in the mine; and none under 14 years in any breaker or around the outside workings of any mine (act approved May 2, 1905, Sec. 1); and no minor of any age may be employed inside or outside of any anthracite mine, in any capacity, without the filing of an employment certificate (Sec. 3). Hoisting engineers to hoist or lower men must be at least 21 years of age (Art. 12, rule 18); no person under 16 years of age may run a car out of any breast or chamber, or on any gravity road (rule 48). For admission to the Miners' Homes of Pennsylvania, miners must be at least 60 years of age (act approved April 22, 1903, Sec. 7); and wives of miners, at least 55 years (Sec. 8). The distances specified in the mining law are as follows: The two openings of a mine must be not less than 60 feet apart underground and 150 feet apart at the surface (Art. 4, Sec. 1); no inflammable structure may be erected over or nearer than 200 feet to any mine opening (Sec. 5). The hoisting frame over a shaft, in sinking, must sustain the sheaves or pulleys at a height above the tipple not less than 20 feet (Sec. 16), and must be erected before the shaft is sunk deeper than 50 feet (Sec. 17); such shaft when deeper than 100 feet must be provided with guides extending to within 75 feet of the bottom of the excavation (Sec. 20). No boiler may be placed under or nearer than 100 feet to any breaker or building where men are employed in the preparation of coal (Art. 5, Sec. 2). A heading or cross-cut must be driven in every chamber not more than 60 feet from face of chamber (Art. 10, Sec. 15). Any heading approaching a place likely to contain much water may not be driven more than 12 feet wide and a drill hole must be kept 20 feet in advance of such heading (Art. 12, rule 15). Any box containing powder must be kept 10 feet away from all tracks, if possible (rule 27); and no lamp or other fire may be brought nearer than 5 feet to such box when open (Sec. 28). Safety holes on haulageways must not be more than 150 feet apart (Sec. 43); a 2-foot space must be provided on one side of track where sprags are used (rule 47); bumpers must be long enough to keep cars at least 12 inches apart (rule 52).

Ques. 25.—What do you consider to be the dangerous points in a breaker; and what would you recommend to lessen such dangers? State fully.

Ans.—The rolls are a very dangerous part of the breaker, these should be covered as far as practicable, and further guarded by a secure fencing that would not permit an accidental slip of the foot to result in accident. Other dangerous places are conveyers, or scraper, lines, belts, and rope drives, pulleys, or large sheaves. These should all be so protected that accident would be impossible.

Ques. 26.—At a colliery where there are two fans, one blowing, the other exhausting, both working under apparently similar conditions, though producing different results, explain the successive steps you would take to determine which is the better fan.

Ans.—Two fans arranged tandem, the one blowing and the other exhausting on the same current, never give satisfactory results. The fans may be exactly alike in all their dimensions and in form, and run at the same speed, but one or the other of these fans, depending on the natural conditions in the mine, will appear to do practically all the work while the other simply churns the air. If the two fans referred to in the question are exact duplicates in respect to form, dimension, number, and arrangement of blades; and only differ in respect to their connection with the atmosphere and the mine, they must be considered and *are* equally good fans. The question as to whether the exhaust or blowing system should be adopted in any particular mine is a question to be decided by the conditions existing in that mine with respect to haulage and ventilation. In mines where the coal moves with the air on the main roads, the blowing system must be used to avoid the use of doors on the main roads; and, for the same reason, the exhaust system must be employed where the haulage is on the intake airway. In general, conditions permitting its use, a blower is more efficient than an exhaust fan, because working on generally

denser air, but this is not always true. A more practical question would be to ask how to determine what size and speed of fan is best adapted to a certain given circulation.

QUES. 27.—In a certain mine, 100,000 cubic feet of air is passing per minute, under a water gauge of 1.6 inches, and the indicated horsepower of the engine is 50 horsepower, the fan making 50 revolutions per minute; what will be the quantity of air passing, speed of fan, and horsepower of engine, if the water gauge is increased to 3.6 inches?

ANS.—This is another difficult question to answer because the question fails to state the cause of the increase in water gauge. Is it due to an increase in the power of the engine and speed of the fan? If so, the conditions in the mine remaining unchanged, the quantity of air will vary as the square root of the water gauge; or the quantity ratio will equal the square root of the water-gauge ratio, and the increased quantity x will be

$$x = 100{,}000\sqrt{\frac{3.6}{1.6}} = 100{,}000\ \sqrt{2.25} = 150{,}000 \text{ cu. ft. per min.}$$

The quantity ratio is therefore $\frac{150{,}000}{100{,}000} = 1.5$; and owing to a slight decrease in the efficiency of the fan when passing this larger quantity of air the speed ratio is equal to the fourth root of the fifth power of the quantity ratio; or the increased speed is

$$x = 50\ \sqrt[4]{1.5^5} = 50 \times 1.66 = 83 \text{ rev. per min.}$$

Practically the power varies as the cube of the speed; or power ratio = cube of speed ratio; and the increase of power necessary to increase the speed of the fan under these conditions is

$$x = 50\left(\frac{83}{50}\right)^3 = 228 + \text{H. P.}$$

These results will, however, be modified by the adaptation of the dimensions and style of any particular fan to the given speeds. If the fan is designed for a speed of 50 revolutions per minute, its efficiency will be less when run at a higher speed; but if designed for a higher speed its efficiency may increase as the speed is increased, which makes the question unanswerable with data given.

Again, the increase of water gauge from 1.6 to 3.6 inches may be caused by a fall of roof, or other obstruction to the flow of air in the mine, or to a change in the circulation increasing the mine resistance without increasing the area of passage. In this case, the air will be decreased, assuming a constant power, in the inverse ratio of the pressure; or the quantity would be $100{,}000 \times \frac{1.6}{3.6} = 44{,}444$ cubic feet per minute. But this decrease of the quantity of air passing through the fan will increase its efficiency, and the quantity of air, under these conditions, will probably reach 50,000 cubic feet per minute. The power applied to the fan shaft remaining constant, the speed of the fan will be slightly increased under the conditions named, say to, perhaps, 52 or 53 revolutions per minute. Such questions as these being wholly indeterminate should not be asked in a practical mine examination. In this case, the cause of the change of pressure should be given, and its general effect on the quantity of air passing, speed of fan, and power required to produce certain specified results should be asked.

QUES. 28.—(*a*) What is meant by the symbols CH_4, CO_2, CO, and H_2S? (*b*) Why is CO_2 heavier than CH_4 (volume for volume)? (*c*) How are these gases generated and what dangers attend the presence of each? State fully.

ANS.—(*a*) They are the chemical symbols, representing the chemical composition of methane (marsh gas), carbon dioxide, carbon monoxide, and hydrogen sulphide, respectively. (*b*) Because the density of carbon dioxide is greater than that of marsh gas. The density of a gas is one-half its molecular weight; thus, CO_2, $\frac{1}{2}(12+2\times16)=22$; and CH_4, $\frac{1}{2}(12+4\times1)=8$. (*c*) This question is fully answered in reply to Ques. 32, page 576, MINES AND MINERALS, July, 1909.

QUES. 29.—State briefly but fully (*a*) how, when, and where must a mine foreman or his assistant measure the air, and when and how must this be reported to the mine inspector? (*b*) Explain fully each step you would take to satisfy yourself in regard to the quantity of air in circulation, and the system of ventilation employed, in any mine; and how would you determine that the reports made to you were practically correct? (*c*) Explain the anemometer, barometer, and water gauge; and state how each of these instruments is used.

ANS.—Art. 10, Sec. 15, of the Anthracite Mine Law requires the mine foreman or his assistant to measure the air-current with an anemometer or other efficient instrument once every week, at the inlet and outlet airways; also, at or near the face of each gangway, and at the cross-heading nearest to the face of the inside and outside chambers where men are working, on each gangway; such measurements to be entered in the colliery report book. (Sec. 16) A report of these air measurements must be sent to the inspector before the 12th day of each month, for the preceding month, together with a statement of the number of persons employed in each district of the mine. (*b*) Examine the colliery report book and compare with the monthly report sent. Visit the mine without previous notice to the mine officials. Note carefully the speed of the fan as registered on the chart of the recording gauge. Measure the air carefully at each of the points mentioned before; and check the correspondence of the inside and outside measurements on each gangway; and especially the correspondence of the main intake and return measurements and the sum of the measurements taken on the gangways. The system of ventilation should be carefully studied by observing each split of air throughout its course to ascertain if any improvement is possible. This examination should reveal any incorrectness in the monthly reports. (*c*) This question is fully answered in reply to Ques. 8, page 605, MINES AND MINERALS, May, 1910.

QUES. 30.—The return airway in a certain mine measures 6 ft. × 10 ft. in section. The air in this airway has a velocity of 400 feet per minute, and carries marsh gas sufficient to make the air just within the explosive limit. What quantity of air must be added to make it (the gas) detectible?

ANS.—It is difficult to understand what this question intends to ask. There are two explosive limits to the mixture of marsh gas and air; these are called the lower limit (1 volume gas to 13 volumes air), and the higher explosive limit (1 volume gas to 5 volumes air). The gas at both of these limits is very much in evidence and needs no addition of air for its detection. However, assuming the reference here is to the lower explosive limit, for every volume of gas in the mixture of gas and air, there is $1+13=14$ volumes of mixture. The volume of the return current is $6\times10\times400=24{,}000$ cubic feet per minute, which contains $24{,}000\div14=1{,}714+$ cubic feet of gas. Then, assuming that air is to be added till the mixture ceases to give a flame cap in an ordinary unbonneted Davy lamp, which takes place commonly when the mixture contains 2 per cent. of gas, the volume of the current for a 2-per-cent. mixture would be $1{,}714\div.02 = 85{,}700$ cubic feet per minute. To obtain this it would be necessary to add $85{,}700-24{,}000=61{,}700$ cubic feet of air per minute.

CHEDDITE

Cheddite, an explosive that has been extensively used for the past 10 years in Europe, is about to be introduced in Canada. The explosive will not freeze and is practically non-gaseous. It will burn in the open air without explosion. Nitric, hydrofluoric, and sulphuric acids when poured over the powder do not cause it to explode. Nitric acid has no effect on it whatever, but it effervesces under the action of hydrofluoric acid and burns brightly when sulphuric acid is poured over it. When it is charged in a drill hole and exploded the smoke is not injurious and men can go back at once to their working places without even obtaining a headache. This explosive will be manufactured near Haileyburg, Ontario, Can.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

WESTERN EDITORIAL OFFICE: 728 Equitable Building, Denver, Colo.
CHICAGO OFFICE: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to MINES AND MINERALS, Scranton, Pa.
CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER.......................................MANAGER
EUGENE B. WILSON, SCRANTON, PA.......................EDITOR
GEORGE F. DUCK, E. M., DENVER, COLO...............WESTERN EDITOR
P. G. MOORE...CIRCULATION MANAGER
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., LTD., DRYDEN HOUSE, 43 GERRARD ST., SOHO, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 PITT STREET, SIDNEY.
SOUTH AFRICAN AGENTS: J. AFFLECK ROBERTSON, CAPETOWN, MANSION HOUSE; JOHANNESBURG, TRUST BUILDING; DURBAN, NATAL BANK CHAMBERS; BLOEMFONTEIN, FLORENCE BUILDINGS.

SUBSCRIPTION RATES

Single Copies...25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada...2.50
To all other Countries...................................3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always give your FORMER as well as your present address.

TABLE OF CONTENTS

THE WIND ROCK COAL MINE, by W. S. Hutchinson.................... 65
BUREAU OF MINES RESCUE STATIONS.............................. 68
STRIPPING COAL BEDS.. 69
EVOLUTION OF MINE HAULAGE, by E. B. W........................ 71
WORKING A STEEP COAL SEAM, by Austin Y. Hoy.................. 77
THE UNITED STATES BUREAU OF MINES............................ 79
COAL FIELDS OF IOWA AND MISSOURI, by Henry Hinds............. 80
HOW HE EXPLAINED THE B. T. U................................. 82
OBITUARY—
Wright S. Prior—William Smurthwa te......................... 82
CORRELATION THACKER FIELD, by Audley H. Stow................. 83
AUTOMATIC COAL SAMPLER....................................... 85
A WIRE-ROPE CUTTING MACHINE.................................. 85
POWER USED IN MINING, by Edward O'Toole...................... 86
A ROOM-AND-PILLAR METHOD, by A. E. Robinson.................. 88
COAL AND COKE SAMPLING, by E. G. Bailey...................... 89
COAL MINING NEWS... 93
EXAMINATION QUESTIONS—
Mine Inspectors' Examination, Scranton, May, 1910............ 95
EDITORIAL—
Sampling Coal—Southern Gold Mining—Standard English.......... 98
BOOK REVIEW.. 100
CORRESPONDENCE—
Putting Out a Mine Fire—To Find Percentage of Grade, Ventilation—Railroad Rates—Monazite—Siphon.......... 101
THE ELIZABETH TUNNEL, by W. C. Aston......................... 102
THE AMERICAN MINING CONGRESS................................. 105
AMERICAN SOCIETY OF ENGINEER DRAFTSMEN....................... 105
PERSONALS.. 105
LOS PILARES MINE, by M. Robb, Jr............................. 106
THE SUPERIOR AND BOSTON MINE, by R. L. Herrick............... 112
METAL MINING NEWS.. 116
NEW MEXICO GOLD GRAVELS, by J. A. Carruth.................... 117
DETERMINATION OF LATITUDE, by C. E. Rowe..................... 119
TRIAL SMELTER AND LEAD REFINERY, by J. M. Turnbull........... 121
SAFE USE OF ELECTRICITY IN MINES............................. 129
THE FRANZ SCREEN... 126
THE HUBBELL ELECTRIC MINE LAMP............................... 127
LUBRICATING BEARINGS OF HEATED ROLLERS....................... 127
TRADE NOTES.. 128
NEW INVENTIONS... 128
SQUIBS—
Shot Towers, 68—Fulminating vs. White Phosphorus for Igniters, 76—Cheddite, 97—Spitzbergen Coal Mines, 101—Tariff on Sawed [illegible], 111—St. Lawrence Power Project, 120—Product of [illegible]........ 126

SAMPLING COAL

THIS issue of MINES AND MINERALS commences a series of articles on "Coal Sampling" that will appeal to those interested in selling or purchasing coal on the heat-unit basis, and also to those employed in testing and sampling coal so purchased.

To such people the matter of sampling is of extreme importance, for while a fair average sample may be taken from a coal bed, there is little hope of coal reaching the tipple or the consumer in this state of perfection. Hand samples can no more readily furnish a fair average sample of coal than of ore, even should the personal equation that always enters into this method of sampling be eliminated. The method of sampling by stretching the knotted rope lengthwise of a car is also faulty, for even should the sampler take a weighed quantity from under each knot, the sample fails to furnish an approximate idea of what is underneath that taken.

The present state of perfection in the mechanical sampling of ore was reached after years of experimenting. It is not to be expected therefore that those unacquainted with rules governing sampling can at once design a mechanical coal sampler. One of the first principles established in mechanical ore sampling, and one which applies equally to coal, and to which coal sampling must conform, is that a fair average sample cannot be taken where the whole of the stream of ore is taken part of the time. Grab samples, rope samples, and the mechanical sampler illustrated on page 85 take the whole stream of coal part of the time, and according to the rule are unreliable methods of sampling.

SOUTHERN GOLD MINING

THE general impression prevails that southern gold properties are more precarious investments than even those in the West. It is true that the record of southern gold mining has not been such as to inspire confidence, yet there are substantial reasons why this is so that ought to be investigated before condemnation. While gold was mined in Virginia, North Carolina, South Carolina, and Georgia before the Civil War, it was done by planters who used their slaves between crops to wash gold in a desultory way. Up to the present time, with two or three exceptions, no systematic quartz mining has been undertaken. One of the exceptional cases is that of Adolph Theiss, who, when he encountered refractory ore improved the old chlorination process and gave the world the perfected one now in use. The majority of the quartz mine enterprises were failures owing to refractory ore, which at the time they were being developed could not be handled by mills and there were no nearby smelters to which it could be sent. The South then became the "mecca" of the patent-process man, which further blackened its reputation. In many cases if the ore be crushed and panned the gold can be seen, and this alluring bait induced people of small means to erect stamp mills

to recover the gold by amalgamation. Stamp mills having proved abortive in the hands of eastern men, expert stamp men were brought from the West, who also failed utterly to recover the gold. The Western mill men would then introduce a Huntington mill or possibly an arrastra under the belief that this surely would recover the gold, and because it did not they, too, condemned the field.

The adventurers by this time, either from lack of capital or faith, would become disheartened and shut down, or if possible sell to some sure-thing man, and on the latter's failure another black mark was chalked up to southern gold properties.

Since the war the Sawyer Mine, in North Carolina, has changed hands several times. It has high-grade ore that will run up to $100 or more, and to recover the gold, arrastras, Huntington mills, grinding pans, stamps, bumping tables, buddles, and possibly other contrivances, have been unsuccessfully employed. There is no question but that this mine, if located in Gilpin County, Colo., would have been a great success. Most southern gold mining adventures have been made on so small capital and under such conditions that $100 ore could not, in some instances, make them succeed, yet the ore is not more refractory than some Colorado ores.

There have been no large investments to build up the industry in the South, although the ore is there and can be as readily recovered at this time as in the West. While a number of things have injured the mining reputation of the South, probably none have greater influence in condemning the gold properties from a business point than the legal guerrilla-warfare litigation that can pursue non-residents through the courts, and which is instituted by shyster lawyers, with no expense to their clients, in the hope that to escape long years of litigation the non-residents will compromise.

One of the most interesting attempts to reestablish gold mining in the South is now being made by Leo Von Rosenberg, a well-know mining engineer in New York City, who has opened several tracts of gold-mining lands comprising more than 1,000 acres in Georgia, and subdivided them into smaller lots of 1,500 ft. × 600 ft. each. These claims he is throwing open to prospectors for development under lease and bond. The company owning this property has more land than it can develop, and mining conditions are exceptionally favorable, there being plenty of water, perfect climate, permitting work throughout the year, and all the timber necessary for mining purposes. The idea is to give each prospector all the ore within the lot to a depth of 100 feet, or, as the case may be, 150 feet. All such ore as he may find worthy of extraction he may do with as he pleases. The company has a milling plant and will be ready to mill any ore under certain agreements. After the prospector has reached the prescribed depth he will be at liberty to buy the claim or continue mining by paying a royalty on all ore removed. The ownership of large tracts of mining lands by individuals or companies in the South has prevented thorough and systematic vein prospecting.

Mr. Von Rosenberg believes that, if the above proposed plan were more generally followed in the South, it would probably result in opening up a number of paying mines. As it is, large areas are tied up with no work being done. If this land were situated in any of the mining sections of the West and opened for location, hundreds of claims would be staked out and an army of prospectors would be at work digging for ore.

STANDARD ENGLISH

THERE is in North America an established English language replete with mining terms to such an extent that educated mining engineers can judge nearly the caliber of the man writing on mining subjects.

As an illustration, a friend handed an experienced mining engineer a prospectus, with the remark that he was capable of reading between the lines and telling whether the venture was material or visionary. After reading a few minutes the engineer said: "I know neither the property nor the promoters, but this I do know, the man who furnished the stuff in this prospectus was illiterate, and not a mining engineer, although probably a colonel or 'mining expert.'"

This diagnosis was correct, as all those who invested in the venture now declare.

The experienced engineer is able to tell if he is reading an article by a young enthusiastic engineer or an entirely theoretical office man, from the words employed. The young engineer, while he may be technically correct in his writing, will almost invariably interject a copious supply of adverbs and a number of localisms to show he is a man of experience. The office engineer will furnish a long diatribe on immaterial things and straddle the material matters, so that he can fall safe either way; in other words his inconclusiveness marks his deficiency, and to cover this, technical verbiage and adverbs abound. Mining matters can be stated usually in such plain English that a layman with a little help can understand them; and as most mining engineers write for laymen it is good policy to return to first principles.

Frequently some one says language must conform to usage. "Mama, the iceman have came" is the established vernacularism in some parts of New York City; however, if you accept it there is no particular reason why you should have studied grammar; and when a mining engineer writes of coal *veins* he puts himself in the same class, for he never was taught in his mining school that there were coal veins. So far the people in these United States have spoken better English with fewer dialects than the inhabitants of the British Islands and there is no reason why they should adopt old English localisms that originated in fun or ignorance. We hear of "inbye" and "outbye." The former had

its inception from some old miner who remarked "are you going in, bye," instead of "are you going in the mine, boy," and "outbye" followed naturally. We hear also of "pit wagons" meaning mine cars, and if it continues we shall hear of tubs, and boxes, which in England have the same significance, according to the locality. Tubs were used in coal mines, as were boxes and wagons, before rails were introduced. They were probably never used in America, although mine cars have been an established part of mine equipment.

Pit means, in our idiom, a deep hole or vertical excavation, not a horizontal excavation, yet men who know better will speak of a shaft mine, and will call a drift mine "a pit," and drift portals "pit mouths." Another innovation is "mine workings," which has about the same significance as "old workings" in that it shows that the mining engineer that uses it is coming down to the lazy tongued miners and not raising the miners' intellect.

As T. A. Rickard said in his article on the "Standardization of English": "When new words are used to express new ideas there is no objection to their introduction." It is, however, a serious perversion of American English for an educated engineer to adopt localisms introduced by some one to represent an old thing which to him was new; and wherever they are used it shows a lack of experience and presumably ignorance in mining.

As previously stated, the experienced mining engineer estimates pretty closely the caliber of another mining man by his written expressions, for which reason, if no other, care should be exercised by mining engineers in the use of *mining terms*.

Book Review

Coal Cutting With an Electric Puncher, by John L. Wagner. This pamphlet is illustrated and is the gist of the lecture given by Mr. Wagner before various societies on the comparative advantages of the different methods of undermining coal, and the most economical application of power to this kind of work.

Who's Who in Mining and Metallurgy, 1910. Founded by George Safford. Published by *The Mining Journal*, London, England. Price 15 shillings.

This is the second edition and is much enlarged. Of the 272 pages, 147 are devoted to biographic paragraphs, and 125 to technical and professional societies. This book was not originated as a biographical hold-up, in other words you do not have to purchase it, although if you do the exchange will not be regretted, for it is the Bertillion system of the engineer and the editor's Rogue's Gallery. "Who's Who in Mining and Metallurgy" took its first list of mining and metallurgical men in America from "Who's Who in America." The second edition, however, increased the number of names considerably, and while it is not yet thoroughly representative it will become so in a few more editions.

There is no professional advertising, but there is a record of engineers which will greatly interest other engineers and will be found of service and satisfaction For example, some one writes on a mining or metallurgical subject. The reader wants to know more of the man and therefore refers to his past in "Who's Who in Mining and Metallurgy."

Map of West Virginia, showing coal, oil, gas, and limestone areas, I. C. White, State Geologist, Morgantown, W. Va. Map can be procured by sending 50 cents to the Geological Survey of West Va.

Proceedings of the Engineering Association of New South Wales, Vol. 24, 1908–9. Contains a number of valuable papers. Those that will appeal most to mining engineers are: Labor-Saving Devices, by G. A. Julius, B. Sc., M. E.; Description of a New Zealand Coal Mine, by C. J. Holroyde. The other papers are: Coolgardie Water-Supply Scheme, by R. J. Vincent; Water at Coolgardie cost from £4 to £1 5 shillings per 1,000 gallons. This paper describes the method of installing a water supply and overcoming obstacles eventually reducing the cost to 2 shillings. It is both interesting and instructive.

Algea in Reservoirs, by James Faulkiner; 'Modern Methods of Steamship Construction, by Walter Reeks; Commercial Motor Vehicle, by E. F. Boult; Design and Use of Speed-Reducing Gear, by Henry Shaw and James McNamara.

Henry V. Ahrbecker, M. I. Mech. E. secretary, 5 Elizabeth Street, Sydney, New South Wales.

Principles of Chemical Geology, by James Vincent Elsden, D. Sc., F. G. S., is made up into 222 pages, 10 mo, with index. There are 10 chapters and 44 diagrammatic illustrations. Published by The MacMillan Co., New York City, and Whittaker & Co., London, England. Price $1.60 net.

The main object of the book is to present in a concise form the application of the principle of equilibrium to certain geological phenomena. To the student of geology nothing is at first more bewildering than the multiplicity of changes which rocks and minerals undergo in nature. It is to chemical and physical laws that we are accustomed to look for an explanation of these transformations; but until comparatively recently many of these explanations were vague and hypothetical, owing to our ignorance of these laws. To elucidate and stimulate interest in this branch of geology, Mr. Elsden has divided the subject into 10 chapters, of which the following are synopses: I, Equilibrium between the crystalline and amorphous states, based on Le Chatelier's principle that "any external change in the factors of equilibrium of a system is followed by a reverse change within the system." II, Equilibrium influenced by viscosity plays an important part in retarding the transition of minerals. III, Diffusion as a factor of equilibrium in geological transformations may be due to gases, liquids, or solids. IV, Surface tension as a factor of equilibrium is considered as one of the most important features in supersaturation. V, Vapor pressure as a factor in equilibrium is based on experiments that show that many solids possess an appreciable vapor pressure at moderate temperatures. VI, Equilibrium conditions of polymorphous forms, that is where transformation of a mineral from one crystalline form to another occurs with differences in specific gravity, melting point, and other physical properties. This subject possesses geological interest owing to the effects produced by dynamical metamorphism. VII, Equilibrium in solution. From the point of view of the phase rule, unsaturated solutions are trivarient systems that possess three degrees of freedom, and can only be defined by the three variable factors, pressure, temperature, and concentration. VIII, The eutectic theory is elucidated from ice to its projection on a plane triangle, making it necessary to examine how far the eutectic theory may apply. IX, The theory of solid solutions applied to geological problems, covers 23 pages, and forms one of the most interesting chapters. X is on the conditions of chemical equilibrium in geology. The book is interesting throughout and will appeal to the advanced mineralogist, physicist, chemist, and geologist. The subject is large and Mr. Elsden has covered it admirably, without entering the domain of geochemistry.

CORRESPONDENCE

Putting Out a Mine Fire

Editor Mines and Minerals:

SIR:—I would like to hear from your readers regarding the means and methods they would follow in putting out the fire

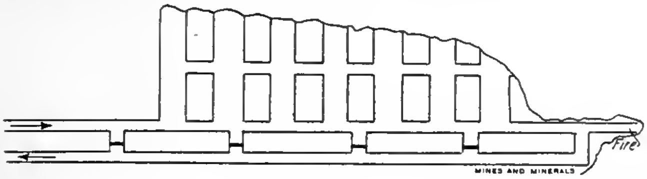


FIG. 1

shown in the accompanying Fig. 1. The fire is on level workings in a shaft mine 130 feet deep. It started at the face of the entry where shown. The rooms give off gas. H. P. J.

Scranton, Pa.

To Find Percentage of Grade

Editor Mines and Minerals:

SIR:—Please answer through your Correspondence Column the following question: What per cent. grade does a slope represent that is driven at an angle of 10 degrees?

Denver, Colo. SUBSCRIBER

ANS.—Look in the table of natural sines and cosines in Coal and Metal Miner's Pocketbook, and find sine 10° = .17365. This is the grade for 1 foot. As per cent. means by the hundred, 17.365 is the per cent. grade.

Ventilation

Editor Mines and Minerals:

SIR:—Can any of your subscribers give a formula for the following problem? A pair of single entries each 1,000 feet in length are ventilated by pipes 18 inches in diameter; at a dis-

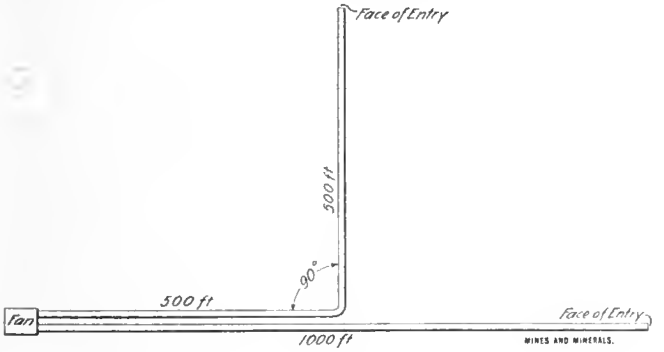


FIG. 2

tance of 500 feet, as shown in Fig. 2, there is a branch pipe 500 feet long at right angles. Can the same quantity of air be obtained in each entry, the rubbing surface and pressure being the same? G. E. D.

Dawson, N. Mex.

Railroad Rates

Editor Mines and Minerals:

SIR:—This side of the Rockies the railroads sure love the dear public. A ships a load of $50-per-ton ore; B ships the same amount of $300-per-ton ore. Because the ore of B is richer than that of A, he is compelled to pay higher freight rates. What kind of railway companies have you in the East? Must a rich man pay more per mile for traveling than a poor man?

Jawbridge, Nev. J. H.

The railroads in the East are even worse to their mining patrons than those in the West. There are several reasons why railroads charge a higher rate for rich ore. They are responsible to the shippers and must pay more damages in case of loss of richer than for the poorer ore. There is more loss in smelting the richer than the poorer ore, for which reason the smelter charges are more. The rich man is supposed to pay as much as the poor man for traveling, but he need pay no more unless he is so exclusive as to travel in a private car. In case of accident the heirs of the rich man would receive more damages than would the heirs of the poor man, a principle of justice (?) the reverse of freight rates on ore.—EDITOR.

Monazite

Editor Mines and Minerals:

SIR:—What is monazite? Where is it found and what are its associate minerals? How can it be detected? Has it any commercial value? W. SAMPSON

Singapore, India

Monazite is a complex phosphate containing cerium, lanthanium, didymium, and thorium. It is found in granitic rocks in North Carolina, Brazil, and elsewhere, in the form of sand, having a yellow, clove-red, or brown color. Thorium is the valuable part of the mineral and is used in the manufacture of gas mantels of the Welsbach type.

Monazite has basal cleavage; resinous lustre; a hardness, 5 to 5.5; and a specific gravity of 5.2 which heaviness causes it to collect as sand bars in running streams. Fused with soda, the mass treated with water, and filtered, the residue dissolved in hydrochloric acid, the solution gives with oxalic acid a precipitate which ignited, becomes brick red (oxide of cerium). The commercial value of monazite is not so great as to induce people to go into the business unless they have large rich deposits, easily recovered.

Siphon

Editor Mines and Minerals:

SIR:—Referring to the siphon problem in the July number, and the answer thereto, Messrs. J. L. P. and W. H. Jones are both wrong, as the effective head drawing water into the siphon, and which consequently will govern the amount discharged, is the degree of vacuum obtained at the summit of the pipe.

The velocity of the water in the shorter leg will determine the quantity discharged, and the head to produce this velocity cannot be greater than the height to which a column of water will be raised by the pressure of the atmosphere at the given locality.

The discharge cannot be calculated from the data given, as the distance from the surface of the water to the summit of the pipe is not given. ALONZO G. COLLINS

Philadelphia, Pa.

SPITZBERGEN COAL MINING

Consul-General Henry Bordewich, of Christiania, furnishes the following information concerning the coal mines at Advent Bay, in the Arctic island of Spitzbergen, in which Americans are interested:

The steamer Munroe, belonging to the coal-mining company, passed Tromso, northern Norway, bound for Spitzbergen with supplies, on May 18, and on May 30 entered that port on its return trip, having accomplished the voyage in 12 days.

The officers report that among the 100 men who passed the winter at work in the company's mines at Advent Bay there was very little sickness. The winter was unusually mild, and the work progressed under the most favorable conditions. About 8,000 tons of coal, pronounced of excellent quality, have been extracted and made ready for market. The excavations are now so deep that work in the mines can be carried on regardless of weather and seasons. Suitable buildings have been erected, and substantial wharves are under construction.—*United States Consular Report.*

THE ELIZABETH TUNNEL

*Written for Mines and Minerals, by W. C. Aston**

Methods by Which Highest American Hard Rock Tunnel Record Was Made

The city of Los Angeles, in California, is constructing an aqueduct about 217 miles long. Considerable of this work requires tunneling; in fact there are 105 tunnels whose aggregate length is 28 miles. The Elizabeth tunnel, which is 26,860 feet long, is being driven from both ends, termed the north and south portals. This tunnel has a cross-section 12 ft. 4 in. × 12 ft. 9 in., a grade of 1 foot in 1,000 feet, and a water capacity of 1,000 second feet. At the present time over 11,000 feet have been advanced from each end, and the work is being pushed forward at remarkable speed.

Upon opening the portal the ground was found to be soft, requiring timber, but as the work progressed, it rapidly hardened,

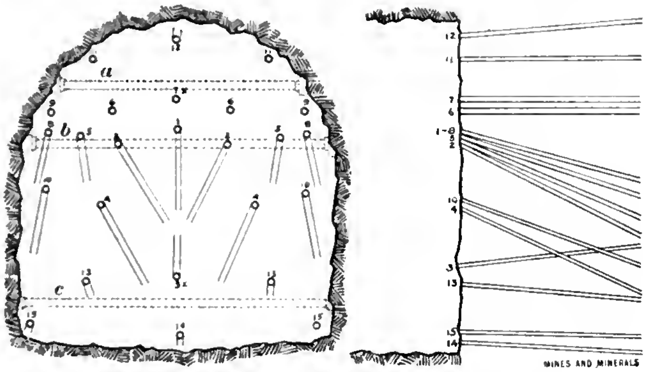


FIG. 1. ROUND OF SHOTS FOR MEDIUM HARD GROUND

and, with the exception of a few belts of softer material at irregular intervals, has remained hard, passing through many stages of massive, blocky, stratified, seamed, and folded formation, and varying through many alterations of granite, gneiss, and schist.

To be able to approach the same rate of speed through the hard rock at the south end that could be maintained through the softer rock of the north end, and at the same time not materially increase the cost, it was decided to systematize the work to a point where the operation would become as nearly mechanical as possible, leaving it elastic enough to admit the unavoidable breaking in of green men and enlarging or contracting the size of a shift. To do this it was decided to assign a specific duty, as far as possible, to each man in the heading, which that man was required to do every day when conditions were normal without waiting orders from the shift boss. All other men working in the tunnel, but not directly in the heading, were placed upon call by the shift boss for work in the heading when necessary, and all surface laborers except compressor men and blacksmiths, were placed upon a reserve list to draw from in an emergency or when ordered by the foreman.

The track boss reporting to the foreman, but working in conjunction with the shift boss with a crew varying from three to seven men and working only during the day shift, puts down all track, air, water, and ventilating pipe, cleans the ditch and keeps the dump in shape. The master mechanic is in charge of all outside labor and is held responsible for the upkeep of all equipment and tools, except the electrical, which are handled by an electrician. One mechanic overhauls all machines, hose, and heading tools for the three shifts, another one repairs all cars, replaces broken wheels and draw-heads, and oils or greases them. The first requisite for rapid work was a good track. The main line both inside and outside the tunnel was centered, ballasted, and connected in the yard by switches and sidings to the powder magazine, timber yard, saw mill and repair shop, using 30-pound rails, stub switches, and ground-throw switch stands.

*Superintendent of South Portal, Elizabeth Tunnel

All timber, lumber, pipe, rail, and heavy supplies were piled so that they could be unloaded from the freight wagons directly on to the piles and loaded direct on to the cars from the opposite side.

The general equipment varies but little from that in use at other tunnels being driven at the present time, and consists of the following:

Two compressors, 520 feet capacity, belt driven from electric motors; two motor-generator sets, 150 horsepower; one 50-horsepower electric locomotive; one 30-horsepower electric locomotive; nine water Leyner drills; 38 rocker dump cars, 32 cubic feet capacity; one drill sharpener.

In addition, the machine-shop equipment included a lathe, drill press, saws, blowers, motors, and the necessary tools for such work. Most of the destructible equipment was supplied in duplicate, and extra machine drills were furnished. Each shift was supplied with a tool box and all the tools necessary for its members' work. These tool boxes were locked and one man on each shift was held responsible. A station was cut in the tunnel where all repairs to machines, hose, tools, etc., were made. Wherever possible each individual is held responsible for the tools he uses.

When going on shift under normal conditions all tools, replacements, drinking water, etc., are loaded on a flat car attached to the end of the train and hauled with the men from the portal to the repair station inside where the machines, manifold, and hose are loaded on. Upon reaching the heading the six machine men pick down the back and sides, and shovel back only enough material to allow of placing the arch bar *a* and upper main cross-bar *b* in position as shown in Fig. 1. The nine shovelers immediately string out along the tunnel, cleaning track, carrying forward the tools, unloading the steel and drills, and piling material where it can be readily reached. A reel holding a double-conductor, armored, extension electric light wire is carried to the end of the permanent wire, unwound and dragged to the heading, and connected to a cluster of lights

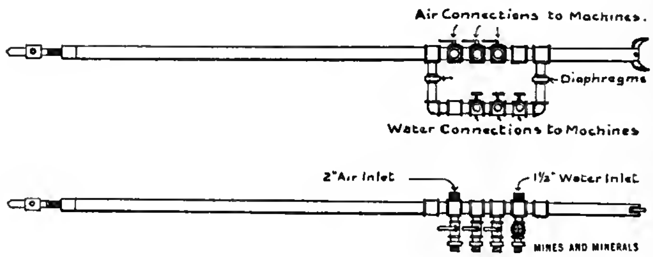


FIG. 2. COMBINATION MANIFOLD FOR AIR AND WATER

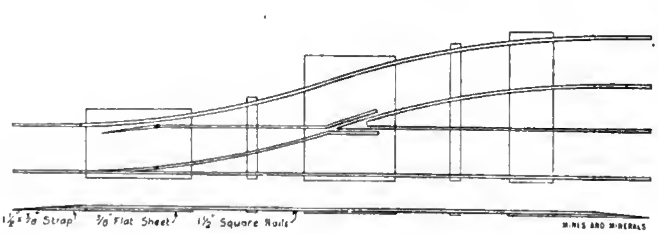


FIG. 3. PLATFORM SWITCH

which is fastened where it will give light to both machine men and shovelers. A compound manifold, shown in Fig. 2, carrying both compressed air and water, and mounted on a light screw cross-bar at a point as close to the drills as convenient, is next erected by the laborers and connected to the main pipe line by 100 feet of 2-inch armored air hose, and the same length

TABLE 1. SOUTH PORTAL ELIZABETH TUNNEL. DETAILS OF ANNUAL SUMMARY OF TUNNEL REPORTS FOR 1909

	Totals	Units	Total Cost	Unit Cost
Footage to date	7,585 feet			
Footage during 1909	4,476 feet			
Required progress	2,920 feet			
Daily footage (365 days)		12.26		
Required daily footage		8.00		
Increased footage over estimate	1,556 feet			
Daily increase over estimate		4.26		
Footage of untimbered section	3,274 feet			
Estimated cost of untimbered section			$131,385.62	$40.13
Actual cost of untimbered section			121,787.45	37.198
Footage of timbered section	1,202 feet			
Estimated cost of timbered section			62,491.98	51.99
Actual cost of timbered section			51,941.11	43.20
Average cost of timbering per foot progress				1.16
Total bonus footage	1,563 feet			
Total bonus pay roll			23,440.23	
Average cost of bonus per foot progress				5.23
Estimated expenditure			193,877.60	
Actual expenditure			173,728.56	
Amount saved over estimated expenditure			20,149.04	
Number of shifts worked	1,055			
Number of shifts lost	40			
Number of men days	21,452			
Men per foot progress		4.79		
Average progress per shift		4.24		
Number of holes drilled	21,066			
Total feet drilled	135,896			
Feet drilled per foot progress		30.36		
Total time drilling (in hours)	3,656.25			
Average time drilling per foot progress (in minutes)		49.01		
Average time drilling one hole 1 foot deep		1.61		
Pounds of powder used (including trimming)	143,659			
Average pounds per foot		32.09		
Number of cars mucked (32 cubic feet)	28,561			
Cars mucked per foot		6.40		
Leyner No. 9, drill repairs (3 machines per shift)			4,116.99	
Average cost of repairs per machine per foot				.30
Leyner No. 2 drill sharpener repairs (for 3,300 feet)			387.13	.117
Energy used, (kilowatt hours)	608,762	135.00		

TABLE 2. SOUTH PORTAL ELIZABETH TUNNEL. SUMMARY OF TOTAL TUNNEL EXPENDITURE DURING 1909

		Totals	Unit Cost
E. W. O. 333—A.	Engineer and superintendent	$ 3,851.24	.86
B.	Excavation	79,607.15	17.78
C.	Mucking	55,556.33	12.41
E.	Drainage	421.01	.09
F.	Ventilation	2,313.44	.52
G.	Light and power	24,750.71	5.53
	Total cost untimbered	166,499.88	37.19
E. W. O. 333—D.	Timbering	7,228.68	6.01
	Total cost of timbered tunnel	$173,728.56	$43.20

TABLE 3. SOUTH PORTAL ELIZABETH TUNNEL. DETAILS OF TUNNEL REPORTS, APRIL, 1910

	Totals	Units	Total Cost	Unit Cost
Total footage to date	9,738 feet			
Footage during April, 1910	604 feet			
Required progress	240 feet			
Daily footage (30 days)		20.13		
Required daily footage		8.00		
Increase over estimated footage	364 feet			
Daily increase over estimate		12.13		
Footage of untimbered section	604 feet			
Estimated cost of untimbered section			$24,238.52	$40.13
Actual cost of untimbered section			15,257.61	25.25
Footage of timbered section	000 feet			
Bonus footage	364 feet			
Bonus pay roll			2,377.48	
Cost of bonus per foot				3.93
Energy used—kilowatt hours	78,379	129.76		
Cost of energy, at $.0185 per kilowatt hour			1,450.02	2.40
Estimated expenditure			24,238.52	
Actual expenditure			15,257.61	
Amount saved over estimated expenditure			8,980.91	
Number of underground men days:				
Foreman and heading crew	1,860	3.079		
Timber, pipe, track, car, and machine repair men	458	.758		
Mechanics, electrician and helpers	154	.255		
Number of mule days	90		81.00	.134
Number of shifts worked	90			
Average progress per shift		6.711		
Number of holes drilled	1,924			
Total feet drilled	16,079			
Feet drilled per foot progress		26.62		
Total time drilling heading (hours)	324			
Average time drilling heading per foot (minutes)		32.18		
Average time drilling one hole one foot (minutes)		1.209		
Pounds of powder used (including trimming)	16,100	26.65		
Number of cars mucked (32 cubic feet) heading and ditch	3,216	5.324		
Drill repairs (three No. 9 Leyners)			248.77	
Average cost of repairs per foot (three machines)				.411
Average cost of repairs per foot (one machine)				.137
Drill sharpener repairs (Leyner No. 2)			61.70	.102
Drill steel broken	380	.629		
Drill steel sharpened	6,865	11.36		
Drill steel welded	445	.736		
Cars repaired	38	.062		
Car repairs			48.52	.08
Car equipment (changing from 12-inch to 14-inch wheels)			260.68	.431

NOTE.—New American hard rock tunnel record established April, 1910, 604 feet.

Table 2a.—Summary of General Expenditures During 1909

	Total Cost
Miscellaneous structures	165.79
Tunnel construction	15,348.27
Miscellaneous construction equipment	5,168.85
Miscellaneous camp equipment	19.22
M. & O. live stock	183.75
M. & O. local telephone lines	31.73
M. & O. water supply	2.34
Division administration	740.86
M. & O. roads and trails	116.05
Total expenditure	$21,776.86
Pay roll	9,419.76
Bonus roll	2,377.48
Material issues	6,915.83
Material receipts	2,841.54

Table 4.—Summary of Tunnel Expenditure During April, 1910

			Totals	Units
E. W. O. 333—	A.	Engineering and superintendent	$ 162.89	.27
	B.	Excavation	7,163.30	11.86
	C.	Mucking	5,275.78	8.73
	E.	Drainage	44.49	.07
	F.	Ventilation	85.10	.14
	G.	Light and power	2,109.55	3.49
			14,841.11	24.56
	D.	Timbering	90.66	
	K.	Back trimming	416.50	.69
		Total expenditure	$15,348.27	25.25

of 1½-inch armored water hose, and to the other side of the manifold three double lines of small air and water hose are connected with the drills.

When the two cross-bars are up, one drill is mounted on the arch bar *a* where it drills three holes, the other two machines are mounted on the upper main bar *b* and drill 15 holes between them. For a while nippers passed the steel, but they were discarded, as it was found that it could be passed readily by the shovelers while the cars were being switched. The nine laborers work in squads of three while loading the cars, one squad loading into the 32-cubic-foot rocker dump cars, shown in Fig. 4, another squad picking and loosening the dirt ahead of them, and the remaining squad resting near the switch.

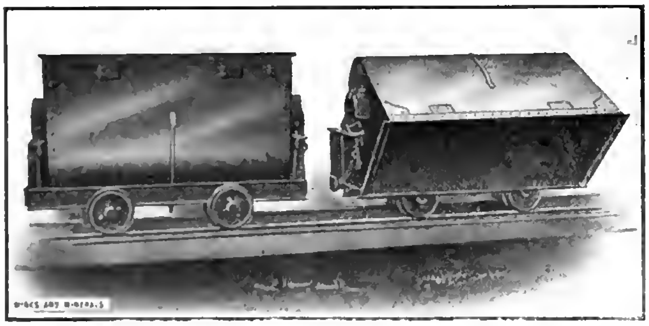


Fig. 4. Rocker Dump Cars Used in Tunnel

From this point the resting squad pushes the empty car as soon as the loaded one is hauled past them by the mule to a point where it is picked up by the electric locomotive and hauled with others to the surface in trains of from 12 to 15 cars each.

The main track and siding are both laid without being cut for frogs, and are connected by a platform switch, shown in Fig. 3. The tracks are made of 1½-inch square iron and riveted to ⅜-inch plates and straps, while the ends are tapered so the cars pass over them with but slight jolt. This switch is carried ahead from time to time and is always kept within 200 feet of the heading. Rails placed on their sides inside the permanent rails with the ball faced against the web of the permanent rail are used for extension track. Ties are laid as rapidly as the dirt is cleared away and these temporary false rails are pushed ahead over them. When the full length of rail has been pushed forward, the track crew cleans off the ties and as a loaded car leaves the temporary track, one rail is turned over to an upright position and both ends spiked before the returning empty car reaches it. While this latter car is being loaded, the remainder of the rail between the car and the permanent track is spiked, and the process repeated with the other rail, without holding up the loading for an instant.

Flat steel sheets are laid over the floor of the tunnel for a distance of 40 feet back from the heading. When the dirt is out, the upper main bar, *b*, Fig. 1, is lowered to its second position *c*; when the drills are late, a second long bar is placed as close to the ground as the operating of the machines will permit, and as each drill finishes its upper hole it is lowered down to this bar and drills the lifters, and if the ground is hard enough to warrant it, also drills two relievers and an up-cut hole. As soon as the upper bar is clear of the holes the blow pipe cleans out the holes drilled and by loading from top and both sides at once the fuse is often spit within 15 minutes after the air has been shut off from the last hole.

As the last change is made on the drills, the manifold is dropped to the floor, a man with a wrench goes to either end and as each machine finishes, the small hoses are uncoupled from the manifold, coiled, and placed on the car which stands just behind already loaded with the dull steel and tools. Machines, clamps, bars, blocking, and jack-bars, each go back in turn and as the last one clears the heading the mule starts away with the car in time to escape the dirt from the blow pipe, as it cleans out the lifters.

The powder charge varies from 15 to 40 pounds per foot of advance, according to the hardness or toughness of the ground.

The men now string out at regular intervals along the 100 feet of main air and water hose and carry them back to the point of safety, such pipe being too heavy to coil and place on the car. They then continue back to the cars, store their small tools in the tool box, board the cars, and wait for the machine men to spit the holes.

A No. 5½ Root pressure blower draws the powder smoke out through an 18-inch pipe until the heading is nearly clear, when the blower is reversed and fresh air blown in, about 15 to 20 minutes being required to clear the heading of gases.

Hinged tables are fastened against the sides of the tunnel and on the 7 a. m. and 3 p. m. shifts hot meals are spread for the crews. These meals are placed in a tight box in the mess-house kitchen and not opened until ready to serve. On the 11 p. m. shift the lunches are served cold.

With slight modifications as conditions changed, or equipment was replaced or improved, this system has worked very satisfactorily during the past year and up to the present time. It worked so well in fact that during the month of April of this year when both labor and machinery were at their highest state of efficiency the full heading complete was driven the remarkable distance of 604 feet, through a rock that was hard enough to require as many as 27 holes per round to break.

Each shift is required to drill and blast, the length of round being regulated by the nature of ground encountered in the first hole drilled.

Discipline and strict attention to business while on shift are required of every one, while at the same time a spirit of friendliness is fostered and every man made to feel that in a large measure the success is due to his own efforts and the interest he takes. We attribute the present efficiency of the men and progress at this portal to the above policy as well as to the close adherence to detail and system which is demanded of all concerned.

A bonus of 40 cents per foot per man for every foot driven over 2⅔ feet per shift is being paid, bringing wages up to a good figure above the scales usually paid in other mining camps and consequently giving us a better grade of men than could otherwise have been obtained.

Tables 1 and 2 give the heading base units for 1909, during which time 40 shifts were lost, but still the average of 365 days was 12.26 feet daily. Table 2*a* shows the general expenditures during 1909.

Tables 3 and 4 show the heading base units for the month of April, 1910, when 604 feet were driven in 30 days, and not only broke the American record for that time, but also the world's record for driving a full heading without bench, wings, or galleries.

PERSONALS

Frederick Burbidge, of Spokane, Assistant General Manager of the Federal Mining and Smelting Co., of Idaho, has resigned. He was general manager of the Bunker Hill and Sullivan Mining Co. for several years.

Fremont S. Rowe and F. Cushing Moore have been examining the Black Bear Consolidated Mine in the Coeur d' Alene district, Idaho.

H. C. George, of the Wisconsin State Mining Trade School, spent July in the Lake Superior copper and iron districts.

A. E. Seaman, of the Michigan School of Mines, has been studying the geological situation of Santa Eulalia, Chihuahua, Mexico.

J. F. Jones has been transferred from St. Louis to Birmingham, Ala., where an office has been opened for him in the Woodward Building by the Wagner Electric Mfg. Co.

Prof. E. C. Holden supervised the four week's work of the mining engineering class of the University of Wisconsin, in Idaho, this past summer.

J. P. Hutchins has A. P. Rogers, Norman Stines, and Ross Hoffmann, as assistants in testing the alluvial deposits in the Nerchinsk district, Siberia. He writes: "Doing business through an interpreter is bad, for he usually does it all for you and only tells you how he arranged it."

Bert. M. Meadows, for several years connected with the Charlotte Supply Co., of Birmingham, Ala., has joined the selling force of the J. Geo. Leyner Engineering Works Co., assisting their Birmingham representative, K. B. Stephens, in handling their drill and drill-sharpener business in the southeastern states.

C. K. Hitchcock, E. M., formerly with the Quincy, later with the Adventure Mining Co., has been made manager at the Lake copper mine of the Copper Range Consolidated.

George T. Holloway announces the removal of his office and laboratories from Chancery Lane to 9–13 Emmett Street, Limehouse, London, E.

Thomas E. Lambert has accepted the position of master mechanic for the Giroux Consolidated Mines Co., at Kimberly, Nev.

J. B. Tyrrell, of Ontario, Canada, has left for a short visit to London, England. His address in London will be, care of Mining and Metallurgical Club, St. Ermin's, Westminster, London, S. W., England.

Edgar G. Tuttle, E. M., has opened an office at 30 East Logan St., Germantown, Philadelphia. He will examine and report on mining properties, design mine and concentration plants and supervise the construction and installation of mechanical and electrical equipment, water-power development, etc.

THE AMERICAN MINING CONGRESS

The American Mining Congress will meet in thirteenth annual convention in Los Angeles, Cal., September 26, 27, 28, 29, 30, and October 1, 1910.

The call for the convention states the purpose to be to bring the representatives of the mining industries of America into conference for the public discussion of national and international questions which concern the welfare and progress of mining and allied industries.

The Los Angeles convention will be composed of the regular members of the American Mining Congress, the accredited delegates appointed by the respective appointing authorities, invited guests, and speakers.

Delegates may be appointed as follows: The president may appoint 10 delegates at large; governors of states and territories may each appoint 10 delegates; mayors of cities and towns, two delegates each; boards of county commissioners, two delegates each; boards of county supervisors, two delegates each; Boards of Trade, two delegates each; Chambers of Commerce, two delegates each; mining bureaus and exchanges, two delegates each; mining organizations, two delegates each; scientific societies, two delegates each; engineers' associations, two delegates each; state mining schools, two delegates each.

It is urged that the name and post-office address of each delegate appointed be forwarded promptly to the secretary, American Mining Congress, Denver, Colo., that advance information as to the subjects to be discussed and matters to be acted upon may be furnished them.

The American Mining Congress is an incorporated body, and only members of the organization can legally vote upon such matters as relate to the permanent business affairs of the congress, the control of which is lodged in a board of directors consisting of nine members, three of whom are elected annually, to hold office for 3 years.

The board of directors will be largely guided by the resolutions adopted by the congress in annual session, which body is composed of members and duly accredited delegates, and will maintain a working force continuously engaged in carrying out the directions of the congress as expressed in resolutions adopted at its regular sessions.

In the regular deliberations of the congress, the introduction and discussion of resolutions and other matters, serving upon convention committees, and in every phase of the meeting of the open body, the rights, duties, and privileges of the regular members of the American Mining Congress and those of the duly appointed and admitted delegates are coordinate in all respects.

Local convention arrangements are in charge of the Sierra Madre Club, of Los Angeles, assisted by the principal commercial bodies of the Southwest. The Sierra Madre Club is arranging to provide very desirable entertainment for all delegates and ladies, during the time not occupied by the regular program of the congress

AMERICAN SOCIETY OF ENGINEER DRAFTSMEN

On June 18 the first steps were taken to form a permanent organization to be known as the American Society of Engineer Draftsmen, embracing every branch of the profession.

The object of the society is the advancement of engineering knowledge and practice and the maintenance of a high professional standing among its members. Inasmuch as draftsmen are very closely related to and represent every branch of engineering, an organization of this character will exert a powerful influence on the future of this field.

Every effort will be made to increase the efficiency of members, by keeping them informed on current topics in relation to mechanical, electrical, civil, architectural, automobile, and aeronautical engineering.

The membership of the society will consist of honorary members, members, and juniors. Stringent conditions will be in force governing the qualifications for admission to each grade, thereby establishing a standard, which has hitherto been lacking in this profession.

The officers chosen for the first term are, President, E. Farrington Chandler; Vice-President, Wm. B. Harsel; and Secretary-Treasurer, Henry L. Sloan; each chosen for his special adaptability to the duties of their respective offices.

Those desirous of joining this society should address Henry L. Sloan, Secretary, 116 Nassau Street, New York City.

LOS PILARES MINE

*Written for Mines and Minerals, by Edward M. Robb, Jr.**

Form of Ore Body Delayed Filling Method of Stoping—Ore Transportation

Location.—The Los Pilares Mine of the Moctezuma Copper Co., is situated in the state of Sonora, Mex., about 75 miles south of Douglas, Ariz. The town of Nacozari, Sonora, shown in Fig. 1 is connected with Douglas, the port of entry for this section of the country, by the Nacozari railroad.

Between the mill at Nacozari and the mine at Povenir, a 5-mile, narrow-gauge railroad, having a maximum 4-per-cent. grade and 40 degrees curve, has been constructed to transport the ore. This mine road ends at the portal of the main adit of the mine, which is driven through a fine-grained andesite for a distance of 2,900 feet, where it cuts the northern end of the ore formation

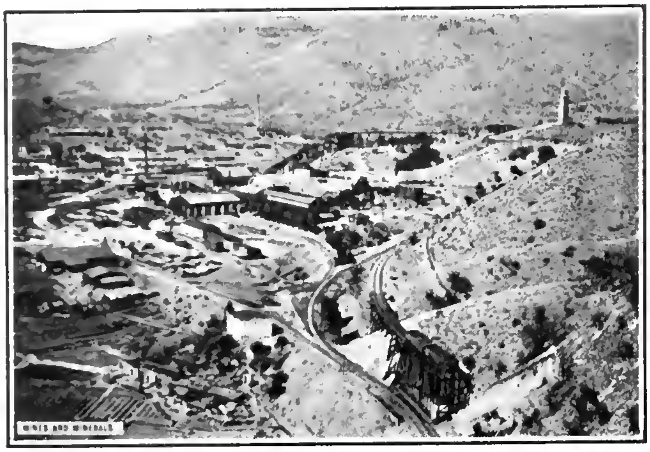


Fig. 1. Nacozari, Sonora, Mex.

and also one of the main working shafts, known as the Y shaft, shown in Fig. 2. It continues on through the ore-bearing formation until a distance of 5,000 feet is reached, where it connects with the main or Pilares shaft, at a depth of 570 feet below the collar and at the southern extremity of the ore body.

At Povenir there is an immense pear-shaped "horse" of rock 2,000 feet long and having a maximum width of 800 feet. The horse *a* as shown in Fig. 4 is surrounded by ore deposits *b* which vary in thickness from 0 to 200 feet. The interior of the horse has been shattered and at the core there are mineralized areas *c*, one of which is 300 ft. × 300 ft. There are seemingly no lateral connections between the ore deposits surrounding the horse and the ore deposit in its interior, each having a separate mineralized zone.

The country rock at, and in the immediate vicinity of, Pilares, is andesite overlain by a rhyolitic capping whose thickness varies from 200 to 600 feet. This is also true of the horse, which is rhyolite near the surface and adesite with depth, but within the pear-shaped area constituting the mineralized zone, the rock is brecciated, while the surrounding country rock is not.

The horse is capped by a brecciated, rhyolitic, iron-stained gossan, varying in thickness from 20 to 75 feet and carrying little or no copper. Occasional deposits of carbonate or sulphate of copper are encountered close to the surface and directly over the workable deposits of ore, but as they are generally of small dimensions and the copper minerals are not concentrated in them they are of very little consequence in the output. The first work on the deposit by its original locators was done in tunneling under some very prominent copper-stained rock pillars that are situated at the top of the hill and in the center of the mineralized zone, Fig. 3. These pillars are of the same rock that is characteristic of the upper workings of the mine, that is, brecciated rhyolite, but this instead of containing sulphides of copper carries micaceous hematite. From these pillars the mine first received its name "Los Pilares."

Below the gossan is the enriched mineral zone, also of variable thickness, but averaging about 100 feet. In this zone, the copper mineral changes from pseudomorphic chalcocite

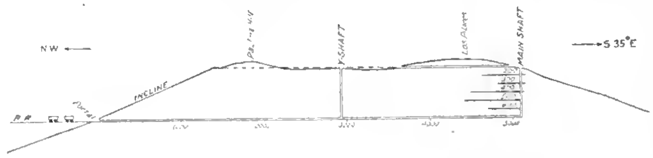


Fig. 2. Profile of Los Pilares Ridge

after pyrite, to pyrite and chalcopyrite with a slight coating of chalcocite. It is seldom, however, that a complete replacement of pyrite or chalcopyrite by chalcocite has taken place. A small quantity of bornite is also found within this zone.

Below the enriched zone are found the primary sulphides consisting entirely of chalcopyrite and pyrite. Throughout the entire ore body, from the surface to the lowest workings, the character of the deposit is the same; a shattered rock with the ores existing as the cementing material. Both the brecciated rhyolite near the surface, and the brecciated andesite below it contain copper minerals.

The peculiar shape of the ore body has been the subject of much speculation, but so far the only geologist of note to publish his conclusions has been Mr. Samuel F. Emmons†.

There is a dike x, y, z, Fig. 4, approximately bounding the southeastern portion of the ore body, but more and more approximately traveling the center of the ore deposit as it is followed to the northwest. This dike of disintegrated diabase. locally known as the *caliche*, has a width varying in different places from a knife edge to about 30 feet.

Near the surface this caliche has a slight dip to the eastward, making it a hanging wall of the ore body, but at the 300-foot level it changes its direction and dips to the west with increasing flatness from about 70 degrees on the 300-foot level to about 50 degrees on the 600-foot level. Beginning with the intersection of the dike at x and following it northward to y the ore is found wholly inside or to the west. For this reason the earlier mine workings were driven to parallel the dike believing

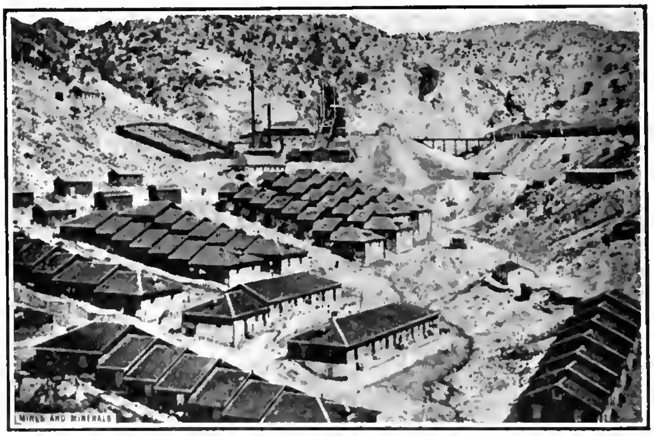


Fig. 3. Pilares Camp

that it was the ore indicator, and no ore would be found to the east of it. Later developments disproved this theory as in the northern half on the eastern side the course, the dike is through the center of the ore. In the northern portion of the deposit, large spurs and splits from the main dike, varying in width

*Mining Engineer Moctezuma Copper Co., Nacozari, Sonora, Mexico.

†See "Los Pillares Mine," Economic Geology, Vol., July-August, 1906, page 629.

from 1 to 25 feet and in length from small to great distances are found running into the eastern country rock. This dike has caused great difficulty in the mining operations, owing to its vagaries of direction and its tendency to slough and cave from above.

From the foregoing it will be understood that the ore body has a definite and fairly regular exterior boundary. The richest ore is apt to lie very close to this boundary, the grade of the ore getting poorer toward the center of the pear-shaped horse until the limit of commercial ore is reached. In what follows, the term "width" refers to the distance from the exterior boundary along a line at right angles to the same to the point at which the commercial ore ceases. This width of ore varies in different parts of the mine, and on different levels, from a few feet to more than 200 feet.

Mine Workings. Two main working shafts have been sunk in country rock about 50 feet outside the deposit from which it has been developed. These are the Pilares shaft *d*, shown by Fig. 4 at the southern extremity of the deposit and the Y or Esperanza shaft *e* at the northern extremity. The Pilares shaft at a depth of 575 feet connects with the Porvenir adit. It has three compartments each 4 ft. × 4½ ft. in the clear. Cages in two of the compartments are operated by electricity while in the third is an emergency cage, steam operated, and also water and air pipes. The Y shaft likewise has three compartments each being 4½ ft. × 5 ft. Two compartments contain electrically operated cages, while the third has been reserved for a permanent ladderway, water pipes, and air pipes. At the depth of 590 feet below the collar of this shaft, it connects with the main adit level but has been sunk 300 feet below this level. At the tenth level, 300 feet below the main adit, another main traction level will be driven, while the shaft will be sunk 60 feet further to allow for ore pockets, skipway, etc. From these pockets an automatically dumped 6-ton skip will hoist to the fifth level where it will dump into a chute to one of the ore bins on the main adit level 200 feet below.

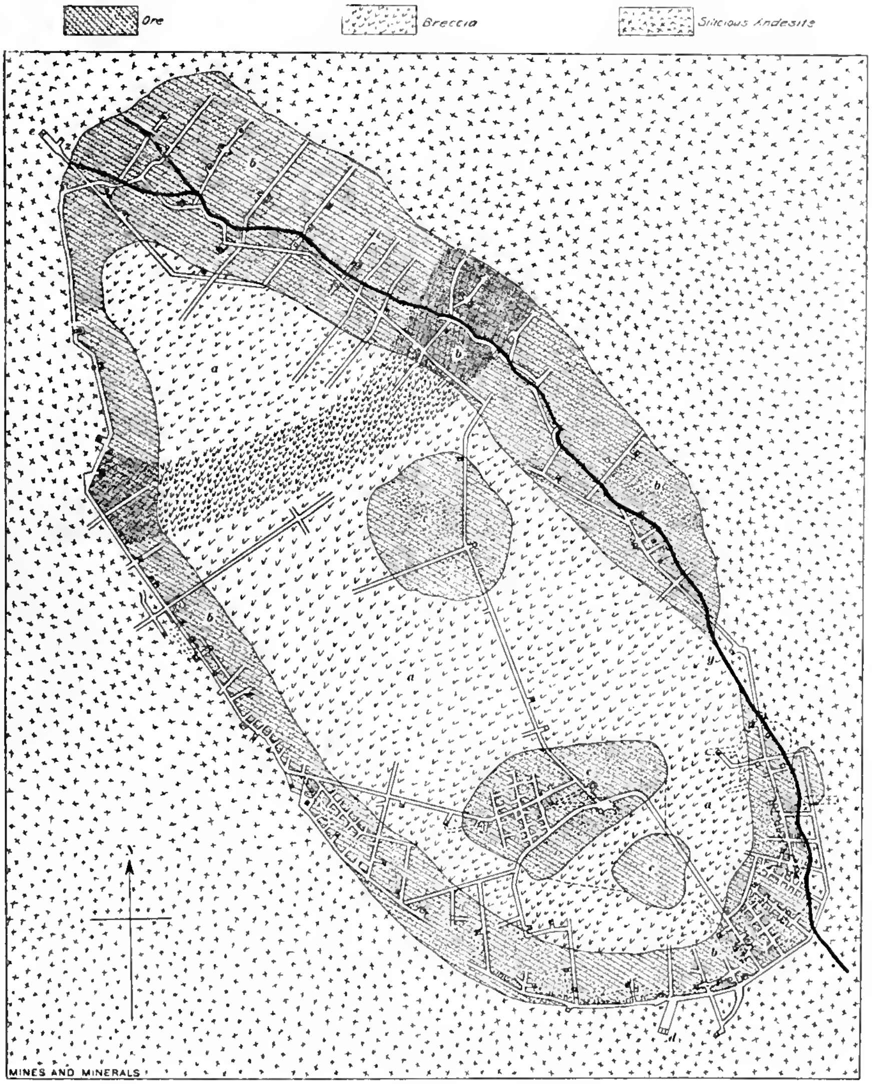


FIG. 4. ORE BODY, LOS PILARES MINE

Levels have been cut approximately every 100 feet from the surface down, with the exception of the adit level, the intervening distance between it and the level above being 70 feet on the Pilares shaft side, and 78 feet on the Y shaft side. Two-, three-, and four-track stations have been cut on all levels, leading from the shaft to the edge of the ore body. From these points, drifting to the east and west along the contact in the ore, as well as cross-cutting through the ore, has been done as indicated by Fig. 4.

The Pilares shaft, being the first sunk, and its stations first cut, it was necessary at the Y shaft to start all levels corresponding to those of the Pilares side and have connections driven from both sides.

In drifting to the west (from the Pilares side) there was found a well-defined regular wall extending northwest for about 500 feet and dipping to the southwest about 10 degrees. At this point, breaks and irregularities in the wall rock were encountered, making it difficult to follow, but at no time causing any uncertainty as to the confines of the ore. From this point, the main drifts were purposely run on lines placed inside the ore a short distance from the wall with cross-cuts driven at intervals (either in the center of a pillar or the center of a stope) to determine the position of the wall. If, by these cross-cuts, it was found that the drift was diverging too much from the wall, it would be turned more toward it. If, on the other hand, the drift encountered the wall, the latter would be followed, if it was well enough defined, if not, the drift would be slightly turned away from it into the ore and the same method followed as before.

On drifting to the right, or east, from the Pilares shaft, the same well-defined wall was found for a distance of 300 feet. At this point the previously mentioned caliche dike was encountered. In the first development of the mine, this dike was followed northwest driving the main drift in the ore and thereby leaving the dike rock as the foot or outside wall. It was very soon demonstrated, however, that this was impracticable and costly, owing to the sloughing tendency of the dike. The sloughing necessitated the timbering of all permanent drifts, and as the dike rock proved to be "heavy ground" the cost of timbering was excessive at this point. After this experience all permanent drifts on this side of the mine were driven away from the dike either in the country rock or else in the breccia below commercial grade to the west of the ore. In some cases

prospecting drifts along the dike, determined the position of the ore and permanent drifts followed later. Sometimes the method used on the opposite side of the mine was resorted to, that is, cross-cuts were driven at certain intervals, to determine the position of the dike, and then the main drift was driven ahead according to results obtained.

Mining Methods.—The width and length of the ore body, the kind of ground found in the deposit, the excessive cost of timber at the mine, and several smaller items, were the considerations which determined the method of ore extraction.

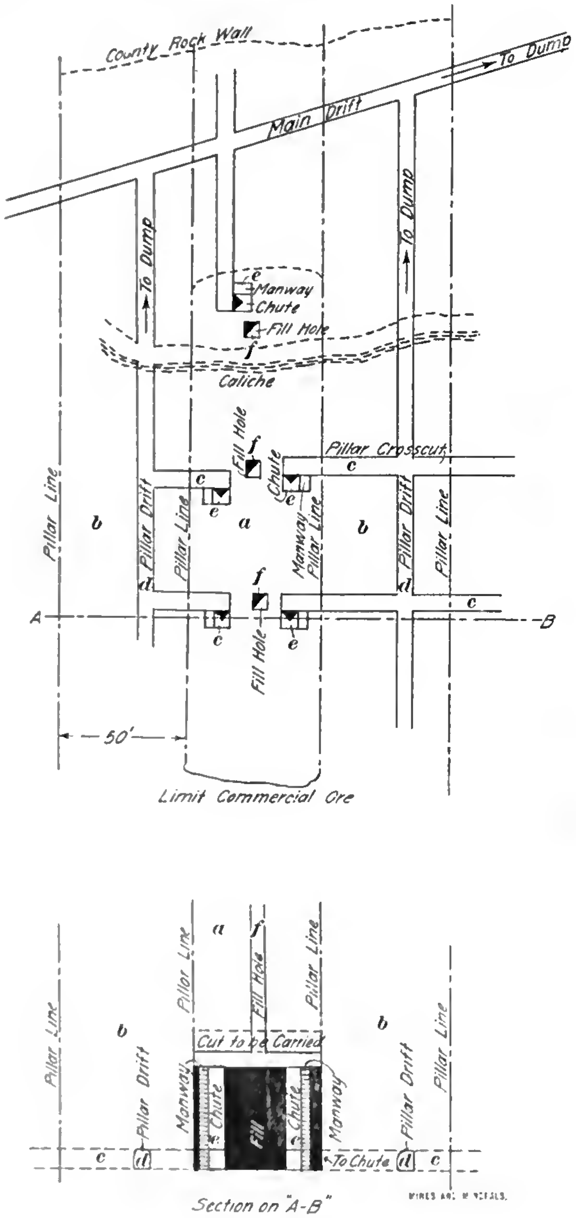


Fig. 5

The pillar-and-stope method is in use throughout the workings The whole deposit, or rather the commercial ore area, is divided up into a series of stopes and pillars, the widths of which vary according to the width of the ore and the character of the ground to be worked.

Pillars and Pillar Lines.—The pillars are bounded by imaginary vertical planes extending from the surface to the bottom of the workable ore. Separate maps like Fig. 4 are kept up to date for each level. On each of these maps, the pillars are accurately plotted, thereby showing the location of every stope and pillar, its dimensions, and also the courses of the several pillar lines.

When a stope is to be "sill-floored," the engineer will set pillar plugs on each side of the stope that calls for a pillar. The position of these plugs will be calculated with reference to their distances from their corresponding pillar lines. The distance from the pillar plugs to the pillar will then be given to the stope boss, and it is his duty to see that the pillar line in question is carried forward and the plugs cared for. For a height of two or three slices, these plugs will be changed by the eye by the stope boss; they will then be checked by the engineer and carried on as before. These pillars vary in width from 25 to 60 feet and are placed approximately at right angles to the country wall. Thus, each stope is bounded on two sides by pillars, while the wall rock on one end and the end of the commercial ore on the other constitute the respective third and fourth sides. When the ore zone is narrow, say from 40 to 100 feet, the above statement applies and the stope will vary from 100 to 150 feet in length along the length of the ore zone, and pillars vary from 25 to 60 feet in length also, measured along the length of the zone, depending on the nature of the ground. When the commercial ore zone is much wider, say 250 feet, the length* of the stope is either lessened and that of the pillar increased, making a stope 50 ft. × 250 ft. and the pillar the same size, or, in case of excessive width of ore, a third pillar is introduced running at right angles to the other two pillars and practically making two stopes between one set of pillars, whereas

Fig. 6. Leveling Waste Filling

if the ore had not been so wide, only one stope would have been excavated.

Sill Flooring.—The stopes may be "sill floored" by two methods. By the first method the entire stope area is cut out on the level floor, while by the second method a floor arch 15 feet thick is left above the floor level, and from the top of this arch the full stope area is carried up. The first method necessitates either a permanent drift in the pillar with cross-cuts running to the stopes and ending in shovelways or chutes, or necessitates the driving of permanent drifts outside the ore with cross-cuts run to chutes or shovelways in the stope; or both classes of drifts may be used for the same stope. The first method of sill flooring is generally used in bodies of high grade ore, or in weak ground. In the second method of sill flooring a permanent drift must be maintained through the stope and for that reason the drift is protected by the floor arch. This is the method used in the case of wide and long stopes, where the main development drift has been driven along the country wall and must be maintained in order to extract the ore. With this second method of sill flooring when the ore is 75 or 100 feet wide, an auxiliary permanent level is driven from the main level through the length of the stope and protected by the floor arch.

Three different methods of stoping are in vogue in this

* In this article the term "length" whether mentioned in connection with stopes or pillars refer to the dimension parallel with the country walls. This is done for the sake of clarity.

mine. First, square setting; second, slicing and continuous filling; and third, slicing and delayed filling.

Square-set timbering and stoping is well known and also but little used here, so it will not be taken up in detail. It is adopted in soft ground that is liable to cave. After extracting the ore and timbering with square sets, permanent levels may be either driven through the pillar or maintained through the center of the stope by lagging over the sets through the center line of the stope. The stope is then filled with waste up to the top floor of the square sets. Chutes and manways are carried up by lining a given timber set all the way up with 3″×12″ plank and dividing it into chute and manway compartments.

Slicing and continuous filling may be used after a stope has been started by either of the two methods of sill flooring. As an illustration of this method with the first sill-flooring system assume a stope *a*, Fig. 5, 50 feet along the ore body and 200 feet wide. Corresponding pillars *b* will also be 50 feet in length. At 50-foot intervals, cross-cuts *c* will be driven into the stope from the pillar drifts *d* on both sides of the stope in question. Working from the ends of these pillar drifts the entire

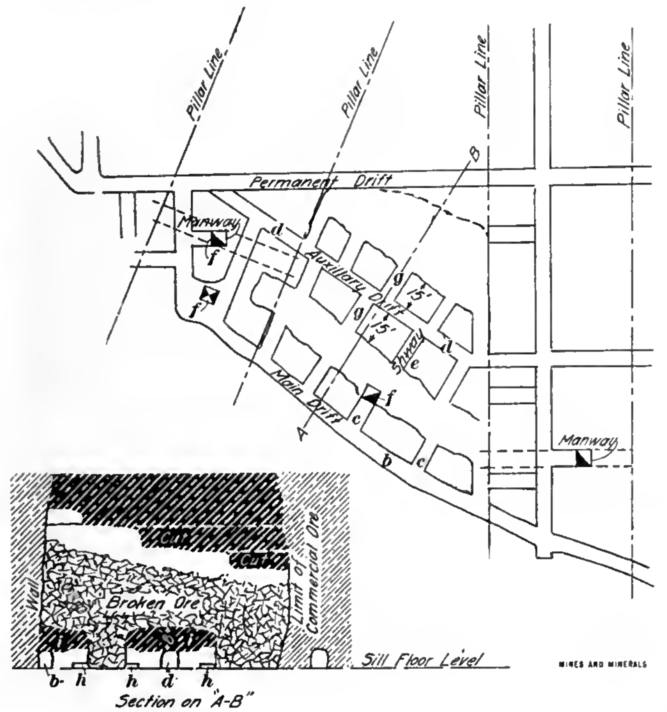


Fig. 7

area of ore within the pillar lines is removed by blasting to a height of 15 feet. This broken ore is trammed over tracks laid through the several cross-cuts from the pillar drifts and by extending temporary tracks from them into the center of the stope. The broken ore is shoveled into cars and taken out via these tracks. After cleaning out all of the ore, the temporary tracks will be removed. Fifteen feet from the pillar *b* inside the stope and adjacent to the extension of the pillar, cross-cuts *c* are built up the chute and manways *e*. This 15 feet of cross-cut is timbered after extracting the ore. The stope will then be filled with waste from the mill holes *f* to within a distance of 5 feet from the roof, the work of leveling off the waste from the different mill holes being done by shoveling and wheelbarrow work as shown in Fig. 6.

These mill holes *f* it will be noted in Fig. 5, are adjacent to the ends of the pillar cross-cuts. It is, therefore, obvious that they are raised to the level above simultaneously with blasting out the first slice. They thus serve not only for dumping down waste for filling, but to ventilate the stope.

Chutes and manways are kept built up sufficiently above the waste packing so as to prevent the filling from running into them. The chutes are generally cribbed with 6″×6″ timbers with a ⅝-inch notch on each end. This leaves a 4¾-inch opening in the crib between each corresponding pair of timbers. Large rock from the packing is built up around the outside of the chute to keep it in place, and the inside is lined with 3″×12″ plank. The manway, usually about 2½ ft. × 6 ft., is carried up along one side of the chute and is built out of 2″×12″ plank. A manway is sometimes deemed unnecessary for a chute, in which case the latter alone is carried through the fill. In the manway, a 2″ × 3″ diameter pipe is placed with its top level with the top of the manway and its bottom about 7 or 8 feet above the floor. This serves the purpose of letting drill steel down without cutting the manway lining and breaking ladders which would result from throwing down the steel. After the filling has been leveled off to its proper height, another slice varying from 6 to 12 feet in height at the breast is blasted from the stope. This slice is started by driving blind raises to the proper height and enlarging these till they intersect, this method enabling the drilling of flat water holes. It is often the

Fig. 8. Unfilled Stope

case that the same slice will be started in three or four different parts of the stope from blind raises. The ore broken in this method of slicing is put into the nearest chutes by shoveling and wheelbarrow tramming. After the slice has been finished, the stope is cleaned of its ore and waste filling is run in again and the same procedure followed as before.

When the ore shoot is narrow, say 30 to 50 feet in width, for quite a length along the country wall, and the ore is of comparatively low grade, the same method of extraction is used, only the main drift is usually protected by a floor arch. This so-called floor arch protecting the drift is shown at *i*, Fig. 7. The main drift is generally run along the country wall or close to it as at *b*. Cross-cuts *c* are driven every 30 or 40 feet at right angles to the main drift in the ore, the two opposite terminal ones being on the pillar lines. If the drift to be maintained is in the ore, then these cross-cuts may be driven at the stated intervals on either side of the drift, generally alternating from the left to the right side. Offset from each cross-cut and set out 8 feet from the center of the main-drift track, a 6′×6′ raise is driven to a height of 22 feet. After raising in each cross-cut the stated height, intermediate drifts

and cross-cuts 7 feet high are run from each raise making the floor of each drift 15 feet above the floor level. These intermediate drifts are connected together and the first floor of the stope is then excavated by enlarging them by blasting both ways till they intersect. All the ore between pillar lines is extracted to the country rock wall in one direction and to the commercial ore limit in the other. Should the ore prove to extend for 20 feet or more in width from the center of the main drift, an 8-foot pillar is left between the main drift and the ore, and an auxiliary drift *d* from the cross-cuts is driven parallel to the main drift. From this auxiliary drift the ore is sliced back, the broken material being trammed out of the various cross-cuts first driven from the main haulage drift. This work leaves a solid arch of rock 8 feet wide on each side and 8 feet thick over the main haulage drift to protect it. When this method is followed chutes are cribbed up from convenient points about 50 feet apart. For such small stopes two manways *f* usually suffice, which are placed adjacent to the two chutes at opposite ends of the stope.

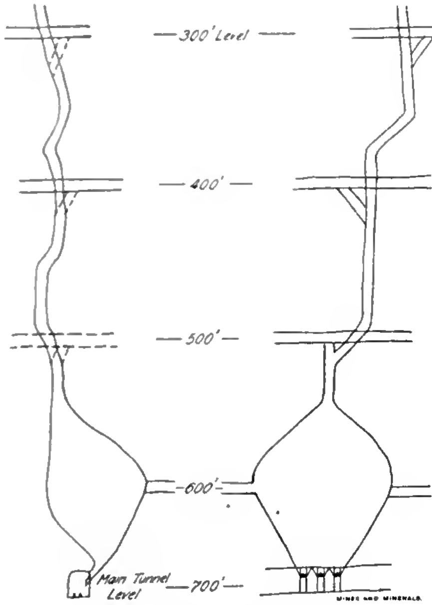


Fig. 9. Sections of Main Ore Bin

After the stope is completed and cleaned of ore, waste is let into the excavation and operations proceed as in the previous case. If the width of the ore is too narrow to admit leaving the ore pillars to protect the drift, it is, of course, obvious that the first system must be followed, after which the drift is timbered and waste filled in over it.

Slicing and Delayed Filling.—Where this method is employed the sill flooring of a stope is practically the same as in the continuous filling method. The ore extracted by this method of stoping has a better grade than that mined by the continuous filling method, and for that reason no chance is taken of mixing it with the waste filling. The ground is also much firmer, allowing a large stoping area without danger of caving and thus losing the stope. At intervals of from 12 to 20 feet cross-cuts from the main drift (which is generally on the wall) are driven into the ore for a distance of 20 feet from the center of the main drift. If the ore body is not wide, these cross-cuts will suffice to draw the broken ore of the stope, as there will not be too much space between the far side of the stope and the end of such a cross-cut which, because of its purpose, is termed a shovelway. If the ore body is wide, as in Fig. 7, an auxiliary drift *d* is driven through the ore at approximately right angles to the pillars and located about two-thirds the distance between the country wall and the limit of the ore. Cross-cuts *g* are then driven at intervals of from 12 to 20 feet from this auxiliary drift to both the left and the right, 15 feet long, and after leaving a 15-foot pillar between the side of the drift and the end of the cross-cuts, the remaining area of the stope is "silled" on the level floor. The pillar thus left forms the base of the floor arch over the drift, and is pierced by the cross-cuts which serve as shovelways. Turn sheets are then placed in the main drift in front of each cross-cut and from 6 feet to 8 feet of track laid in each cross-cut. A platform *h* raised 2 feet above the rails is then placed in the cross-cut at the far end of the track and a flat iron sheet placed on top of it, at the same time raising the roof above the platform about 1 foot. This track arrangement allows the car to be turned into the shovelway, leaving the main track open. The broken ore runs down on to the iron sheet over the platform from which it is shoveled, thus giving the car man an easier task in filling his car.

Fifteen feet above the level floor, the stope is cut out back over each drift, leaving it protected by the pillars and floor arch *i* as seen in the section shown in Fig. 7, but thus acquiring the whole area between the pillars overhead for stoping. From the top of the arch up, the stope is worked by two and three slices being carried forward at the same time. The broken ore accumulates in the stope and only enough is drawn from below through the shovelways to allow the miners to drill by standing on ore. Manways *f* are carried up through the center of the pillars limiting the stope on each side. From one manway 20 feet above the level floor an intermediate cross-cut is driven to the stope and at each succeeding 20 feet similar cross-cuts are driven. From the pillar manway on the other side, the first intermediate cross-cut to the stope is driven 30 feet above the level floor, and others at intervals of 20 feet above it. These manways and their connecting cross-cuts give an inlet and outlet to the stope for every 10 feet. By this method, a stope may be carried up 100 feet, or even 200 or 300 feet, before being drawn and filled. Again a stope may be worked from two or three levels at the same time, each level being driven up till only a thickness of 12 to 15 feet of solid ground separates two stopes. At this point the uppermost stope is drawn through its shovelways, after which the floor arches, etc., protecting the haulage drift are shot down. The solid ground separating the two stopes, one over the other, is then drilled with a large number of holes, say 50 to 60, from the top of the broken ore, the holes heavily loaded and shot down


Fig. 10. Dumping Station, Main Ore Chutes

thus making the two stopes one. By this method all the ore left for floor arches, etc., is eventually recovered except that on the lowest level worked. [From the above description of the slicing and delayed filling system, it will be evident that it differs from the well-known "filled stope" or "shrinkage stoping methods" mainly in its method of protecting a drift in the stope and its elimination of chutes by substituting shovelways for them.—Editor.]

These great stopes may, or may not be filled with waste rock soon after drawing off the ore. When the mine was first

opened they were sometimes left standing empty for months without accident. In recent years, however, the tendency is to fill as soon as possible after drawing the stopes. As illustrative of the standing qualities of the rock, no better example can be cited than that of the old No. 1 stope worked out during the first years of mining. This was located near the Pilares shaft, was 100 ft. × 100 ft. in floor plan and was worked up from the 400-foot level clear to the oxidized capping, a vertical distance of 280 feet. Although the capping was here only 25 to 30 feet thick, the empty stope stood for 18 months without caving before it was filled. Fig. 8 shows the top of a great stope several hundred feet in height about to be filled. The cars of waste are dumped from the track seen in the foreground, the curved-up rails engaging the end wheels and preventing the car from dropping into the stope as it is dumped.

From the above description of the different methods of work at Los Pilares, it will, be realized that the particular method applied at a given point in the mine depends on a number of conditions. The size of the ore body, its richness and the standing qualities of the rock are the main determining factors. The maintenance of ore supply for the mill is another factor of prime importance to be considered. The continuous filling system allows the milling of the ore soon after mining, while the delayed filling system necessitates its accumulation in the stopes for some time.

Filling the Stopes.—In filling stopes mined by the slicing and delayed filling method, fill holes are run direct to the surface over the stope where they are widened out to a size of 12 ft. × 12 ft. On the surface this 12′×12′ hole is then further enlarged to a roughly funnel shape by churning holes from 12 to 30 feet in depth above the edge of the fill hole. These churn drill holes are sprung with dynamite, and shot with black powder, the rock breaking from such shots falling directly into the stope. By having the fill hole of this large size, it is seldom choked with large rock.

In the filling of stopes mined by the slicing and continuous filling system, another method is used. One main fill hole is raised to the surface from a level located 100 to 200 feet above the stopes to be filled. This main fill hole then serves for from three to five stopes or more. From the stopes to be filled, up to the level on which this main fill chute is located, from two to four fill holes are driven for each stope. These fill holes are placed either beneath the center of the track or inclined over to the center from one side. Bearer timbers are laid over these holes and the track run over the bearers. A 5-horsepower electric motor, with a train of 10 cars, of 1 ton capacity pulls the fill rock from the main fill chute to the particular stope and fill hole where waste is called for. In this main fill hole at a distance of 75 to 125 feet below surface (distance varying according to the point of approach or entrance) a grizzly station is cut and a grizzly put in over the fill hole. This grizzly is variable in size, ranging from 8 ft. × 20 ft. to 10 ft. × 10 ft. The mouth of the fill hole is chambered out to a convenient size on a side away from the hole leading to surface. Bearers of 12″×12″ timber spaced about 3 feet apart are placed across this hole. On them are bolted 40-pound rails so as to form openings of 15 inches square. All rock falling on the grizzly must be broken fine enough to pass through these openings. When boulders come down larger than 15 in. × 15 in., they catch on the grizzly and are broken up. This prevents the hole from "hanging up" between this grizzly point and the chute below and also eliminates any trouble at the chute in the loading of the cars.

Ore Transportation.—On the main tunnel level, and located so as to reach all stopes advantageously, six ore pockets or bins similar to the one shown in Fig. 9 have been cut out of the rock within the ore zone, each having a capacity of from 1,000 to 10,000 tons. Each one of these bins is provided with from one to three sets of two chutes each, one set of chutes filling a 30-ton Ingoldsby bottom-dump ore car. Two 10-ton General Electric traction motors running in tandem pull a train of from six to eight of these cars to the main tunnel mouth, Povenir, which is the terminal of the mine railroad. From here a 60-ton Baldwin locomotive takes a train of 14 cars to the concentrator at Nacozari. On each of the succeeding levels above the main tunnel level as far up as the 200-foot level, continuous connections have been made to each ore bin as shown in Fig. 9. With dump stations provided for each bin on each mine level, similar to that shown in Fig. 10, the ore from each working finds its way into the nearest dump. This does away with the hoisting of the ore, a costly item, necessary at most mines.

Compensating Mexican Labor.—It might be said that all work done underground is done by contract. All development work such as sinking, raising, drifting, and cross-cutting, is contracted to the native Mexicans at so much per foot driven; the company furnishing steel, powder, fuse, caps, etc., the contractor only having to keep his working up to regulation size and in some cases running his dirt to the chute. In case of encountering waste in a working, it will be dumped into a fill stope. Prices of the work per foot vary with the kind of rock and also depend on whether the drilling is preformed by machine or hand. In the stopes both machine and hand drills are used, the miner being paid so much per foot drilled with a machine, or so much per foot with hand steel. In the stopes the supervision of the holes is not limited to the number drilled but they must be drilled as "pointed" by the stope boss to the stipulated depth. Car men receive so much per car trammed, the price varying with distance traveled and whether the ore is shoveled from sheet iron, or a rough bottom, or drawn from a chute. In the stopes, shovelers dump wheelbarrows into chutes and are paid by the number of cars drawn from the chutes, which are counted up at the end of the day. As a check, the height of the ore in the chute is taken before starting to work and after finishing; the number of inches in the chute to the car being known. In filling stopes men are given a task of so many wheelbarrow loads per day for a certain wage. All wheelbarrow loads over or under this number are figured and paid for in proportion of the task to the wage. About 1,200 men are usually on the monthly pay roll with an average daily working force of 800 men. From 1,500 to 2,000 tons of ore is sent to the mill daily.

TARIFF ON SAWED TALC

Talc sawed into cubes for use in making gas burners and insulators, the sawing being not merely to remove foreign matter and to put the material in shape for transportation, but to put it into certain desired dimensions, has been advanced in value and condition, and is therefore excluded from paragraphs 519 and 614, tariff act of 1897, relating to crude chalk and to minerals not advanced in value or condition. Talc in the form of cubes, which is used in making gas burners, is dutiable as French chalk by similitude, under paragraph 13, tariff act of 1897. The merchandise was invoiced as crude talc which the importers claimed was entitled to free entry under the following paragraphs:

519. Chalk, crude, not ground, precipitated, or otherwise manufactured.

614. Minerals, crude, or not advanced in value or condition by refining or grinding or by other process of manufacture, not specially provided for in this act.

French chalk, a magnesian silicate, is defined as a variety of talc. The article is imported in the form of cubes, it having been finished by sawing and is chiefly used in gas burners and electric insulation. The sawing of the talc prior to its importation is not merely to remove foreign matter and to put in shape for transportation, as there is a demand in the trade for "finished pieces" of certain dimensions, and the material is ordered by the importer accordingly. Such sawing of the talc advances its value and condition.

THE SUPERIOR AND BOSTON MINE

Written for Mines and Minerals, by R. L. Herrick

System of Mining and Pump Arrangement at McGaw Shaft, Globe, Arizona

Among the newer mines of the Globe, Ariz., district, whose engineering features most impress the visiting engineer, is the Superior and Boston. It is located about 4 miles to the east of Globe, a short distance beyond the little mining settlement of Copper Hill. The accompanying illustration, Fig. 1, shows the main hoisting shaft of the property, the "McGaw," at the right, with shaft and smelter of its neighbor, the Arizona Commercial, at the left. Nearly all of the surface equipment of both mines showing in the view has been installed during the past year. Up to March, 1908, the property had been worked in a somewhat desultory fashion, but never adequately prospected. At that time, the property was examined by Frank H. Probert, of Los Angeles, and, based upon his report, a vigorous campaign of development was inaugurated, Mr. Probert being retained as consulting engineer.

With a comparatively small amount of work, the upper levels of the mine were quickly made to produce a considerable tonnage of oxidized ores while development at depth proceeded, and at this writing there is every prospect that large bodies of sulphide ores will soon be developed in the enriched zone. The present production of oxides is about 80 to 100 tons daily, assaying on the average of 6 to 8 per cent. copper or better. Satisfied with the future prospect of a heavy production of sulphides, however, the beginning of the year 1909 saw the start of installing a surface equipment, and a remodeling of the underground facilities which today make the property one of the "show mines" of the Southwest.

Mining Notes.—The McGaw shaft, shown in Fig. 1, is the main hoisting shaft of the property from which a drift on the 400-foot level connects with the old prospecting shaft, the "Great Eastern." Its depth is at this writing 640 feet, with sinking still progressing. As water enters the shaft at the rate of 350 gallons per minute, considerable difficulty was experienced at first in firing the shots, owing to the occasional soaking through of the best makes of waterproof fuse. After considerable experimenting, the difficulty was eliminated as follows:

The fuses to be used for a given round were cut to the required length, their caps crimped on, and the junction of fuse and cap protected by covering with the well-known rubber protectors. The whole length of fuse and cap was then dipped in liquid asphaltum and hung up to drip and dry. When dry in a few hours time the asphaltum forms a thin elastic layer, making the fuse perfectly waterproof. There have been no failures reported of fuse protected in this way.

The present mine production is made mainly from the stopes of the 550-foot level. Here the vein dips about 58 degrees and varies in thickness from 7 to 15 feet, but averages from 9 to 10 feet. The foot-wall is hard and smooth, but on the hanging wall the ore is "frozen" and there is no defined wall. These conditions are mentioned because of their bearing upon the novel method of mining, devised by Superintendent John D. Wanviz.

Until recently the ore has been mined by the common square-set system, but in the few months since the first trial of the new system, both the timber and labor, not to mention loss of ore fines, have been so materially reduced, with a consequent reduction in costs, that the new system may be fairly pronounced a success.

System of Mining.—Reference to the accompanying Fig. 2 will make clear the following data:

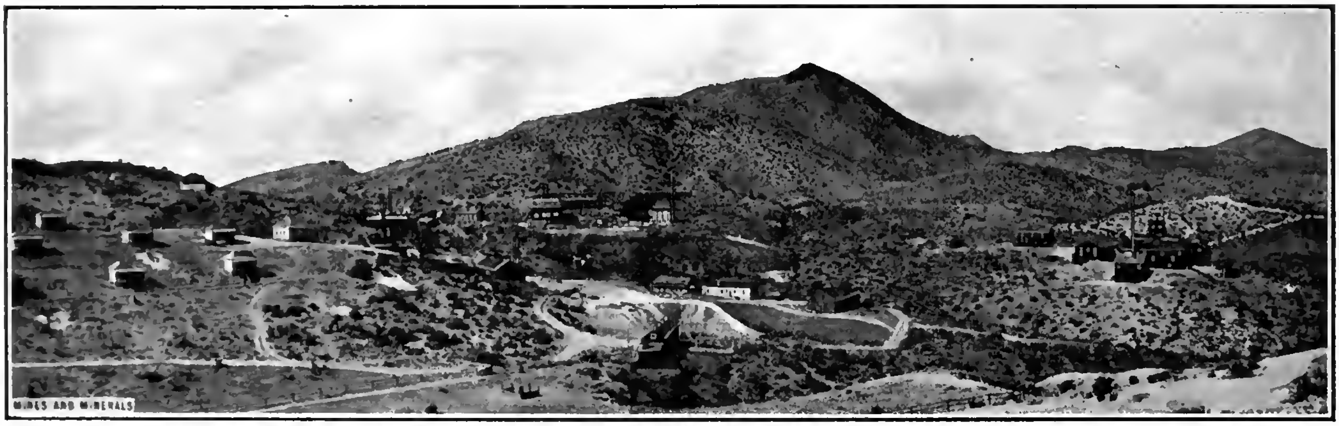


Fig. 1. Superior and Boston Mine, Globe, Arizona

Drifts 5 ft. × 5 ft. are carried 100 feet apart vertically or about 117 feet along the 58-degree dip of the vein.

Two-compartment raises are put up at 100-foot intervals to connect the drifts; the chute compartment is 4 ft. × 4 ft., and the ladderway 2½ ft. × 4 ft. These raises are on the foot-wall and are timbered only by two lines of stulls with head-boards and one set of lagging.

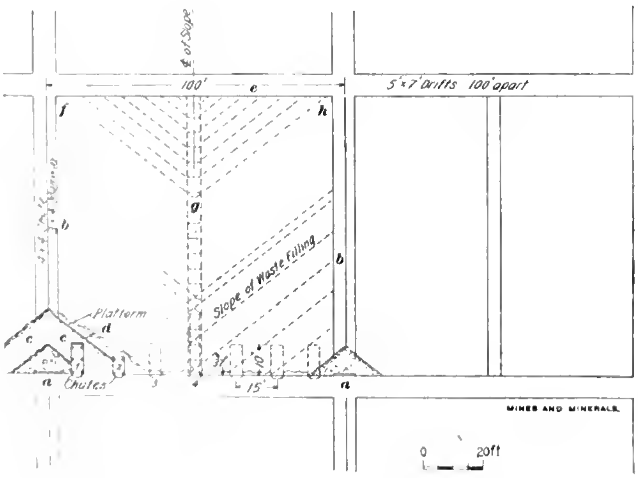


Fig. 2. Method of Stoping. Section Parallel to Dip of Vein

The division between the two compartments is formed by the first line of stulls which is plank lagged. The second line of stulls is carried on the other side of the ladderway compartment, but this is unlagged. The outer wall of the chute is thus constituted by a rock wall unstulled and unlagged and the outer wall of the ladderway is likewise constituted by a rock wall, but this is stulled. The main drifts are all timbered with drift sets spaced on 5-foot

centers and heavily lagged. Chutes 4-feet wide are spaced on 15-foot centers, the chute gates all being placed at once, but the chutes themselves are put up one by one as stoping proceeds above. The chute mouths afford one of the ways of access to a stope as it progresses, since there is always an irregular space 1 to 3 feet high between the ore back and the top of the drift lagging.

Stoping is started at the lower corners of two adjacent blocks formed by the intersection of a raise with a drift, such as *a*, Fig. 2. These two blocks are simultaneously stoped up along the raise and retreating from it horizontally as shown. At such a corner the back is drilled with stoping drills and broken down on to the drift lagging for a distance of 8 to 10 feet. The broken ore is at first discharged into mine cars standing in the drift below by pulling out the lagging and letting it run down.

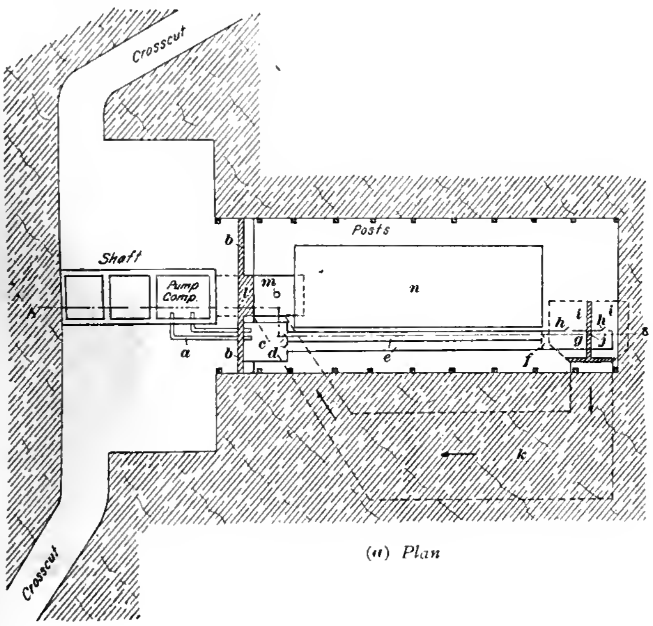
The first chute *1*, Fig. 2, is then placed. This chute is formed by carrying up two lines of stulls 4 feet apart along the strike of the vein and lagging them on the outside with plank.

Fig. 2 shows the stulls inside the plank lagging instead of outside, as in ordinary practice. The reason for this is found in the fact that when the chutes are abandoned they are not filled with waste, so that empty they have to stand the pressure of the waste filling on all sides of them.

The entire width of vein from foot to hanging wall is broken out between the rows of stulls. With the first chutes in place, the timber in the lower part of the raise *b*, is taken out and waste *c* for filling dumped down the chute from the drift above. This waste assumes a natural slope of about 37 degrees from the horizontal, and it accordingly spreads out from the foot of the raise into the stopes on each side of it at that angle, running farther and farther horizontally into the stopes as they are advanced and the pile of waste increases in height. In the first stage of the work, however, advancing toward chute No. 1, the waste is dumped down till the foot of its slope almost reaches the chute. Dumping waste then ceases for the time being. With the sloping floor of the waste now brought within 5 to 7 feet of the ore back, a sloping wooden floor *d* is laid down on top of the waste extending from the top of the chute to the manway of the raise. Its purpose is to receive the ore broken from the back and discharge it into the chute, thus serving the double purpose of separating the ore from the waste and eliminating all shoveling and tramming in the slope. It will be evident that since the waste will run at an angle of 37 degrees, the ore will certainly run on the wooden planks which are laid over the waste on the same angle.

The width of the plank floor *d* is everywhere made that of the vein from wall to wall, and its length, of course, increases up to a maximum of about 60 feet in extending from the center chute of the block up to the raises on either side of it. The floor consists of 2-inch plank in 15-foot lengths nailed together by cross-pieces to form sections about 2 feet wide. In laying down these sections over the waste, the ends are made to form a butt joint; i. e., the end of an upper section butts directly against the end of a lower section, to insure a clean run-off of the ore. With the floor laid down, the stope drills are put at work to break down the new back, stulls and planks being placed where necessary to afford a good secure footing next to the back. When drilling operations are completed, the stulls are pulled, thus recovering them for further use. The chute at the foot of the sloping floor is always kept nearly full of ore so that wear upon it may be kept at a minimum. After drilling and shooting the back, the ore is at once drawn down to the top of the chute, thus clearing the floor. A number of stulls are next put in as close to the back as possible and the sections of the platform are then raised and one of each placed see-saw fashion over the stulls so that the weight of a larger end of a given section causes the smaller end to press up against the ore back and thus be firmly held. With the platform sections thus disposed conveniently at hand for their next period of use, the top of the chute is now built up and more waste dumped down the raise till the foot of its slope has nearly reached the level of the chute top. The laying of the floor and the mining of another diagonal slice now proceeds as before.

An inspection of Fig. 2 will now make evident the fact that a given chute can be used at the foot of the sloping platform only till the angle of slope carries down the ore in line with the center of its top. After that the top is lagged over, the chute emptied of its ore, but not filled with waste, and it is abandoned. Fig. 2 shows chute No. 1 just abandoned and buried in waste with the use of chute No. 2 just begun. The illustration also makes it evident that of the seven chutes raised in a 100-foot block, all but the one in the center are limited to a height of about 10 feet, as at that height the angle of waste slope carries the flow past to the bottom of the next adjacent chute, which is likewise built up. By working from the two opposite lower corners of the block simultaneously, and thus advancing the diagonal slices toward the center line of the block, they finally intersect at the bottom of the centrally placed chute No. 4. From this point on, chute No. 4 receives all the



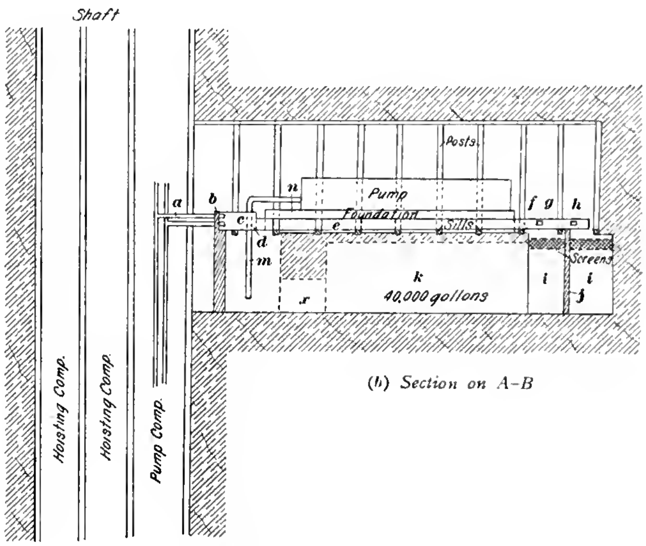


FIG. 3. ARRANGEMENTS OF SUMP AT SUPERIOR AND BOSTON MINE

ore of the block. It is built up from time to time just as the other chutes were, with the exception that it is more heavily and completely lagged by placing 2-inch lagging outside the stulls and 4-inch plank lagging inside the stulls which bear the wear of the chutes. Its height, as shown in dotted lines, is limited only by the floor of the drift *e* above. It will likewise be evident from the illustration that when the top of the waste filling has reached the approximate position of the dotted line *f*, *g*, *h*, the drift *e* must be abandoned for through traffic, or its floor supported on stulls. From this point on, the raises cannot be utilized for throwing down filling, so that waste is dumped from points along the drift itself advancing toward the center line of the block till the central chute reaches the drift and the block is entirely mined and filled.

During the mining of the lower portion of the block, access is had to the stope from the manways of the raises or from the chute mouths in the lower drift roof. After the diagonal slices have connected at the central chute, No. 4, however, access is had only from the drift *e* above by descending through the raise manways into the stope. No objection to this limitation of accessibility has yet been found in several months of work.

this is the case it is doubtful if the cost of timber in the Superior and Boston mining method much exceeds, if at all, that in the "shrinkage method."

Compared with the recently abandoned method of square-set mining, the total costs thus far are hardly more than one-third that of the square-set method.

Pumping Arrangements.—As the lower levels of the mine are developed, the quantity of water pumped is expected to increase above the 350 gallons per minute now handled. The pumping plant recently installed, therefore, was designed for future as well as present needs. The main pump station is on the sixth, or 550-foot level. The shaft at this writing is 90 feet below, and from it two sinking pumps lift water to the main-pump sump through 6-inch diameter pipes. The sinking pumps are of the Prescott duplex type, size 14 in. × 8 in. × 12 in.

In the design of the main station sump, several novel features were embodied which are detailed below. Reference to the accompanying illustration, Fig. 3, will make the situation clear.

In excavating this station sump, instead of the procedure commonly followed; i. e., sinking a short winze and blasting out

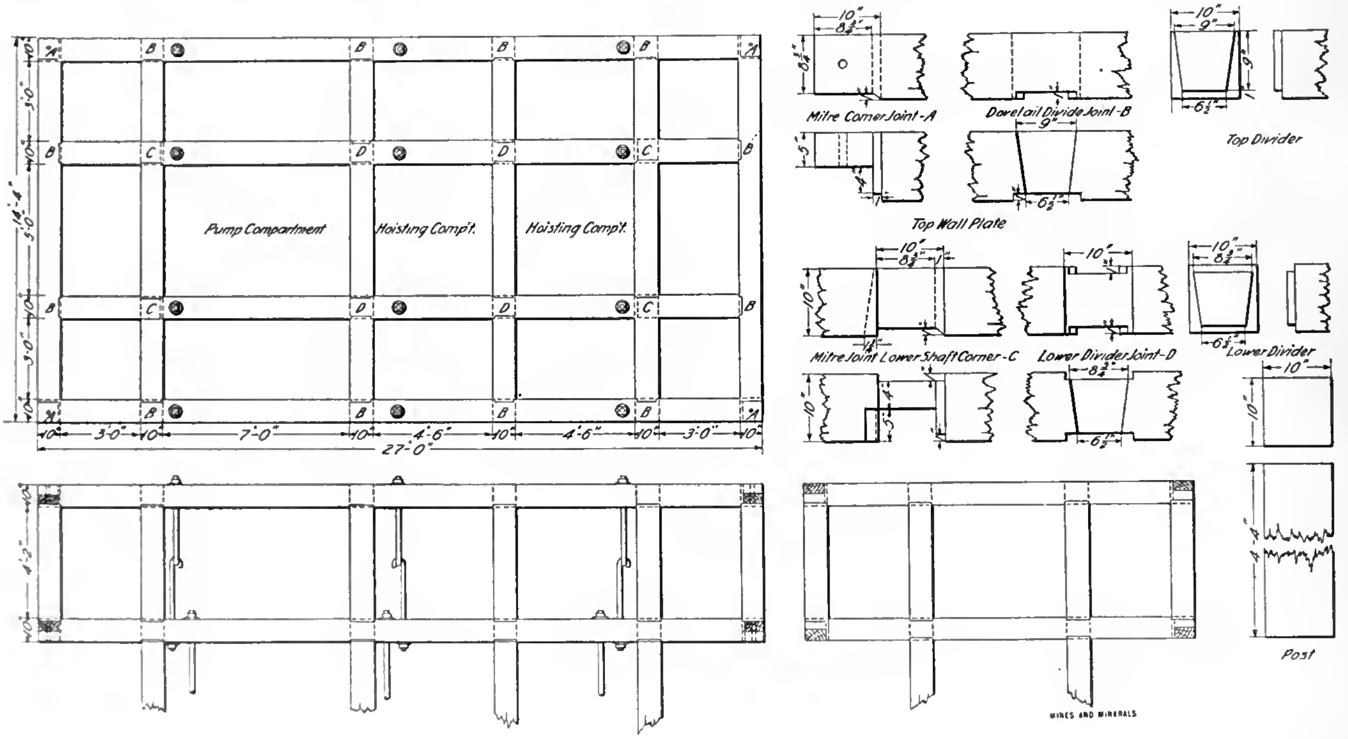


Fig. 4. Collar Set

If found desirable, however, the floor on one side or the other could always be kept down and over it the men could pass in and out to avoid danger of starting the loose waste.

From what has been previously said concerning the loose nature of the hanging wall, it will be evident that working the mine by a "filled stope" method, perhaps better known as "shrinkage stoping," would be inadvisable, because caves from the hanging wall would seriously dilute the ore with waste.

The system adopted, as described above, therefore, seems the next best method of reducing the use of timber to a minimum, eliminating tramming and shoveling in stopes, and recovering all the fine ore. The system has the advantage over "shrinkage stoping" of making all the ore broken available at once instead of only about 25 to 40 per cent. until such time as the stope is finished. Another advantage over "shrinkage stoping" is that the drift timber sets and chutes once placed require no reinforcement during the drawing of ore as is frequently the case in drawing a stope completed by the shrinkage method. Where

a sump from its bottom, the work was started from the main shaft itself. The pump station was cut out as shown in the plan Fig. 3, and the sump then started by drifting in beneath the station at a point two shaft sets below the station level.

This drift was cut out 10 feet high by 5 feet wide for a distance in of 12 feet. The height was then reduced to 4 feet, as seen in the sectional view (*b*) at *x*, and the direction turned 45 degrees, running it well to the side of the pump station, as best seen in the plan. The sump *k* was then excavated about 10 feet wide, 30 feet long, and 8 feet high, till opposite the end of the pump station, when a right-angle turn carried it again beneath the station floor, which was broken through, as shown in the plan. All of the rock excavated in this work was trammed directly to the shaft via drift *x* and dumped via a chute into a bucket run in the pipes compartment. In this way the usual hoisting by windlass at the top of the winze, as is commonly done, was entirely eliminated, so that the work proceeded more quickly and much more cheaply than ordi-

narily. With the excavation completed, a concrete dam l, Fig. 3, 3 feet thick, sealed the drift next the shaft. The dam was pierced only by a 4-inch diameter drain pipe set close to the floor and controlled by the usual valve. At the level of the station a concrete dam b was built from wall to wall of the pump station for a height of 2 feet above its rock floor in order to give additional storage capacity if ever needed in emergencies. This dam raised the floor of the pump station 2 feet above the floor of the shaft station. Another concrete dam and dividing wall j was built as shown at the rear of the sump, beneath the station floor, thus separating the main sump k from the main settling box i. At the top of the concrete wall j were placed the 8-mesh screens, best shown in the sectional view, Fig. 3. Then followed the construction of the plank receiving box c, and the double-compartment flume e, connecting with the settling box i.

In building the flume, its bottom was purposely laid on the rock floor just beneath the 12″×12″ transverse timber sills, so that the latter would form riffles for the flume. The heavy floor planks were then laid flush with the top of the concrete dam and over the flume and connecting boxes.

The course of the water will now be apparent. Arriving at the station level from the sinking pumps via the pipes a which pierce the concrete dam b, the water flows into the receiving box c. Vertically sliding gates d direct it into one compartment of the flume e. Here the riffles formed by the floor sills cause the coarse material to settle out at once while the finer silt finds its way into the main settling box i. By providing two compartments, both in the latter and in the flume, a given compartment may be cleaned while the other is running. The fine silt settles in box i, while the screens at the top of the wall j, over which the water flows, prevent chips, candle snuffs, etc., from going into the main sump k. The suction pipe of the pump condenser draws from the sump just beyond the screens, and its hot discharge returns to the sump at its further end just beneath the receiving box c. The pump suction pipe m draws from the sump close to the foot of the concrete pump foundation n. The aim of this unusual sump is to settle out all silt from the water and thus reduce wear on the pump valves, packings, etc., to a minimum. The advantage of getting the silt to settle in the riffled flume and settling box where it may be readily cleaned out, instead of in the inaccessible sump, is apparent.

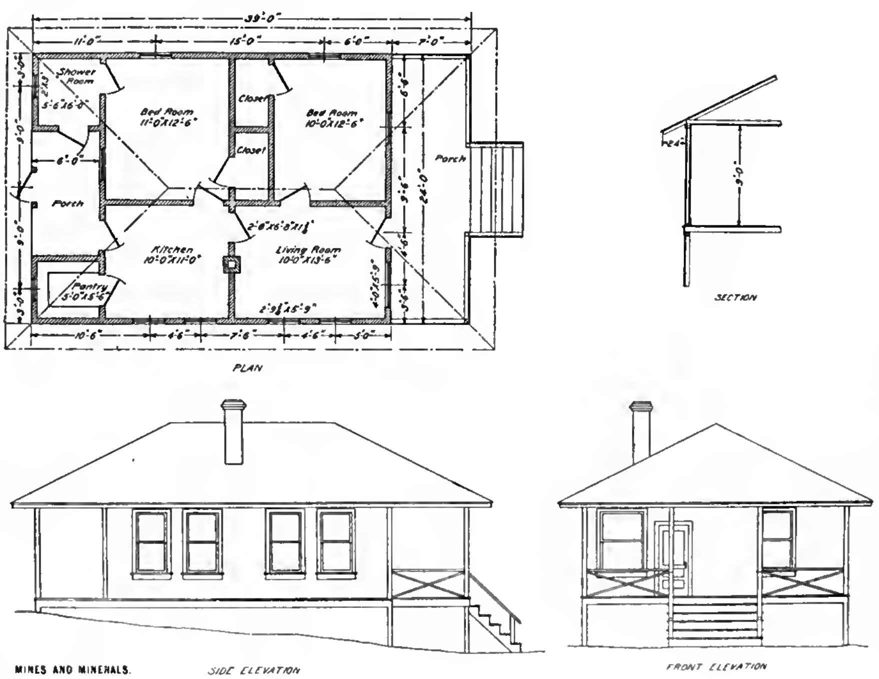


FIG. 5. MINER'S HOUSE

The main pump is a Prescott duplex, triple-expansion, condensing machine of the pot-valve type, size 16 inches and 25 inches and 42″×10½″×24″ stroke. Its capacity is 1,000 gallons per minute for a 1,000-foot lift. The column pipe is 12 inches in diameter, and between the pump and the pipe elbow at the shaft is set an expansion joint. The purpose of the latter is the absorption of the shocks and vibrations of the pipe due to the striking of the elbow by the water. These shocks, it is claimed, are apt to eventually cause the breaking of the elbow, thus seriously delaying pumping from the sump, as several mines in the Southwest can testify.

Before leaving this subject of pumping it may be of interest to call attention to a certain feature of the shaft construction which greatly facilitates the lowering of the heavy and bulky pump parts. This feature is the collar set of shaft timbers which is illustrated in the accompanying Fig. 4. While by no means new, the use of the collar set is not general and hence details concerning it may be of interest. The construction and framing of the timbers are plainly detailed in the illustration. It will, of course, be understood that from a constructive standpoint, the collar set takes the place of bearing timbers at the shaft collar, but gives considerably more rigidity to the shaft than bearers could. It remains only to remark upon the convenience the set affords. In lowering the heavy pump machinery it is of vital importance to get it properly centered at the start in order to avoid its being jammed between the timbers a short distance below. This centering is easily and quickly accomplished where a collar set allows a ready access to three sides of the compartment and an unhindered observation of the machinery from a roomy safe vantage point. The quick and easy repair of the shaft chairs at all times through such accessibility, whether hoisting is in progress or not, is likewise an obvious advantage.

Surface Equipment.—With the idea of considerably increasing the mine production, the hoisting, and indeed the entire surface equipment, has been designed for the future, rather than for the present output of 100 tons daily.

The boiler plant comprises two batteries of two units each, a unit consisting of an oil-burning Stirling boiler, water-tube type, of 250 nominal horsepower. There is a Cochran feedwater heater from which the boilers are fed by a duplicate set of Snow duplex pumps. The oil burners are of the Hammel type, to which the fuel is pumped by a duplicate set of Moore fuel-oil pumps.

Thus far the ore is hoisted in cars by cages, the hoisting being balanced, two of the shaft compartments serving this purpose. The hoist is a Nordberg, reversibly geared, duplex, size 14″ and 14″×28″. The two drums have a 7-foot diameter with a 36-inch face. This machine embodies some new features of design over the ordinary Nordberg type and is the only one of its kind thus far installed in the Southwest. The main new feature is the placing of both drums at one side of both the steam cylinders, an arrangement which has obvious construction advantages.

Compressed air is supplied by a Nordberg machine compressing to 90 pounds, and having a capacity of 1,200 feet of free air per minute. The compressor is of the two-stage, cross-

compound, Corliss valve type, with steam cylinders 13 inches and 24 inches and air cylinders 21 inches, and 12¾″×36″ stroke. It has two 300-square-foot surface condensers, and an attached 10″×12″ air pump.

Pipes and machinery in the pump compartment are handled independently of the cage hoist by another small hoist, provided for the purpose. It is noteworthy that for convenience all of the surface buildings are grouped about the shaft within a short distance of it and nearly inclosing it in a hollow square.

Besides machine and carpenter shops, comfortable change quarters with lockers, shower baths, etc., are provided for the men, all within 100 feet of the shaft. The greater danger of fire to all the buildings of such a group is anticipated by providing an ample number of fire hydrants conveniently disposed.

The Superior and Boston Co. has prepared for the comfortable housing of its married men by providing a number of model frame cottages. As the design is one which may appeal to others it is herewith reproduced in Fig. 5.

In the short time since active development was started on the property, under the supervision of Frank H. Probert, as consulting engineer, and the management of Superintendent John D. Wanviz, record progress has been made upon which the company is to be congratulated. The mine itself is expected by the writer to develop into a large producer of the Old Dominion class in a few years more, and as for its equipment, the visitor to the district is advised by the foregoing notes to actually inspect it for himself.

METAL MINING NEWS

Exports of manufactures in the fiscal year ending June 30 exceed those of any earlier year and imports of manufacturer's materials in 1910 were also the largest on record.

The statement issued by the Department of Commerce and Labor shows that 22 out of 50 of the principal articles of export are derived from our mineral resources. They are in the order of their relative importance to all exports in millions of dollars as follows:

Relative Importance	Article Exported	1909 Millions	1910 Millions
2	Copper metal	82.0	83.6
3	Illuminating oil	71.8	62.5
34	Paraffin	6.4	7.9
45	Crude oil	6.9	5.3
12	Lubricating oil	18.3	20.9
10	Bituminous coal	23.0	25.9
18	Anthracite coal	14.3	14.6
24	Rails	6.9	10.5
27	Pipes and fittings	8.0	9.6
32	Wire	7.3	8.4
38	Tools	5.9	6.7
43	Structural iron and steel	5.1	5.8
6	Mowers and reapers	14.1	11.1
30	Fertilizers	9.3	8.7
31	Electrical devices	6.1	8.7
36	Sewing machines	5.9	7.5
39	Locks, hinges, etc.	5.5	6.6
37	Typewriters	6.9	7.2
40	Electrical machinery	6.4	6.0
41	Metal-working machinery	3.6	6.0
44	Locomotives	3.5	2.4
50	Railway cars	3.5	2.2

To this is to be added the excess exports of gold and silver over imports which are for the fiscal year ending June 30, 1910, $75,273,310 in gold and $10,106,687 in silver. In the fiscal year ending June 30, 1909, the gold exports amounted to $92,527,574 in excess of imports. While the balance of trade is in favor of the United States on raw materials and manufactured articles, the exports of gold are made regularly to pay the expenses of American royalty, travelers, and expatriates. The interest on American securities held abroad is paid in part in gold but largely by bills of exchange.

Platinum in the form of sperrylite, a diarsenide, has been found in an abandoned copper-mine dump near Stobie Falls, Vermilion River district, Canada.

The Middlesbrough Steel Strip and Hoop Co., Ltd., is interested in a project for the establishment of an iron and steel plant, including blast furnaces and rolling mills, on Vancouver Island, near Victoria, British Columbia. The plant is to be placed close to deposits of magnetite at San Juan and Harris Creek, 60 miles northwest of Victoria. A sum of $110,000 has already been spent in preliminary development work at San Juan, and there is now a shaft 300 feet down with cross-cuts at 100-foot, 200-foot, and 300-foot levels. High-grade magnetite has been found at the 100-foot level, and the deposits are considered to be the largest workable on the Pacific Slope.

The Giroux Consolidated Mines is working 200 men, and appliances with machinery for development are now at the five-compartment Alpha shaft. It is said that ex-Senator W. A. Clark is interested and will extend the Clark San Pedro Railroad into the district through the Deep Creek country.

The control of the Giroux Consolidated lies in the Cole-Ryan syndicate, who also control the International Smelting and Refining Co., and the smelter of that company just completed at New Tooele. The ore from the Alpha shaft is direct smelting and not concentrating.

The Ely-Central Mining Co. is sinking two shafts, the Eureka and Clipper, from which cross-cuts are to be driven to the porphyry deposit supposed to be outside the boundary lines of the Nevada Consolidated. There is considerable confidence displayed in the outcome of this venture, for machinery has arrived to put the mines on a shipping basis. These mines it is also supposed will ship over the proposed Clark railroad.

To increase the daily capacity above its present 8,000 tons output, the Nevada Consolidated is having a few large locomotives constructed. Seven trains of 21 cars, each car carrying 55 tons of ore go daily to the concentrating mill at McGill. The smelter capacity is not equal to the quantity of concentrate made at this mill. This property is controlled by the Guggenheims, through their control of the American Smelting and Refining Co. It is stated that in May the Nevada Consolidated produced copper at a cost of 5.95 cents per pound. If so, this is a world-breaking record in low copper costs and was made in the face of the fact that only seven-eighths of the plant was in operation during the month of May. Had the plants operated full, production for the month would have exceeded a rate of more than 80,000,000 pounds a year.

It is reported that the South Utah Copper Co., the relict of the Newhouse mines and smelters, is to ship its product to the International Smelting Co. Previous to the break-down there was a contract between the American Smelting Co. and the Newhouse mines. It is assumed that the reorganization of the old company cancelled this contract.

The annual imports of iron ore into South Wales exceed 1,500,000 tons, of which Cardiff in 1909 received 662,000 tons. This ore, 50 per cent. in iron, comes from Spain and averages $5.10 per ton. The total output of iron and steel in South Wales is 900,000 tons per annum. The local demand for this iron and steel is for tin plates and galvanized sheets, of which South Wales is the leading district in the United Kingdom. Mainly owing to the great growth in the demand for oil cans, the tin-plate industry has not only suffered no reaction after the great boom of 1907, but has steadily increased in prosperity. Shipments from South Wales ports in 1909 were 375,969 tons, and during the first 5 months of 1910 there was an increase of 25,000 tons over the same period of 1909. Mills and machinery are working full time, additional mills are in course of erection, and workingmen have steady work at high wages.

The tin-ore deposits in the Malay Peninsula are virtually in the hands of an English proprietary company which precludes others from the right to mine, or, if they mine, to export or smelt at a profit. The United States buys their tin, coats oil cans with it, fills the cans and sells the whole thing to the natives. Reports say the natives use the empty oil cans for almost every conceivable purpose.

NEW MEXICO GOLD GRAVELS

*Written for Mines and Minerals, by J. A. Carruth**

The Rio Grande del Norte (Great River of the North) rises in the mountains that surround the San Luis Park, in Southern Colorado. This park lies at an elevation of from 7,500 to 8,000 feet. Its southern extension is in New Mexico, where it is known as the Taos Valley. On the east the main range of the Rocky Mountains rises abruptly many thousand feet, while further away from the river, on the west, lie the also high mountains of the Continental Divide. The San Luis Park and the Taos Valley are great level appearing plains between these ranges. In the San Luis Park the Rio Grande runs almost on the level of the country, having hardly any banks at all, but a few miles north of the New Mexico line it commences to cut into the sand and gravel of which the park is composed, and in a few miles it is several hundred feet below the level of the country, in a narrow gorge, usually from a half mile to a mile wide, and varying in depth from about 600 feet at the New Mexico line to over 1,000 feet in the White Rock cañon, 100 miles further south. Standing on the level plain in the Taos Valley there is no sign of a river, but moving ahead a short distance you come to a cañon 800 feet almost perpendicular, and the Rio Grande at the bottom looking like a little creek. From a higher point of ground the river can be seen as a black streak through the level plain for a distance of 60 miles. The most of the ground that has been washed out was gravel, though in the northern part of the valley there is considerable lava mixed with the gravel.

Deposits of Gold-Bearing Material of Great Depth and Extent, With Values Evenly Distributed

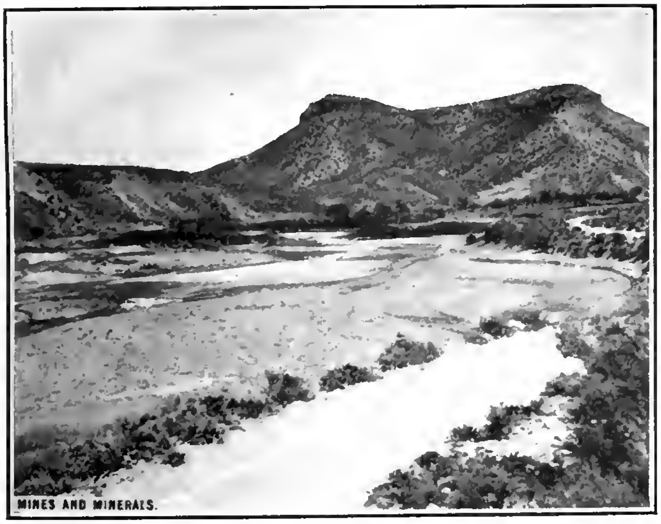


Fig. 1. Bluff near Cieneguilla

Fig. 2. View Looking Down Stream from Bluff

Commencing a little above the entrance of the Rio Colorado, or Red River, into the Rio Grande from the main range on the east, about 20 miles south of the Colorado line, these gravels are found to carry gold. The Red River heads in a highly mineralized section, and going south, in succession, are found the San Cristobal, Rio Hondo, Taos and Embudo rivers, all emptying into the Rio Grande from the east, and all carrying auriferous gravel. The western banks of the Rio Grande are not as rich in gold as those on the eastern side of the river.

In the early 80's Governor Norton, of Nebraska, and parties interested with him, secured the services of the late Prof. Benjamin Silliman, of Yale University and secretary of the American Association for the Advancement of Science, to examine and report on the section of land between the Red River on the north and Embudo Creek on the south, a distance of 40 miles. Professor Silliman spent 3 or 4 months in this examination, and his report, comprising over 40 typewritten pages, is very full and explicit. Extracts taken from it, give better ideas of the country, and its richness than can be had from other sources. After a little preliminary matter Professor Silliman says:

"Here are millions of tons of gold quartz reduced by the great forces of nature to a condition ready for the application of the hydraulic mining process, while the entire bed of the Rio Grande for over 40 miles is a sluice, on the bars of which the gold derived from the wearing away of the gravel banks has been accumulating for ages and now lies ready for extraction by the most approved methods of river mining.

"The thickness of the Rio Grande gold gravels exceeds in many places 600 feet, or nearly three times that of similar beds in California, while the average value per cubic yard is believed to be greater than in any other accumulations yet discovered. Only those portions of this area, however, which are within reach of the waters of the Rio Grande, and its affluents, are at present available.

"The area demanding our attention is confined chiefly to the valley of the Rio Grande from a point near the entrance of Embudo Creek from the east, about 45 miles north of Santa Fe, and following the course of the river for about 40 miles. The whole of this area contains vast accumulations of auriferous gravel, not less where I have measured it than 600 feet in thickness in many places. I have made a reconnaissance of the whole of this ground along the Rio Grande, and have examined with all the care possible in the time at command the character of the gravel and its content of gold, also, to some extent, the source from whence it has been derived, its geological peculiarities and its relations to water as the means by which it may be worked to advantage. Nothing, I am persuaded, since the discovery of California and Australia is comparable for its measurable reserves of gold available by the hydraulic process to these deep placers of the Rio Grande.

"I distinguish two chief varieties of gravel in these Rio Grande beds. First, that which may probably be called the gray or blue gravel, and second, the yellow gravel. The first is coarser than the latter and as a rule carries more gold than the second variety. A place of convenient access where these two distinguishing varieties of gravel may be seen is at Cieneguilla, about 54 miles from Santa Fe. At this point is a bold cliff of gravel surmounted with a cap of mal pais, on the face of which the strata of these two varieties of gravel are seen in the alter-

*Santa Fe, New Mexico.

nate beds. The cap of volcanic rock is about 30 feet thick, and the first bed of blue gravel beneath it, which contains abundant free gold, rests unconformably on the lower strata, which have been tilted up from the east to an angle of 10 or 15 degrees from the horizontal and curved at the same time. The yellow gravel here gives 8 or 10 colors to the pan, while

FIG. 3. VIEW LOOKING DOWN CAÑON

the blue is worth probably 50 or 75 cents per cubic yard. A photographic view of this remarkable cliff, which is approximately 600 feet high, would be highly instructive.

"One searches in vain for the light gravel, poor in gold, which forms the top dirt, for example, at the Malakoff shaft in the North Bloomfield gravel beds, in Nevada County, Cal., but the great mass of these heavy beds are compact auriferous gravel, and contain boulders of quartzite, with blue or gray stains and seams of magnetic iron and rusty quartz, with comparatively few granite or syenite pebbles, and fewer of porphyry or greenstone, and no volcanic debris or ashes. I estimate that the quartz and quartzite pebbles form at least 60 per cent. of the gravel in these beds.

"The vitally important question of the value of the gravel has received the careful consideration of a considerable number of good prospectors and miners accustomed to judge of the value of auriferous gravel. You have their written statements and opinions all agreeing as to the main fact that this auriferous gravel is unusually rich. They differ only as between 50 and 75 cents to the cubic yard, with examples of very much higher values than this. The average yield of these gravels as far as known is 50 cents per yard. The gold in these alluviums is diffused with remarkable uniformity so far as observed. It is bright, but not so thin and worn as in most of the California gravels. Much of it is, in fact, angular and rough, like quartz gold."

Fig. 1 is a view of the bluff mentioned by Professor Silliman. The dredging ground and river show in the foreground. The river flows on the left of the bluff and on the right is a deep arroyo The bluff is 900 feet to the surface, which is the level of the surrounding country. Both the bluff and the low bluffs on the left carry gold in good paying quantities.

Fig. 2 is a view taken part way up the bluff shown in No. 1, but is looking down the stream All on the right is gravel, but the cliffs on the left are 1,400 feet high and are quartz.

Fig. 3 is a view looking down the river cañon from the level of the country, or mesa, 900 feet above the river, on the west side. Indistinctly seen over the level mesa on the east side at the left of the view, is the main range of the Rocky Mountains, some 40 miles away The bluffs on the right are some of the foot-hills.

Fig. 4 is a view of an old channel of the river, about 300 feet above its present channel. It can be traced for a couple of miles.

Some years after Professor Silliman made his report, Cecil A. Deane, a Denver mining man spent an entire summer going over these gravel beds. His report, like Professor Silliman's, is too long for print in full, so extracts are given:

"At some remote period a vast overflow of lava, coming from the northwest, spread over thousands of square miles in this region, and is of unknown thickness. The Rio Grande has forced a channel through the basalt, and the bed of this stream is now hundreds of feet below the level of the adjacent plateaus. It is stated that in a distance of more than 100 miles there are but three places where the river can be crossed with wagons. From the east the Taos flows through a like cañon for about 8 miles, where it joins the Rio Grande. Not only over the great area between these two streams, but extending to the south and east of the Taos, and from the Red River on the north to the Arroyo Hondo on the south, a region embracing hundreds of miles in area, and resting largely on the nearly level surface of the lava, are found the largest now known deposits of auriferous gravel. The causes which produced such vast accumulations are not definitely known; for while the deposits plainly show evidence of attrition due to the force of water currents, they are in depth and width so great as to preclude the idea that such are wholly due to river action.

"Glacial forces, when other than present climatic conditions prevailed, may have produced such deposits, when vast fields of ice were forced down from the Sangre de Cristo range, crushing and grinding and moving forward from the mineralized rocks now found there in place on the western slope. The numerous moraines in the cañons of the Taos and Rio Grande could only have been so left by melting glaciers.

"Along the Rio Grande and its tributaries mining operations by means of ground sluicing have been carried on in a small way ever since this locality was first possessed by the Spaniards. Where the Taos joins the Rio Grande evidence of their workings is plainly seen in the ruins of a stone fort and their nearly filled up acequias. While the altitude is nearly 7,000 feet, yet because of this locality being sheltered by the encircling mountains, and because of its more southern latitude, placer mining can be carried on during fully 9 months of each year.

FIG. 4. OLD RIVER CHANNEL

"At a central point, about 5 miles northeast of Taos, is found abundant evidences of labor, probably employed in placer mining, yet so unusual in many respects as to warrant other conclusions: An area of some 3 by 5 miles, extending toward the southwest from the abrupt foot-hills of the Sangre de Cristo

Range, is so marked, and whereof no authentic record is now known to exist. Over these many thousand acres of gently sloping ground, now supporting only a dense growth of sage brush, nurtured by a very sandy soil of less than a foot in depth, we may observe parallel ridges of quartzite, water-worn pebbles and stones, the largest not exceeding one-fourth of a cubic foot. Such ridges are seldom more than 20 feet apart, from 1 to 4 feet in height, frequently extending in direct lines for several hundred feet, and at intervals having like ridges of similar stones connecting at right angles the opposite side.

"Excavations made among such low walls prove the absence of broken pottery, of flint arrow heads and of all other usually found traces of resident occupation. A probable solution is found in the fact that where the gravel has been undisturbed small quantities of gold occur, and it is a historical fact that in 1640 this locality was wholly under the control of the Spanish government, that for quite 40 years thereafter the native race was subjected to servitude of the most cruel nature, that during such period their foreign oppressors caused them to work in mines or in raising subsistence for captor and captive.

"It is stated that there is now among the archives of the Mexican capitol a document dated prior to the rebellion of 1680, made by the then governor at Santa Fe, asking the Spanish viceroy to furnish an armed escort to convey a shipment of gold dust, valued at more than $2,000,000, from a point north of Santa Fe, and that such troops were furnished and shipment safely made. So far as now known no evidences of extensive placer deposits are found north of Santa Fe except in this locality.

Even with the aid of the crude appliances as then used, with no cost for labor or supplies to be met, and such labor employed for many years, the gold product, even if yielding but a few cents per day for each laborer, the aggregate accumulation of the precious metal would be very great.

"Recently numerous tests have been made in this section to determine the depth of the gravel and its value in gold. The results of tests made of many yards washed in sluice boxes prove that the gold is uniformly distributed.

"When the limit of the lava overflow is reached the deposits of gravel are of unknown thickness. Some years ago Mr. A. Gusdorf, a merchant of Ranches, of Taos, sunk a well to a depth of 425 feet at his place for the purpose of securing artesian water. Throughout its entire depth gravel only was met, and the material removed proved the presence of gold for the whole distance.

"In a great majority of placer deposits obstacles are encountered which prevent their successful working; the supply of water may be insufficient, the gold-bearing deposits may be commingled with boulders of such size that the use of derricks and powder must be employed in their removal, or the fall be so small as to afford little space for disposal of debris after it is discharged from the sluices; but such obstacles are not encountered here; the supply of water is limited only by the cost to be met in making it available, no stones are found in these vast deposits exceeding one-fourth of a cubic foot."

All the available land on the Rio Grande is held at present by different companies who are making plans to be worked in the near future. One of these put down an air caisson in the river bed to bed rock, 33 feet, and the values were found to be very evenly distributed after the first few feet, the value at bed rock being $6.25 a yard, with an average for entire distance of $3.03 per yard.

The Mutual Placer Co., of New Mexico, has recently made arrangements for raising the funds needed to put in machinery. The photographic views shown herewith were mostly taken on its ground and give a good idea of the general appearance of this section. With good machinery all of the companies interested there will become good and long-continued dividend payers.

DETERMINATION OF LATITUDE

*Written for Mines and Minerals, by C. E. Rowe, B. S. (C. E.), E. M.**

It is often of interest to know the latitude of a place, and, furthermore, it is necessary to use the correct latitude in determining the meridian. If a solar attachment is used to find the meridian it is necessary to set off an angle equal to the latitude or colatitude of the station before the observation is made. If a direct solar observation for meridian is to be made, as described in MINES AND MINERALS†, the observation does not depend upon the latitude, but the calculation does. Good results in either method usually require the latitude of the station to be known within an error of ½ of a minute.

Method of Direct Observation Using an Ordinary Mining Transit

The method of determining the latitude, as here described, is simple both in observation and in calculation. The reader is referred to the article mentioned in the foot-note for certain details which are needed in the calculation.

FIG. 1

The Observation.—The direct solar observation for latitude is made with an engineer's transit at noon, that is, when the sun is at his highest position, and to make the observation the sun is followed until he reaches his zenith. The only data to determine are the vertical angle to the sun's center and the approximate standard time of observation. The accuracy of the resulting latitude will be exactly equal to the accuracy with which this angle is found.

For convenience it should be known approximately when the sun will be at the greatest height, which is usually not more than 45 minutes from noon by standard time, depending upon

* Head of School of Mines, University of Texas, Austin.
† Vol. XXX, page 488.

the location in the time belt and the "equation of time," which is the difference between *apparent* and *mean* time. Knowing this it will be unnecessary to begin the observation many minutes in advance.

The transit used should preferably have a full vertical circle so that a normal and reversed telescope observation can be taken, the average of which eliminates the error of adjustment as a level and the index error, but does not eliminate the error due to "leveling" the instrument. The leveling operation is finished when the bubble of the telescope vial, which is much more sensitive than the plate levels, stands at the center of its run when pointed in any direction. This is true whether it is in adjustment or not. A normal and reversed telescope reading can be taken because the sun remains at practically the greatest height for several minutes. Perhaps the best method of being sure that the sun has been observed at his highest point is to continue the observation until he begins to descend. If the meridian is known, the observation can be made when the sun crosses it, or if the longitude is known the exact time for making the observation can be calculated. Of the four things which an engineer may determine from solar observations, the latitude is usually the first required, and does not depend on the others. The meridian is the second, depending upon the latitude. Longitude and time are the third and fourth, being mutually convertible and depending upon the meridian for their easy determination.

During the spring and summer the altitude of the sun for

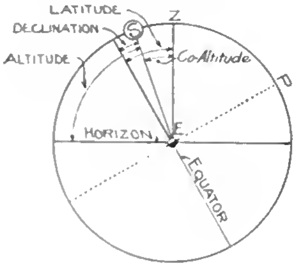


FIG. 2

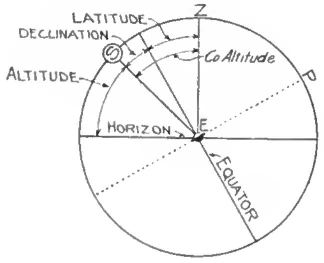


FIG. 3

the southern part of the United States reaches about 83 degrees (i. e., at latitude 30 degrees). On account of the interference of the plates in attempting to measure high angles, it may be necessary to use a prismatic eyepiece attachment to turn the sun's image out of the regular eyepiece at a right angle to the telescope. Furthermore, it may be necessary for the same reason to receive this image upon a screen held about 5 inches from the eyepiece, as shown in Fig. 1. The colored glass of the eyepiece is not used in this case. In order that the cross-hairs may be visible, the eyepiece must be drawn out a little further than when looking through it. Receiving the image directly on a screen‡ is objected to by a few surveyors, yet it is preferable to using the prismatic eyepiece for moderately high angles, because a notebook or some flat blank surface is always available for such a purpose. The glare from a small white screen does not affect the eyes more than the direct view through a colored glass, and a tinted surface is usually better than a white one in this respect. For low angles the best equipment is a piece of dark green glass set in the sliding cover of the eyepiece, since it is always at hand, requires no refocusing, and the view of the sun's image in relation to the cross-hairs is excellent.

The sun being about 32 minutes in diameter, some special method must be used for accurately obtaining the vertical angle to his center. There are four good methods:

1. In one observation a very small segment may be cut from the top of the sun's image with the cross-hair, and in the other an equal segment may be cut from the bottom, the average vertical angle being used.

2. A small segment may be cut from one side of the image with the vertical cross-hair and the segment can be quite accurately bisected with the horizontal cross-hair.

3. The instrument may be provided with solar hairs, of which the "Davis inclined square" is an excellent type for the transit, making it possible to accurately set upon the sun's center by cutting off four small segments of the image by means of hairs crossing the reticule at an angle of 45 degrees, as shown in Fig. 1, on the image.

4. If the observation is made by looking into the telescope through a colored glass, the horizontal cross-hair may be placed tangent to the sun and the vertical angle corrected for the sun's semi-diameter.

‡ Fig. 1, page 488, Vol. XXX.

The Calculation.—By reference to Figs. 2 and 3 the method of making the calculation is apparent. These sketches show a side view of the imaginary celestial sphere. The sun, S, is shown in Fig. 2, at north declination, and in Fig. 3 at south declination. The planes of the equator and of the horizon intersect the plane of the paper along the lines indicated. E is the earth, Z the zenith point, and P the north pole produced. The coaltitude of the sun is 90 degrees minus the true altitude. If we call north declination plus, and south declination minus, it follows from the sketches that

$$\text{Latitude} = \text{coaltitude} + (\pm \text{ declination})$$

The altitude of the sun is the vertical angle to the center, *minus* the correction for refraction. In the article March, 1910, MINES AND MINERALS, referred to in the foot-note, a table of mean refractions is given. The refraction may be calculated from the formula:

$$\text{Refraction} = 57 \text{ sec.} \times \text{cotangent apparent altitude}$$

The declination of the sun is obtained from a "solar ephemeris."

EXAMPLE.—On April 13, 1910, the average of a normal and a reversed observation gave the vertical angle to the sun's center as 68° 38.75′ at 12:35 P. M., 90th meridian time. Find the latitude of the station.

SOLUTION.—

Vertical angle by observation	= 68° 38.75′
Correction for refraction = 22 sec.	= .37′
True altitude	= 68° 38.38′
Coaltitude = 90° − altitude	= 21° 21.62′
Sun's declination at Greenwich *mean* noon (= 6 A. M., 90th meridian time)	= + 8° 49.60′
Difference for 1 hour = + 54.62″. Diff. for 6 h. 35 m. = (6.6h × 54.62) ÷ 60	= + 6.00′
∴ Declination at 12:35 P. M.	= + 8° 55.60′
Coaltitude (above)	= 21° 21.62′
Hence, Latitude = coalt. + (± dec.)	= 30° 17.22′

This is the result of an actual observation, and does not give exactly the known latitude of the station, the error being ¼ of a minute, which is fully as close as can be expected since the observation was taken with a 5-inch mining transit graduated to read only to minutes, but by estimation was read to the nearest ½ minute.

ST. LAWRENCE POWER PROJECT

There was a bill before the Canadian Parliament for development of power from the St. Lawrence River. It is stated that parliament will not sanction the erection of a dam in nor across the river, and no additional work in connection with the enterprise may be undertaken without the direct approval of parliament. Sir Wilfrid Laurier has denied that the government was in any way responsible for the bill or concerned with its fate, and states that the St. Lawrence is so bound up with the history of the country that any attempt to impede its flow is something like sacrilege. For the present he will not agree, for any consideration, to dam the river, although in course of time the demand for electric energy may become so great as to compel parliament, under the direction of the International Waterways Commission, to reconsider the whole question.

TRAIL SMELTER AND LEAD REFINERY

*Written for Mines and Minerals, by J. M. Turnbull, B. A., S. C.**

A Description of the Equipment, Methods of Smelting and Electrolytic Refining

The smelter and lead refinery of the Consolidated Mining and Smelting Co., of Canada, Ltd., are situated at Trail, in the West Kootenay district of British Columbia. A branch line of the Canadian Pacific Railway affords the works connection with a number of the mining camps of southern British Columbia.

Practically all of the lead ores and most of the other custom smelting ores of the interior of the Province are treated here, owing to central location, low freight and smelter rates, and diversity of operations.

For the year ending June 30, 1910, the ore smelted averaged 9,500 tons per week of all classes, of which 1,100 tons were lead and dry ores or concentrates. The metal production for the same period had a gross value of just under $6,000,000, of which over 40 per cent. is gold. This is $500,000 more than the previous year. The total production since 1896 having amounted to $43,700,000.

The first furnaces completed in 1896 had a capacity of 200 tons per day. The Canadian Pacific Railway bought them in 1898 and sold to the present owners in 1906. Lead furnaces were installed in 1899, and the lead refinery, with Betts patent electrolytic process, in 1902. Changes and enlargements have been almost continual, and although some disadvantages of arrangement have been inherited in this process, the plant as a whole is modern and efficient.

FIG. 1. GENERAL VIEW OF TRAIL SMELTER

The location of the plant, as shown in Fig. 1, is on the corner of a bench of land, which rises 200 feet above Trail Creek and Columbia River valleys. The sides of the bench are steep, permitting advantages to be taken of gravity on both slopes, comparatively little elevating of ore or materials being required. The slag is granulated and sluiced to the dump by means of an ample supply of water from several mountain creeks, which reaches the furnaces under a head of 140 feet. Slag disposal is thus cheap and easy.

The municipality of Trail is at present building a launder from the furnaces over the town, where the granulated slag will be used to fill in a large depression in the town site below the furnaces.

The compact clump of buildings on the left of the illustration is the refinery. Residences of officials are in the background. The storage yards and large sampling mill are in the center, as well as a 6,000-ton pile of coke. Further to the right and behind are the roasting furnaces and Huntington-Heberlein plant, while across the front from left to right are the old sampling mill, the blower room, the matte plant, and blast-furnace building, 225 ft. × 70 ft., and 70 feet high. The slag dump and part of the town of Trail are in the foreground, with the Columbia River in the rear.

The general equipment includes railway yards, with 3 miles of sidings, railway scales, ore bunkers, storage yards with a capacity of 30,000 tons, three sampling mills, five large copper blast furnaces, two lead furnaces, with seven rotary blowers, two O'Hara furnaces, six Bruckner roasters, seven Huntington-Heberlein furnaces, and 25 converters, a lead refinery, briquetting plant, assay office, machine, carpenter, and boiler shops, together with a large amount of smaller machinery and equipment, including an extensive installation of pumps, hydrants, etc., for fire protection.

Electric power is used throughout, being generated at Bonnington Falls, 33 miles distant, and transmitted at 20,000 volts to the works, where it is stepped down to 550 volts by a battery of transformers. The total power consumption is about 3,000 horsepower.

The ores received are weighed over a 100-ton recording Fairbanks scale, dumped into bunkers and drawn therefrom to the proper sampling mill. The bulk of the copper and low-grade ores is handled in the large mill, whose equipment consists of a No. 8 and a No. 4 McCully crusher, together with Vezin samplers, smaller crushers, rolls, etc. The first sampler cuts out one-tenth, following which, in order, are cuts of one-fifth, one-quarter, and one-tenth. The corresponding crushing sizes being usually 4, 2, 1, and ½ inches. One two-thousandth of the original ore is thus delivered to the quartering floor where it is quartered down by hand or in Jones riffles. Part of the final sample goes to the assay office and part is kept as a check until the shipper is paid.

The old mill has a smaller but similar equipment and is used as a reserve and for sampling special lots, or as convenience dictates.

The third mill handles lead and dry ores and concentrates. Its equipment consists of Blake and Farrel crushers and rolls, with Vezin and Brunton samplers, the capacity being 30 tons per hour. Oxidized lead and dry ores are first crushed to a 6-inch ring, the final sample being ¼-inch, and the reject going direct to the furnace charge bins. Galena ores and coarse concentrates are crushed to ¼-inch and bedded. In each case the samplers cut out one five-hundredth part and deliver it to the quartering floor. Fine concentrates are usually sampled by the fifth-shovel method and go to the lead beds. The large mill was designed so that all sampling could be done on one shift with resultant economy and more efficient supervision, and it is now being arranged to handle lead and dry ores as well.

The blower plant comprises seven rotary blowers operated by individual motors. The older ones are Connersville type while the newer ones, four in number, are of the later Root type. The largest delivers 33,000 cubic feet of free air per minute at a pressure of 36 ounces and consumes 350 horsepower. The total power required for the plant is 1,200 horsepower. A small

*Mining Engineer of the Consolidated Mining and Smelting Co., of Canada Ltd., Trail, B. C.

rotary blower in the roasting plant furnishes air for the converters.

The haulage and distributing system includes a high-line trestle running through the sample mills and having branch trestles over the storage yards, lead beds, etc. Twenty feet lower in elevation is the second haulage line. This runs below all the sample mill bins and into a tunnel under the coke and storage yards. The ore and coke are drawn through chutes directly into cars and run over the furnace charge bins which have a capacity of 4,000 tons. These haulage lines are connected by a pair of balanced elevators.

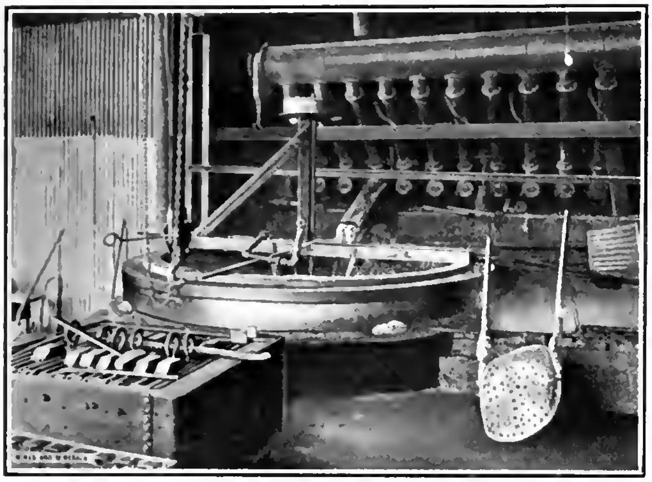


FIG. 2. LEAD WELL AND ANODE MOLDS

The third line, 25 feet lower, is on the level of the copper furnace feed-floor. It runs under all the furnace charge bins and has branches connecting it with the ground floor of the roasting and briquet plants. Each copper furnace has a track on either side on which the charge trains are alternately run.

An incline operated by endless chain connects the lower furnace floor with the second and third haulage levels to raise the bullion anodes, slag, matte, etc., to higher levels.

Ten Jeffrey electric trolley locomotives on the three upper lines are required to handle the traffic, hand tramming and shoveling being reduced to a minimum. Auxiliary hand tram roads are used where convenient or economical, as in the lead beds, etc.

A motor driving two 100-kilowatt generators is used to generate the necessary direct current. An auxiliary or reserve set of 85 kilowatts is on hand to prevent the serious complications which would follow a break in the haulage service.

A railway spur on a trestle 12 feet above the high line delivers coke to the storage yard, and a large store of coke, some 6,000 tons, is kept on hand to provide against contingencies at the coal mines. This trestle extends over the bins which deliver ore from their chutes directly into the mouth of the large sample mill crusher.

An extensive system of brick and metal flues is installed, connecting with two brick stacks, one for the copper and one for the lead furnaces. The copper furnace stack is 185 feet high and 144 square feet internal area. The flue dust, amounting to 2 per cent. of the ore, is briquetted and resmelted. The briquet plant includes two Chisholm, Boyd & White presses with pug mills, elevators, etc. Three to 5 per cent. of lime is used as a binder.

The assay office, built in 1902, is completely and modernly equipped. An electric motor drives the crushers, and electric heaters are used. It has four large coal-fired muffle furnaces, and a complete equipment for wet and electrolytic work. A great variety of assay and analytic work is called for, including that of lead-refinery products and solutions.

The machine, boiler, and carpenter shops are sufficiently equipped to handle all ordinary repair and construction work connected with the smelter.

The ore supply is drawn largely from the East and West Kootenay and Boundary districts of British Columbia, with some silicious gold ores from the state of Washington.

Rossland camp, 11 miles distant by rail, supplies about 4,000 tons of ore per week, averaging about 1 per cent. copper and 9 to 12 dollars per ton in gold, with 45 per cent. silica and 7 to 8 per cent. sulphur in the form chiefly of pyrrhotite. Most of this is from the Consolidated company's own Center Star mines. The Boundary district supplies 2,000 to 4,000 tons per week of self-fluxing ore, running about 1.3 per cent. copper and $1.50 in gold and silver. This is from the Snowshoe Mine on which the Consolidated company have a lease, and is shipped a distance of 110 miles in quantities to suit the demand of the smelter. The Consolidated company also owns a group of properties adjacent to the famous Granby mines with similar ore, but these are not operated at present, being held as reserves.

The largest and steadiest supply of lead ores comes from the St. Eugene mines, of East Kootenay, owned by the Consolidated company, which has been the largest lead producer of the province for some years. The balance of the consumption comes from a large and variable number of smaller shippers and includes high- and low-grade copper ores, galena ores, and concentrates, gold quartz ores, and gold-mill concentrates, with some dry silver and gold ores.

Coke comes from the Hosmer Mines, Ltd., near Fernie, B. C., a distance of about 249 miles, about 225 tons per day being used. Limestone is derived from the company's quarry at Fife, 68 miles away, and costs $1 per ton in the charge bins. About 225 tons per day are required, mostly as flux, a little being burned and used as a briquet binder. No other barren flux is used, except a little scrap iron on occasions.

The five copper blast furnaces are of the following sizes: Two 42 in. × 240 in., two 42 in. × 300 in., one 42 in. × 263 in. at the tuyeres. The height of charge is 9 feet above the tuyeres, with air pressure of 36 ounces. Two forehearths are used, the second catching only from .25 to .5 per cent. of the total matte.

An electric locomotive takes a charge train of five cars of 1,000-pounds capacity, usually one loaded with coke, one with lime rock, and three with ore. When Snowshoe ore is being charged, two such trains alternate with one carrying one car of coke and four of Snowshoe ore, which is limey. The cars are dumped one at a time into the side of the furnace, the train moving the full length of the furnace for each car dumped, thus distributing the charge evenly, coke first, followed by lime, and lastly ore. An electric recording gauge in the superintendent's office shows the time that each charge goes on each furnace.

FIG. 3. LEAD REFINERY

In the low-grade furnaces the charge is varied according to the ore, giving a slag of the following analysis: Silica SiO_2, 43 to 46 per cent.; iron (FeO), 14 to 15 per cent.; lime CaO, 20 to 22 per cent.; copper, .1 per cent., with trifling amounts of gold and silver.

The practice at present is to smelt the ores raw to a low-grade first matte, which carries a variable amount of copper, say about 15 per cent. This appears to be an efficient gold collector. Snowshoe ore can be smelted alone in one operation to a 45-per-cent. copper matte, but it is found more advantageous to run it on the high-grade furnace with the roasted low-grade first matte; part of it sometimes being put on the low-grade furnace, as circumstances dictate.

The first matte running about 15 per cent. copper, 27 per cent. sulphur, and 56 per cent. iron, is roasted in two O'Hara furnaces of 9 ft. × 95 ft., and 12 ft. × 97 ft. hearths, respectively, to about 10 to 12 per cent. sulphur, then blown in the lead converters to a product which runs 1 to 3 per cent. sulphur. The converter operation is described under the lead operation later.

The roasted matte is smelted in the high-grade furnace with Snowshoe ore and some low sulphur ore to a matte running about 40 per cent. copper, a higher percentage not being economical under the circumstances.

This matte is shipped to Tacoma, Wash., for further treatment, no copper converters being installed in the works as yet.

The high-grade slag runs about 40 to 42 per cent. silica, 26 to 28 per cent. iron (*FeO*), 16 to 18 per cent. lime, and 10 to 12 per cent. alumina, with .35 per cent. copper and .2 ounce in gold. Some of this slag is put on the lead furnaces, as noted later, and the gold partly recovered.

The first low-grade matte is granulated by being poured into a stream of water which is caught in a tank, the overflow from which runs through a series of settling tanks, the granulated matte being elevated into the matte plant, drained, and run to the O'Hara furnaces. The high-grade matte is tapped into pots, cooled, crushed, elevated into the matte plant, sampled, and shipped direct.

The lead smelting and refining is perhaps the most interesting part of the work. An excellent paper by Mr. A. J. McNab, general superintendent, appeared in the *Canadian Mining Journal* for July and August, 1909, describing the lead smelting and refining at that date. The following is partly abridged therefrom, and is brought up to the present date, Mr. McNab having kindly checked the information and facts given in the whole of the present article.

The lead ore supply is mainly heavy sulphide with comparatively little dry ore, the reverse of usual American conditions, the lead furnaces carrying a charge of from 35 to 45 per cent. lead, compared with the usual 10 to 15 per cent. More than 80 per cent. of the ore received will run from 20 to 75 per cent. lead, the balance consisting of dry gold and silver ore, a little oxidized lead ore, and pyritic mill concentrates.

The general procedure is to treat the heavy sulphides with lime rock, lead matte, etc., in the Huntington-Heberlein roasters and converters. The resultant lead sinter is smelted in the blast furnace with the addition of certain dry ores, etc. The bullion from the lead well is cast into anodes, the dross being skimmed and returned to the furnace. The anodes go to the refinery where they are electrolytically treated, the final products being pure lead, pure gold and silver, bluestone, and sometimes antimony.

In general it may be said that proper proportioning of the roaster and furnace charges is the most vital point in getting good results. Provided this is well done the particular percentages may, and often do, vary widely from the figures given. These figures represent the latest average practice, but do not hold for any length of time, owing to the variations in the ore supply.

The lead beds are made up with heavy ores and concentrates only, contain some 600 tons each, and carry about 38 to 40 per cent. lead. Other constituents of the roaster charge are added later, because, if mixed with the beds, it would be more difficult to alter the charge quickly.

The making up of the Huntington-Heberlein roaster charge is important, as a poorly proportioned charge will neither roast well, nor will the resultant product smelt in the furnace without considerable trouble.

The roast at present is being run on approximately the following analysis: Lead, 40 to 44 per cent.; iron, 10 to 13 per cent.; silica, 8 to 11 per cent.; lime, 7 to 10 per cent.; the zinc being kept under 10 per cent., if possible. The lead content has varied to date between 36 per cent. and 50 per cent. Ores of either percentage have been roasted and converted with satisfactory results by carefully proportioning the other ingredients, and it is probable these limits might be exceeded if occasion demanded. The iron should at least equal the silica, and had better be 1 or 2 per cent. higher. Two to 4 per cent. excess of silica over iron has led to serious furnace troubles, decreased furnace tonnage, slags higher in lead, and a tendency to run too hot. Attempts to correct this by varying the blast-furnace charge or altering the lime in the roast have not been very successful. The higher silica seems to make the charge more fusible, melting higher in the furnace, and thus causing trouble. Lime in the roast is usually 7 to 10 per cent., though it is possible that higher iron, if available, might allow of lower lime. This would save some money, as the iron would be in ore on which treatment charges are paid, whereas the lime costs both to roast

FIG. 4. LEAD REFINERY, OPPOSITE END FROM FIG. 3

and to buy. Pyritic concentrates and matte form part of the charge.

Properly made up, carefully fired, roasted, and converted, the product will be fairly hard, yellowish or grayish yellow, showing some litharge, and be easily broken up with little fines. In the furnace it will run with cool top and bright tuyeres.

There are seven circular Huntington-Heberlein roasting furnaces, 26 feet in diameter, with hearths revolving once in 3 minutes. Stationary cast-iron rabbles work the ore through in about 2 hours.

The firebox and discharge are on the periphery and the feed in the center, the heat thus increasing toward the finish. The feed carries 14 to 17 per cent. sulphur, and the roast 8 to 9 per cent. The capacity is 35 to 45 tons per day per roaster, but varies with the per cent. of sulphur. For good desulphurization in the converters, the roast should not run over 9 per cent. sulphur, and preferably 8 per cent., according to experience at Trail. The temperature, a dull red heat, requires close regulating to roast fully without fusing, as fused ore will rapidly crust up the furnaces, which must then be stopped and cleaned. The roasted product should consist of small, uniform, semi-fused globules, showing neither lime rock nor ore. The roast is elevated and dropped through a water spray into a brick bin. The moisture, about 5 per cent. thus imparted, seems to prevent the converter blast from blowing through too quickly and keeps the heat in until the converters are working uniformly, and the temperature becomes high enough to ensure fusing of the charge.

There are 24 converters, 8 feet $8\frac{1}{2}$ inches in diameter, with cast-iron diaphragms, made in four sections and bolted together. One-piece diaphragms tend to crack, and steel ones to bulge with the heat. A fire of slabs is started in the converters and when burning freely the charge of 10 tons of roast is dropped in from a hopper, and a 10- to 12-ounce blast turned on. The time of converting varies widely with the charge, averaging three charges per 24 hours, the blast decreasing to 2 ounces at the close. The heat from the oxidation of the sulphur, iron, etc., gradually works up through the charge to the top and burns out, leaving a solid mass of porous sinter. This is dumped on to an iron cone which breaks it up into large pieces. These in turn are broken by hand and fed to a 20″×20″ Blake crusher, crushing to 6-inch ring. The crushed sinter is elevated and goes to the furnace charge bins.

The chemical reactions of converting are somewhat obscure. Some lead is reduced and desulphurization is pretty thorough.

The two lead blast furnaces are, respectively, 45 in. × 160 in. and 45 in. × 140 in., at the tuyeres. They have smelted 260 tons per day, and will average about 190 tons of ore, besides by-products, matte, slag, etc., of from 20 to 30 per cent. of the charge. They are standard type, with brick crucibles water-jacketed to the top of the bosh, above which is a fire-brick shaft. The height of the charge is $17\frac{1}{2}$ feet above the tuyeres, and the blast used is 32 to 34 ounces. Smaller tuyeres set at 15-inch centers were substituted for ones set at 20-inch centers without much difference in results.

The bullion is tapped at the side of the furnace into a kettle of 50 tons capacity, from which it is pumped by a $1\frac{1}{2}$-inch centrifugal pump into vertical anode molds. Fig. 2 shows the kettle, molds, chain block with electric motor, and dross skimmer.

FIG. 5. INTERIOR OF TANK ROOM

Special frame cars holding 10 anodes are used, the cars going up the haulage incline to the level of the railway cars, in which they are transported to the refinery. The matte and slag run into a large receiver at the end of the furnace, the slag overflowing into the granulating launder, and the matte being tapped into pots.

The charge usually has about 85 per cent. of Huntington-Heberlein roast, varying considerably with the ores, and some oxidized and siliceous dry ore and lime rock. The coke is about $12\frac{1}{2}$ per cent. of the charge, not counting slag and easily smelted by-products. One hundred to 300 pounds of slag from the high-grade copper furnace is added to each charge to keep the charge open, the furnace action uniform, and reduce the percentage of zinc. The lead in the charge has been successfully run as high as 50 per cent., and averages about 40 per cent. Sulphur is kept below 4 per cent., the lower the better, as the matte fall is less. The speed of running decreases very markedly with a small increase in sulphur. About 30 to 40 per cent. of the sulphur in the charge is eliminated in the blast furnace.

The slag is kept at about 31 to 33 per cent. silica, 18 to 20 per cent. lime, 24 to 30 per cent. iron and manganese (FeO,MnO), 7 to 12 per cent. zinc (ZnO), and 8 to 16 per cent. alumina. It carries about .4 ounce silver and a trifle over 1 per cent. lead. With high zinc the lime is reduced to keep the slag fluid. It is not profitable to run siliceous ores to bring the slag much above 33 per cent. silica, as the furnace slows up too much. Alumina gives no serious trouble. The iron and manganese may vary between the percentages given without much difference in the running. Scrap iron is added occasionally, although it is not necessary when the charge is good, but is good medicine if things are not running smoothly.

One furnace is being run at present, producing about 75 tons of bullion per day. As high as 130 tons of bullion has in one day been made. An average analysis of the products is as follows: Copper, .22; manganese, nil; zinc, .098; antimony, .32; arsenic, .28; nickel, cobalt, and cadmium, nil; bismuth, .0133; lead, 98.5 per cent.; silver, 100 ounces; and gold, .1 ounce. Smelting campaigns last about 7 months, then if accretions have accumulated sufficiently to make it profitable to blow out and clean up, the alternate furnace is blown in.

The refinery is situated on flat land along the railway, about 200 yards from the main smelter buildings. Figs. 3 and 4 show the buildings from opposite ends. The long building on one side is the tank room, the slimes plant is in the center, and the bluestone plant on the opposite side. Fig. 5 shows the interior of the tank room with part of the new tanks in the foreground, and the melting pots and the molds in the middle distance. Fig. 6 shows the machine for making starting sheets with the corner of one sheet turned up from the cooling plate.

The general procedure in the refinery is as follows: The anodes are electrolyzed in tanks, producing refined lead and slime. The lead is melted, run into molds and shipped. The slime is treated in a separate plant with the production of fine gold and silver. Antimony, when commercial conditions warrant, can be saved from the slime by a special electrolytic process discovered by Mr. McNab. A by-product solution of copper sulphate is crystallized out in the bluestone plant, making a commercial product used by the farmers to kill the smut in wheat.

Three motors of 650 kilowatts total capacity are installed, which drive generators and furnish current at 3,500 amperes and 80 volts for the tanks. A smaller dynamo furnishes current for lighting and minor uses. Coal is used under the melting pots, evaporating kettles, and to furnish steam for heating, etc., but not for power purposes.

The tank room contains 240 tanks, 3 ft. × 8 ft. × $3\frac{1}{2}$ ft., made of coast fir, lined with asphalt, arranged in six double rows in a series of cascades. The electrolyte is kept in circulation by pumps. One hundred and eighty more tanks are in course of construction, which will raise the plant capacity from 75 to 100 tons per day and permit of lowering the current density. Each tank has a capacity of 20 anodes. An electric crane carries a spacing frame in which it picks up the anodes from two cars and lowers them to their place in the tank properly spaced. The anodes weigh 370 pounds each, and are cast

with shoulder lugs which rest on copper current bars on the sides of the tanks, the bodies of the anodes hanging down into the tanks. They are spaced $4\frac{1}{8}$ inches apart.

The cathodes are thin sheets of pure lead made on a special machine, Fig. 6, invented in the refinery. It consists of a tipping trough at the head of a $\frac{3}{4}$-inch plate of cast iron, steel surfaced, the size of the cathode, inclined at 1.8 inches per foot. The trough is filled with molten lead and tipped over the plate. The lead spreads uniformly over the plate and solidifies into a uniform sheet about $\frac{1}{16}$-inch thick, which is trimmed by running a knife around the edges, after which it is stripped off the plate. These sheets are hung on $\frac{1}{2}'' \times \frac{3}{4}''$ copper bars, bending the end of the sheet once around the bar. Twenty-one cathodes, spaced equally between and outside of the anodes, go in each tank. They are slightly larger than the latter to prevent short circuiting and are placed in the tanks by hand.

The electrolyte used is lead fluosilicate with free fluosilicic acid, averaging 12 per cent. acid (SiF_6) and 5 to 6 per cent. lead. This is stable with small losses. The acid is made in a special plant. Fluorspar, silica, and sulphuric acid are mixed in proper proportions in a cast-iron pan and the fumes condensed in towers with water sprays, the acidified spray water being used over and over until the strength is about 30 per cent. fluosilicic acid. Free sulphuric acid is removed by lead dross. The clean acid is added to the electrolyte as needed. About .5 to 1 pound of glue per ton of lead is added to the electrolyte daily, as without it the lead deposit would be soft and non-coherent and the tanks would short-circuit rapidly.

The tanks are operated with a current density of 16 amperes per square foot of cathode area. It is proposed to reduce this to 12 amperes, which experience shows will give better results. The voltage averages .32 volt per tank, the contacts causing a loss not over .02 volt per tank. A tank is worked out in about 8 days, 15 per cent. scrap going back to the melting pot after washing. Most of the slime adheres to the anode scrap, which is scraped and washed off in a special tank. Some slime settles in the electrolytic tanks, which are cleaned out once in a month. The cathodes are taken out, melted after washing, and cast into molds to suit the trade, the Chinese trade, for instance, requiring pigs of 190 pounds to suit the carrying capacity of the coolies.

An average analysis of 2,000 tons of shipping lead ran as follows: Arsenic and bismuth, nil; zinc, .0005; silver, .0013; copper, .00075; lead, 99.9938; iron, .00075; tin, .0001; antimony .0028. The average analysis of the slime is: Silver, 35 per cent.; antimony, 25 per cent.; arsenic, 20 per cent.; and copper, 8 per cent.; with some iron, bismuth, and silica, and traces of tellurium and selenium.

The slime from the tank room is taken in copper cars to the slime plant, where it is agitated in tanks with hot water. The wash water is afterwards drawn off and evaporated in steam coil tanks to recover the electrolyte contained. The slime is filtered and dried in large cars which are run into the furnace flue for the purpose. The slime is then melted in a water-jacketed reverberatory furnace lined with magnesite brick. The impurities are oxidized off and dore metal, running 960 to 975 parts gold and silver, is obtained. The fumes go through a series of inverted U cooling flues, which recover most of the volatilized metals, the condensed fume going to the blast furnaces.

The dore metal is parted in sulphuric acid. The silver sulphate is run into steam-heated tanks, where the silver is precipitated as a slime by bars of metallic copper. The silver and gold are collected, melted, and cast into bars. The silver bars weigh about 80 pounds each, and are 999 fine, the gold being 995 fine. The copper sulphate solution goes to the bluestone plant.

The McNab antimony process includes dissolving the antimony and arsenic from the slime by boiling with sodium polysulphide, and electrolyzing the solution with lead-sheet anodes and steel-sheet cathodes, the cathode deposit being a dense hard antimony, with 2 per cent. arsenic, which is removed by melting with alkaline fluxes.

In this case, the slime, after treatment with polysulphide, is roasted in a muffle furnace and treated with 10 per cent. sulphuric acid, which extracts 90 per cent. of the copper and 10 to 75 per cent. of the silver. This solution formed is heated with copper as before to precipitate the silver. The residual slime is then treated in the reverberatory as described.

Fine gold and silver are sold to the Canadian and American mints at Ottawa and Seattle. Fine silver is largely shipped to the Orient. Lead is sold in Eastern Canada and the Orient with smaller local markets.

Treatment rates vary widely on different classes of ores. The practice is to quote rates f. o. b. the smelter as far as is possible, making them independent of variable freight tariffs. Formerly inclusive freight and treatment rates were the rule.

The lowest treatment rate is \$1.50 per ton on low-grade, easily smelted, self-fluxing Boundary ores. This compares favorably with the widely advertised cheap costs of the Boundary smelters.

Rossland siliceous copper ores receive a \$3 rate, and other copper ores receive rates not far from this, depending on quantity and quality. Indirect deductions are usually about .4 per cent. from the copper wet assay, and 4 cents per pound from the New York price. The situation of the smelter making marketing costs comparatively high, 95 per cent. of the gold

Fig. 6. Casting Machine for Lead Cathodes

and silver is paid for at \$20 per ounce and New York price, respectively.

Lead sulphide ores are treated on a basis of \$8.50 to \$9.50 per ton for ore carrying 70 per cent. lead, adding 10 cents per unit for each per cent. under 70, to a maximum of \$10.50 to \$11.50. The lower are contract and the higher are open rates. Ninety per cent. of the lead fire-assay is paid for, if over 5 per cent., at London, England, prices, less 1 cent per pound marketing charge. Ninety-five per cent. of the gold and silver is paid for as above. Settlements are usually based on quotations 3 months from date of receipt of ore in the works.

Dry ores, iron concentrates, etc., receive a variety of rates, usually based on a \$3.50 to \$4.50 treatment rate with sliding scales. Excess iron is allowed for and excess silica is penalized at 7 cents per unit. Sulphur is penalized at 50 cents per unit. Zinc on all ores is penalized 50 cents per unit, for each unit over 8 per cent.

To give averages of variable smelting costs, without an extended analysis of conditions, would be of little benefit. It would require an article longer than the present one to do the subject justice and will not be attempted here.

The climate of Trail is moderately dry and mild, with snow for 3 months, and a warm summer with cool evenings. Operations are very seldom interfered with by the weather to a serious degree. A good class of labor is employed, many of the employes having been with the works for many years, and owning their own homes in the town. There has never been any serious labor trouble since the plant started. Some 550 men are employed, many of foreign extraction, but there are no Japanese, Chinese, nor Hindus.

SAFE USE OF ELECTRICITY IN GASEOUS MINES

Some years ago, when electric machinery was first being used in mines, the writer in conversation with the late Eckley B. Coxe, the great anthracite coal operator, said: "Mr. Coxe, in your experimenting and trying out of new methods and machines, have you done anything with electric machinery at any of your mines?" Mr. Coxe answered: "Electric machinery is all right for many purposes above ground, but I will not use it underground. While we have never had any gas at any of our mines, there is no telling when we will have some. We employ fire bosses, and daily examinations for gas are made. If I was sure that we never would have gas, I certainly would try out electrical machinery, but, as you know, where electrical machinery is used, there is always great liability to sparking, and an electric spark will ignite gas." Some two weeks later the writer met Mr. Coxe on a Lehigh Valley train, and he at once said: "Do you remember our talk about electrical mining machinery a couple of weeks ago? Well, I told you then we had no gas in any of our mines. Yesterday morning one of our fire bosses was quite severely burned while making his rounds. He was careless and did not use his safety lamp as he was instructed to, probably because he felt secure in the absence of gas on his many previous rounds. Now, you can see how reasonable my objections to electric machinery in coal mines are."

While the use of electrical mining machinery has become very common since then, it is frequently restricted by law to mines known as non-gaseous mines, and in Pennsylvania its use is positively forbidden "except where so insulated as to prevent the emission of sparks and flames."

Flame-proof types of electrical mining machines, which receive their power from insulated feed-lines, have been in use now for several years. But with such machines, and a well insulated feed-wire, the difficulty of making connections between the main feed-lines and the feed-wires of the mining machines still remained and was an element of danger. Now, however, this is removed by the use of the interlocking safety switch manufactured by the Interlocking Safety Switch Co., of Cleveland, Ohio.

This interlocking safety switch is an air-tight switch or junction box to be attached to the main feed-wires and fastened to the wall in butt entries at every second or third room opening. It consists of a cast-iron box which is filled with oil and the knives of the switch making and breaking the current in the oil makes it spark-proof. The plugs, which are fastened to the machine cables, are inserted in the switch nearest the room desired to be worked and are so arranged that when the current is turned on the plugs are locked in place and are absolutely immovable until the current is released, making it perfectly safe under the most dangerous conditions. It has been tested at the testing station of the United States Geological Survey at Pittsburg, Pa., and proved all that is claimed for it, and a prominent state mine inspector says, under date of March 1, 1910: "I think it is the best I have seen on the market." These switches have been in successful use at the Harwick Mine of the Allegheny Coal Co., in the Pittsburg region, for the past 2 years. The Manifold Mine of the Youghiogheny and Ohio Coal Co., the Pike Mine of the Peoples Coal Co., the mines of the Republic Iron and Steel Co., and the Ellsworth Collieries Co. are also equipped with them.

Many managers of gaseous mines have refused to consider the use of electric machines on account of the danger of sparks, notwithstanding they admitted the convenience and economy of such machines, and many managers of non-gaseous mines, held the same views as did Mr. Coxe, and for the same reason have not used such machines. This device has, with the production of flame- and spark-proof coal cutters, made safe the use of electric mining machines in both hard and soft-coal mines.

THE FRANZ SCREEN

Written for Mines and Minerals

The worst feature of those machines employing an endless screen for sizing mill pulp is the uneven wearing of the screen cloth. It is common experience that a certain portion of the screen cloth often wears through or breaks from bending, within a few days after its installation. The screen cloth must then be repaired by patching, and in course of time the number of patches becomes so great as to seriously interfere with the efficiency of the screen as a whole, necessitating the discarding of the screen belt.

The new Franz screen aims to eliminate the necessity of patching the screen cloth by presenting to the pulp flow a screening surface not consisting of a continuous belt of screen cloth, but of a series of screen sections, as shown in Fig. 1. When any of these screen sections becomes worn, that particular section may be immediately removed and quickly replaced by another. In this way the efficiency of the screening surface is at no time reduced by patches, and it will readily be seen that the expense of replacing a worn by a new screen section is considerably less than that of installing a whole new endless-belt screen. Fig. 1 plainly shows the method of construction. The series of sections are mounted upon endless rubber belts which pass over the supporting pulley wheels, the screen sections themselves being given a slight curvature, as shown in Fig. 2.

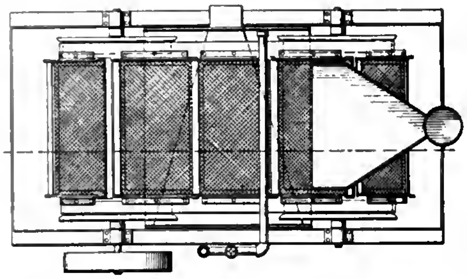


Fig. 1

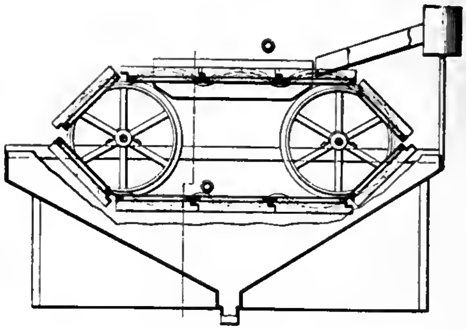


Fig. 2

The inventor of this machine is Mr. Frank Franz, master mechanic of the Hercules Mining Co., at Burke, Idaho, where several of these screens have been installed at the mill for more than a year with favorable results. In these machines the 30-mesh screen cloth on the machine treating about 75 tons per 24 hours lasts from 4 to 5 months, while the 12-mesh screen cloth on the machine treating 150 tons per 24 hours lasts from 3 to 4 months.

In addition to the machines treating this fine material there are two screens in use at the sorting plant where the ore from the mine is washed and sorted before going to the mill. The endless belts of these two machines are equipped with perforated metal plates, the perforations being 12 millimeters in diameter. At the time of writing, these plates have been in commission 7 months and still do not show any signs of wear, although the daily capacity is 300 tons each.

The production of Swedish pig iron in 1909 amounted to 443,000 tons, and the production of the steel and malleable iron was 150,000 tons. This is the smallest in many years. The export of iron and steel shows only a reduction of 17,000 tons.

THE HUBBELL ELECTRIC MINE LAMP

Written for Mines and Minerals

Ordinary incandescent electric lights are used in lighting mine roads, and that proposition is one which calls for little improvement. Such lamps are stationary, being supplied by electric currents from dynamos, so that, their use being limited to one place, the miner must, if he uses electric light at all in his travels and work, depend upon a storage-battery lamp. The main feature ot a portable electric mine lamp is the storage battery and its ability to hold a charge that may be relied on for a given period. The Hubbell storage battery has plates of silver and cadmium which are submerged in alkaline solution and then charged at the rate of 1 ampere for 8 hours, with a direct current of from 55 to 600 volts

FIG. 1

A simple rule for determining the positive from the negative wires, if no special pole-finding paper is at hand, is to place the two terminals about $\frac{1}{4}$-inch apart in a glass of water, when hydrogen gas will be given off from the cathode wire that is the one from which the current is leaving the water, and which is called the negative pole or terminal.

Electric hand lanterns have not been very successful in mining work, owing to the battery plates, which for a time would render good service, suddenly going wrong, particularly after a period of idleness.

Unlike the lead plates of ordinary storage batteries the silver and cadmium plates are said not to be injured by overcharging, by the reversal of the charging current, by periods of long idleness, or by complete exhaustion. The method of charging the lamp from an ordinary electric light wire is shown in Fig. 1. Next in importance to the battery is the lamp. This must have sturdy filaments in order to withstand the shocks that are incidental to mine use and which the best of care may not always avoid. The Hubbell electric lamp bulb has a tungsten filament designed and anchored in such a way as to resist ordinary shocks, but the makers have not stopped here, and have equipped their lamps with a safety emergency bulb which can be utilized in case of the service bulb becoming injured or burned out. The filaments are further protected by a combination switch and rheostat, which has a series of resistance coils that increases the life of the electric lamps, because the current is applied gradually without sudden overloading.

FIG. 2

The most valuable feature in connection with the electric lamp in mines is that it will not, unless broken suddenly, set fire to gas. To avoid such firing the lamp bulb under consideration is so supported that should the "bull's eye" be shattered, the bulb would leave the supports before crushing, and, becoming disconnected instantly, allow the filament in the lamp to cool.

Mr. James Hillhouse, Chief Mine Inspector of Alabama, was so impressed with this lamp when used by members of the United States Geological Survey after the Mulga and Palos mine disasters that he became anxious to know more concerning them, although supplied with another make of electric lamp.

The lamp shown in Fig. 2 is designed for general use about the mine and for rescue work; when supplied with a holder, as in Fig. 3, it makes an excellent headlight for a motor, or tail-light for a trip. These lamps utilize a four-volt battery, weigh 5.5 pounds, and have a lighting capacity of 12 hours. Another smaller lamp is made that weighs 3.25 pounds, has a two-volt battery, and a lighting capacity of 12 hours. The cost of operation of these lamps compared with gauze safety lamps is less than one-half; at the same time they are the safest safety lamps, and can be used with helmets where gauze safety lamps would blow up or go out.

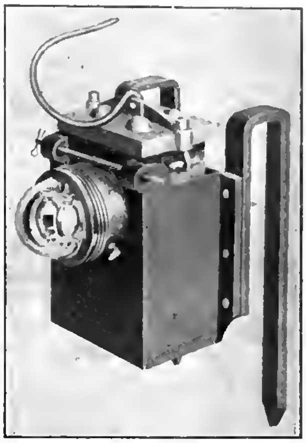


FIG. 3

A combination shot firer and lamp is shown in Fig. 4. This arrangement has a 6-volt battery of sufficient power to discharge five electric detonators simultaneously. By means of a red indicator light, the operator can from a position of safety ascertain whether he has an open or shut circuit, or a "hang fire," and is thus warned not to approach the blasting face.

The Hubbell lamps are manufactured by the Portable Electric Safety Light Co., Newark, N. J.

LUBRICATING BEARINGS OF HEATED ROLLERS

A certain manufacturer had an order for a machine that included three hollow gun-metal rollers, one weighing 1,386 pounds, the other two weighing 752 pounds apiece. These rollers are heated by gas to a temperature of about 700 degrees, and it was found that any oil or grease would carbonize and cut the journals in a very short while. In this predicament the use of graphite on the rollers was suggested. This was done, Dixon's flake graphite being used alone, and some months later the makers wrote the Joseph Dixon Crucible Co. that the scheme had met with perfect success.

FIG. 4

"The method of applying the graphite to the journals is very simple, the channels for conveying the lubricant to the journals are cut in the boxes about $\frac{3}{8}$-inch wide and $\frac{1}{4}$-inch deep, one on top and one at a little above and on each quarter. Besides this, a spiral groove of the same dimensions is cut for about two turns, commencing at about 1 inch from the other end of the box and near the bottom. These grooves are half round in section. Into the top straight groove, a $\frac{1}{2}$-inch pipe hole is drilled and tapped, a piece of $\frac{1}{2}$-inch pipe screwed into this with a reducing socket on the top end to $1\frac{1}{4}$-inch pipe. A $1\frac{1}{4}$-inch nipple with a $1\frac{1}{4}$-inch cap completes the cup. A piece of $\frac{3}{16}$-inch round C. R. steel with one end on the journal, with the other end up near the top of the cup, completes the device. The journals take about a dessert spoon full of graphite per day to each journal, the machine attendant occasionally removing the cap from the cup, churning down a little of the graphite with the $\frac{3}{16}$-inch rod." This experience may be valuable to readers of this matter in case they should meet a similar difficulty.

TRADE NOTICES

The branch managers and principal salesmen of the Taylor Instrument Companies held their annual "get-together" meeting July 11–13, inclusive, at the factory in Rochester, N. Y., reviewing general conditions in the thermometer industry and outlining new measures and policies for the coming year. Business for the past year has been most successful and although the present plant was occupied only in 1906, the growth has been so rapid as to necessitate material additions, to facilitate in the making and handling of "Tycos" products. A wood-working shop 100 ft. × 160 ft., a dry kiln, and a new wing being added to the main building, will make an addition of more than 40,000 square feet of floor space to the present plant.

The Wagner Electric Mfg. Co. announces the removal of its Charlotte, N. C., office to the Woodward Building, Birmingham, Ala., to be in charge of Mr. J. F. Jones.

The C. O. Bartlett & Snow Co., of Cleveland, Ohio, have leased the property recently vacated by the McMyler Mfg. Co. near their present works. It consists of buildings, yards, trackage, and complete power plant; and additional machinery is now being installed. The property is situated in the center of the city, and contains 76,000 square feet of floor space.

The Nelson Iron Works, of Nelson, B. C., has taken over the business and stock of the Rossland EngineeringWorks, of Rossland, B. C., and will succeed that company as British Columbia agents for the Sullivan Machinery Co., of Chicago. An increased stock of Sullivan rock drills, diamond drills, hammer drills, air compressors and supplies for these machines will be maintained at Nelson.

Factories, barns, etc., are probably more at the mercy of burning sparks and embers than other types of buildings, because they are usually covered with so-called ready roofings, and nearly all roofings of this type are made of wool felt, rag stock, paper, coal tar, and other highly inflammable materials. There is one roofing of this type, known as J-M Asbestos Roofing, which seems to overcome the objections to all others of this type. It is said to be so fireproof that it will withstand the flame of a blow torch for an hour without being injured. This roofing is made by the H. W. Johns-Manville Co., of New York, well known as manufacturers of asbestos products.

At the annual meeting of the stockholders of the Joseph Dixon Crucible Co., the old board, consisting of Geo. T. Smith, William Murray, William H. Corbin, Edward L. Young, Geo. E. Long, William H. Bumsted, and Harry Dailey, were unanimously reelected. The board of directors reelected the former officers; namely, Geo. T. Smith, president; William H. Corbin, vice-president; Geo. E. Long, treasurer; Harry Dailey, secretary; J. H. Schermerhorn, assistant treasurer and assistant secretary. William H. Corbin was also reelected as counsel.

The Jeffrey Mfg. Co., of Columbus, Ohio, has changed the location of its Denver office to rooms in the First National Bank Building. This company also maintains a corps of engineers at its branch offices in Chicago, St. Louis, Denver, Montreal, Pittsburg, Charleston, W. Va., Boston, New York, and Birmingham. There are also nearly 100 Jeffrey agencies in other cities in the United States and abroad.

The Western Electric Company reports that during the past twenty months it has sold over a quarter of a million of the No. 1317 type rural telephones. This is an unprecedented figure for these sales, and it is a certain indication that the rural telephone movement is progressing rapidly.

The Pan American Mining Company, which has offices in Philadelphia and mining properties in Dutch Guiana, S. A., is doubling the capacity of its concentrating plant. Allis-Chalmers Company furnished the Tremain steam stamps for the original plant and has orders for additional equipment.

NEW INVENTIONS

Complete specifications and drawings for any of the following patents can be obtained from the Commissioner of Patents, Washington, D. C., at the following rates.

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
At entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED JULY 5 TO JULY 26, 1910, INCLUSIVE

No. 963,611. Door for mines, quarries, elevator shafts, etc., Charles Matthews, Pittsburg, Pa.

No. 963,721. Ore cleaner, Alexander McDougall, Duluth, Minn.

No. 963,488. Ore-sizing apparatus, Charles Pierce Watterson, McGill, Nev.

No. 963,111. Process of treating precious metal-bearing materials, Paul W. Avery and Eugene C. Knowles, Deadwood, S. Dak.

Nos. 963,277 and 963,278. Ventilating apparatus, William Clifford, Jeannette, Pa.

No. 963,416. Extraction of zinc, Charles Skinner, Brand, Troon, Scotland.

No. 964,444. Centrifugal coal separator, Nelson Mowery, Wilkes-Barre, Pa.

No. 964,219. Waterproof fuse cap, Thomas M. Daniels, Chicago, Ill.

No. 963,787. Mine-ventilating system, David R. Martin Opelika, Ala.

No. 964,261. Ore classifier, Frank G. Janney, Salt Lake City, Utah.

No. 964,083. Ore concentrator, Lucien I. Blake, Boston, Mass.

No. 964,425. Ore concentrator, John F. Isbell, Salt Lake City, Utah.

No. 964,024. Ore-grinding machine, Charles R. Hotchkiss, Oakland, Cal.

No. 964,183. Rock crusher, De Witt C. Prescott, Chicago, Ill.

Nos. 964,429 and 964,430. Process of transferring material from a high to a lower level and piling the material, Arthur C. Johnston, Wyncote, Pa.

No. 964,733. Automatic car lock for mine cages, Julius H Alpenfels, Denver, Colo., and Harry A. Williams, Deadwood, S. Dak.

No. 964,635. Coke retort oven, Victor Dominique Fernand Fieschi, Douai, France.

No. 965,069. Process for making fuel briquets, Eugene Bonstein, Shickshinny, Pa.

No. 964,652. Hydraulic ore concentrator, John G. Kirksey, Milwaukee, Wis.

No. 964,567. Ore-separating machine, Aaron G. Seberg and Edwin G. Seberg, Racine, Wis.

No. 964,870. Treatment of ores, Charles Morris Johnson, Avalon, Pa.

No. 964,875. Rock drill, Harry Johan Hjalmar Nathorst, Gellivare Malmfalt, Malmberget, Sweden.

No. 965,054. Rock-drill extractor, William Edgar Weeks, Salt Lake City, Utah.

No. 964,605. Feeding mechanism for rock drills and the like, George R. Bennett, Denver, Colo.

No. 965,115. Rock-gathering machine, Charles C. Moore, Carthage, Mo.

No. 965,500. Miner's candlestick, Daniel B. Beaton, Cottonwood, B. C., Canada.

No. 965,808. Apparatus for testing drill holes, Matthias Garvey, Mineville, N. Y.

No. 965,760. Device for thawing explosives, George S. Sneer, Silverton, Col.

No. 965,830. Ore crusher, Isaac Lemon Mitchell, Cedar Rapids, Iowa.

No. 965,767. Fluid-distributing pipe for ore-treatment vats, Charles Edwin Draper Usher, Johannesburg, Transvaal.

No. 965,714. Process for extracting or separating precious values from ores, James S. Island, Toronto, Ontario, Canada.

No. 965,294. Jigging machine for dressing ores, Henry Richard Hancock, Burnside, South Australia, Australia.

No. 965,474. Rock drill, Edwin M. Mackie and Percival F Doyle, Franklin, Pa.

Mines and Minerals

Vol. XXXI—No. 3. SCRANTON, PA.—OCTOBER, 1910—DENVER, COLO. Price, 25 Cents

HYDRAULIC AIR COMPRESSION

Written for Mines and Minerals, by E. B. W.

It is now possible to supplement the article by C. H. Taylor in the April, 1910, issue on "The Cobalt Hydraulic Air Compressor," at Ragged Chutes, in the Montreal River, Canada. The principle on which the plant is constructed and the details of its construction having been fully described, it is the intention in this article to furnish information on this kind of air compressor which has not previously been given.

Further Particulars in Regard to The Hydraulic Air Compressor at Ragged Chutes

Hydraulic air compressors, being designed for the purpose of furnishing a predetermined quantity of air at a definite pressure per square inch, are calculated with the same degree of care which the mechanical engineer uses with a machine compressor. In the design of the hydraulic plant the engineer must include the quantity of water needed to entrain the desired number of cubic feet of air per minute; the diameter of the shafts; their different depths necessary to furnish the desired pressure and proper working conditions; the air chamber and its relative height with reference to the shafts, quantity of air compressed, blow-off and air-pipe connections; the diameter of the main pipe line and branches, relative to friction in the transmission of compressed air, and the consequent reduction in pressure.

There are matters which the engineer is unable to calculate, which nevertheless he knows will occur and must be anticipated in the design. He makes allowance for variations in the lengths of pipe due to differences in temperature; he attempts to regulate the shrinkage in the volume of air due to extreme cold weather; and the variations in the quantity of aqueous vapor in the air, etc.

In Fig. 1 is shown the spillway of the dam over which the surplus water flows. It is 660 feet long, constructed of concrete, and when it is remembered that the location is in the wilderness, where supplies must be delivered in the summer, the engineer at least will realize in a measure a few of the difficulties with which Mr. Taylor had to contend. The supplies were hauled on rafts up the river by gasoline boats, and some remarkable feats in transportation were performed in this way. The 20-inch pipe at the surface connecting with the air chamber 351 feet below ground is shown in Fig. 2. This is a seamless flanged steel pipe made in Germany, in 40-foot sections, and is extra strong, weighing 75 pounds per foot. No pipe line constructed without regard to the wide differences in temperature in Canada could survive a season, therefore concrete piles are constructed at intervals along the line to which the pipes are anchored, and between which expansion joints are placed. The valve on this 20-inch line which governs the flow of air is also in evidence in Fig. 2. The air flows through 9 miles of 20-inch diameter pipe from which there are two 12-inch diameter service mains, one of which circles the town of Cobalt, while the other goes out to the Nova Scotia Mine. At a point about 7 miles from the compressor, at the Waldman, another 12-inch pipe line branches to the Kerr Lake district. A 6-inch line also extends in the Gillis Limit from the Waldman to the Red Jacket on the other side of the main track of the

Fig. 1. Spillway and Dam at Ragged Chutes

Fig. 2. Air Pipe Line from Compressor

T. & N. O Railway. The tendency of aqueous vapor to saturate air during extremely cold weather is sometimes an aggravation to the users of compressed air, and to obviate this difficulty Mr. Taylor has altered the head pipes, with the result that there is practically no moisture shown at the exhaust of the machines, and hence the danger of freezing in winter is remote. In the first few days the air was turned on, it shut down 15 steam-driven compressors, and more will undoubtedly follow, as one

Fig. 3. Blow-Off from Compressor

mine manager finds that his power bill per month has been reduced $2,300. Mention was made of the blow-off when air accumulates above a certain pressure in the air chamber. In Fig. 3 this safety arrangement is shown, spouting water to a height of probably 150 feet, although under full pressure it spouts as high again. Another view of the valves at the junction of the discharge with the main air pipe is shown in Fig. 4. The concrete piers to which the pipes are anchored from time to time are constructed about on the plan shown. The blow-off is seen spouting to the rear of Mr. Taylor, the designer. Previous to the Ragged Chutes plant several other hydraulic compressors were designed and put in operation.

The first plant installed by Mr. Taylor was at Magog, Quebec, when in 1895 a 155-horsepower hydraulic power plant was built to supply power for the textile business.

The second was at Ainsworth, British Columbia, 2 years later, when a 550-horsepower plant was installed to supply hydraulic-compressed air for mining purposes. Then, in 1899, at Peterboro, Ontario, for the Dominion Government, Mr. Taylor erected the lift-lock dam, at which 45 horsepower of hydraulic-compressed air was used.

Two years later, at the Victoria mines in Michigan, Mr. Taylor installed what was his most ambitious and greatest undertaking in the hydraulic line until he came to Cobalt to harness the Montreal River. At Victoria a 500-horsepower hydraulic plant was put in operation and has worked with splendid success from the day that the water was first turned into the gigantic tunnel.

On a visit to Montreal over 4 years ago Mr. Taylor heard of Cobalt and after investigating the mining possibilities there and the probable need of power, he was convinced that Cobalt's mining proposition would need a large power plant, and he only waited until the development of the mining industry here had demonstrated the need for power. The Cobalt plant develops 5,500 horsepower and with a reserve capacity available will actually supply [illegible] 1,000 horsepower more than this amount if necessary.

The pressure of 120 pounds per square inch is so high it is not practicable to turn on the full force of the air at the mines until a reducing valve has been put in. The force of the air was shown when the different sections of the 20-inch main were "blown out." The valves were opened and a piece of boiler plate fixed in the opening, when all sorts of articles such as horseshoes, pieces of scrap iron, monkey wrenches, etc., hit the boiler plate with the force of bullets, cutting deep gashes.

The one apparent defect in the air delivered from the hydraulic compressed-air plant is that whenever the air is turned off underground the candles go out.

In regard to the difficulty experienced in keeping alight candles in the air supplied by the Cobalt hydraulic air compressor, a letter has been received from Mr. Geo. Hooper, Superintendent of the Victoria Copper Mining Co., Victoria, Mich., where all drills are run by air supplied by the Taylor system. The letter was written to Mr. Hugh Park, manager of the Nipissing, who had requested information on the subject. The following is an extract:

"I may say that since we have been using this air we have never had a man affected in any way by it. When we were using mechanically compressed air we were bothered continually by men being 'knocked out.' We have been working a stope during the past year 2,000 feet deep and 1,000 feet from the shaft, and stope is 100 feet high and 500 feet below ventilation, yet we have absolutely no complaint from the men on account of the air. The writer put in 5 hours on the third level without the least difficulty. Our pressure is 116 pounds per square inch and air conditions must be about the same. We have drifts and cross-cuts that are breasted over 2,000 feet from the shaft, so you can readily see why I say that your men must draw on their imagination.

"The only complaint we have to make is on the light question. I trust the 'Sunshine' will particularly relieve your difficulty and if your men receive any injury from the air you will be having a different experience from what we have had and I do not know where you can find a mine as deep as ours without ventilation."

An analysis of the air from the compressor made at Mon-

Fig. 4. Air-Pipe Valves and Anchorage

treal showed 17.7 of oxygen against 20.6 and 20.8 in the ordinary air of the laboratory in which the test was made.

An analysis made by Dr. J. T. Dunald, of Montreal, Official Analyst of the Dominion Government, of samples of compressed air marked "Buffalo Mine," received from Cobalt Hydraulic Power Co., May 30, 1910, gave oxygen, 17.7 per cent.; carbon dioxide, traces; nitrogen, 82.3; total, 100 per cent.

Various devices are being tried in place of candles. An

oil called "Sunshine" is being used at the Coniagas in lamps with two burners specially designed, and acetylene lamps are also popular. No difficulty is experienced in mines that are well ventilated and the trouble with the lights is only felt in long drifts and cross-cuts and in the top of stopes.

The theory of hydraulic compressors was made the subject of a paper before the recent International Mining Congress, from which the following is abstracted. The author is P. Bernstein, of Cologne:

"Water is passed from some source at level L_1 through a nozzle, which carries air, into a fall pipe and down into an air chamber, in which, at level L_2, the air separates from the water into the air chamber to escape through a special pipe. The water rises in the air chamber in which the fall pipe is placed, to the level L_3, at which point it is discharged. If the whole level difference is $H=L_1-L_2$, $p=L_3-L_2$, and $h=L_1-L_2$, so that $H=h+p$, then p is the pressure under which the air column stands. For a given head H, the head h and pressure p can be varied, and the pressure in general is not dependent upon the head of water; a large pressure could be obtained with a small head." The paper described experiments conducted by Professor Gutermuth, and the author, at Darmstadt, and also installations made by the Wasserkraft-Druckluft Syndikat, of Mulheim-on-the-Ruhr.

The nozzle used in these experiments consisted of a funnel surrounded by a ring pipe connected with the funnel by three rows of small radial tubes, and the air was taken up through small vertical tubes rising from the ring. The efficiencies were calculated for isothermal and for adiabatic expansion; the latter efficiency sometimes exceeded 90 per cent. Other experiments were made in installations of the firm in the Harz Mountains and elsewhere; in one of these—in the Victor colliery, near Rauxel—the discharged water had to be pumped up again. The air was free from oil and dust, and less wet than the air from mechanical compressors, because the compression took place under continuous natural cooling; and the author showed by comparative estimates that these simple compressors worked far more economically than plunger compressors driven by turbines, and compressors driven by electric motors in hydro-electric plants.

RECLAIMING ZINC-LEAD FINES

Written for Mines and Minerals, by Lucius L. Wittich

Sludge from a zinc and lead concentrating plant of 150 tons capacity per shift, containing less than 1½ per cent. of zinc blende and no galena, is treated successfully on a miniature jig operated by the inventor, L. B. Hunter, who has his remarkably small contrivance located near a roadway on the C.W. Squires land in the Galena, Kans., district.

While only a working model, the undersized jigging apparatus has a record of cleaning 200 pounds of concentrates in 8 hours time, and Hunter says he could make a comfortable income from handling dirt running 8 or 10 per cent. in ore. Candy and tobacco buckets are used as ore bins, the illustration showing that the bins are almost as large as the jig. The working principle of the Hunter model is different from anything else in the district, excepting the old style hand jig. In this model the principle of gravity jigging is obtained by an upward and downward movement of the cells, not the water, as is the case in the steam jigs of the district. In the steam jigs the movement of the water is the result of the action of plungers or pulsators; in the Hunter jig a similar effect is obtained, the inventor claiming, however, that his jig can be operated with less motive power, the cells being balanced. A full-sized jig, he states, can be operated with ease by a person turning a crank. In the illustration the crank is seen at the left-hand corner of the jig. As no effort has been made to manufacture the jig on a large scale the inventor is contenting himself with treating the slimes and sludge on the Squires land, selling an occasional bucket of concentrates and affording boundless entertainment for the throngs that visit the tiny plant which is considered the most unique mining venture in the district.

An assay of the concentrates of the Hunter jig showed zinc blende 54.80 per cent. with 1.5 per cent. of iron. Considering that the Galena camp blende at its best rarely exceeds 58 per cent., the percentage of recovery from the sludge is considered a rather remarkable showing.

The miniature jig consists of 4 cells, two on each side of a stationary shaft. No matter how great the load on the screens the cells are evenly balanced at all times and the power required to operate them, therefore, is reduced to a minimum. Cross-bars, balanced on the stationary shaft, extend to a revolving shaft on which are attached eccentrics. The cells are suspended from these cross-bars, while the vertical movement is obtained by the eccentrics on the revolving shaft, an alternate motion being given as the crank is turned. The two rougher cells are located at the upper end of the jig, the cleaner cells at the lower end. An equal amount of dirt is supplied to each set of cells. Each cell is 16 inches long by 10 inches wide, with $\frac{15}{16}$-inch strips of sheet iron set vertically across the screen, which is of one-millimeter mesh. The cross-strips, which are 4 inches apart,

Hunter Jig, Galena, Kan.

hold the mundic, or iron ore, bedding in position while the concentrates sink through to the hutch, from where they are drawn off. As many cells as desired may be used. The inventor says four are sufficient for ordinary purposes.

Only dirt that will pass through a one-millimeter mesh is used on the jig as it now is operated, although on a larger model much coarser material might be handled. Passing through the fine-mesh screen the dirt is fed on to the cells through a sluice. There also is a water feed from a launder above, leading from a barrel, to be regulated as required. The rise and fall of the cells when in operation is ¼ inch; the speed of operation being quickened or retarded at will. The screens in the cells are detachable and may be changed at a moment's notice. The flow of water from cell to cell is protected by canvas connections. The overflow is discarded as tailing.

The jig tanks, which may be of any size or shape, are held in position by 2″×2″ timbers.

TIN MINING IN CHINA

Consul George E. Chamberlin, of Swatow, learns that a company with a capital of 2,000,000 taels ($1,146,000) has been formed in Yunnan, China, to develop the tin mines of that province. Under the old system some 360 tons of ore has been mined annually, but with machinery it is expected that 30 to 40 tons of ore per day can be taken out.

PUMPING AT BISBEE, ARIZONA

*Written for Mines and Minerals, by C. C. Austin**

The development of mine pumping within recent years has brought about some interesting engineering problems. It must be remembered that in all underground work the engineer is hampered by close quarters. Much of the machinery must be specially designed to meet the conditions; there must be no part of any machine too large to go down the shaft and through the drifts to its place. Arrangements must be made to keep the pumps running continuously, as the flow of water into a mine is continuous and would soon overflow the workings, if not disposed of.

The Junction Plant of the Superior and Pittsburg Copper Co.

One of the most notable undertakings in this line of engineering is the installation at the Junction shaft of the Superior and Pittsburg Copper Co., at Bisbee, Ariz., of one of the largest high-duty mine pumping stations in the world. It has a normal capacity of 7,200,000 gallons in 24 hours against 1,000 feet head, with provisions for 2,160,000 gallons additional when needed. The station, piping, and accessories were designed and erected under the supervision of Mr. W. E. McKee, the company's superintendent of machinery.

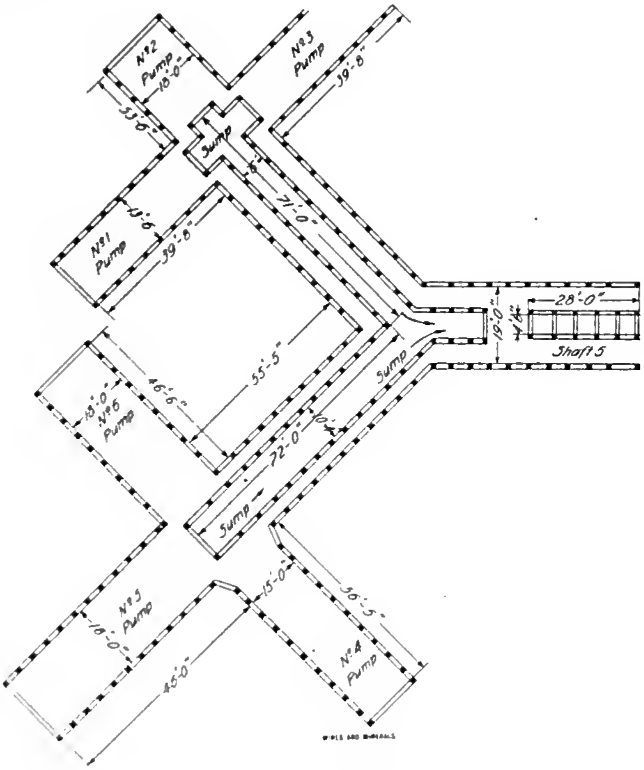


Fig. 1. Plan of Pumping Station, Junction Shaft

The plant was designed to pump the water from the Briggs, Hoatson, and Junction mines. The lower head against which this water may be pumped at the Junction and the economy in the operation of one large station instead of several small ones led to the construction of the large Junction station. From the Briggs and Hoatson, the water runs by gravity in a flume through the drifts to the Junction, where it is pumped to the surface. The collar of the Junction shaft is 124 feet below that of the Briggs and 150 feet below that of the Hoatson; thus the economy of this common station can be readily seen.

Fig. 1 shows the plan of the station, the peculiar arrangement of the pumps being used because it facilitates the superintendence of their operation. Pumps Nos. 1, 2, and 3 are direct-acting pumps which have been working since April, 1907, while Nos. 4 and 6 are crank and flywheel pumps which were erected in September, 1907. No. 5 will be installed when needed, furnishing the additional 2,160,000 gallons capacity mentioned before. The drifts lead from the main entry at 45 degrees in order to minimize the loss resulting from right-angled turns in the water columns. The sump extends from a point 10½ feet from the shaft to where it divides, one branch going to each set of pumps. It is 7 feet deep and holds 65,000 gallons.

Pumps Nos. 1, 2, and 3 are triple expansion and direct acting, with 15-, 23-, and 39-inch steam cylinders, 10-inch water cylinders, and a 24-inch stroke. Pumps Nos. 4 and 6 are of the crank and flywheel, cross-compound, Corliss-engine type with 20- and 38-inch steam cylinders, 6½-inch water cylinders and a 36-inch stroke. The duty of the direct-acting pumps is 80,000,000 foot-pounds, while 100,000,000 foot-pounds duty is obtained from the crank and flywheel pumps. Pumps Nos. 1, 2, and 3 are jacketed on the intermediate and low-pressure cylinders, but the low-pressure cylinders of the crank and flywheel pumps have jacketed heads only, because with jacketed cylinders, they could not be taken down the shaft. Each unit has a capacity of 1,000 gallons per minute, the direct-acting pumps running at 31 revolutions per minute and the crank and flywheel pumps at 48½ revolutions per minute. Each pump has a 25-per-cent. overload capacity, making a total available capacity of 6,250 gallons per minute, and for a short time, these pumps could all be run at 50-per-cent. overload. This heavy overload capacity is necessary in mining work as a heavy flow of water may be encountered at any time.

The steam piping of this station differs from that ordinarily used. In general practice the steam pipes are placed in the shaft and along the roof of the levels to the pumps; with this arrangement at the Junction, however, the large number of steam pipes required would make the station almost unbearably hot. To avoid this the steam pipe was placed in the shaft to a point 50 feet above the station. Drifts were then driven from here to points directly over pumps Nos. 1, 2, 3, and 4, and duplicate steam lines were run down to each throttle valve. Besides disposing of the heat on the station, which it does admirably, this plan creates a circulation of air through these drifts and raises, and keeps the air fresh.

On all the pumps jet condensers are used. The crank and flywheel pumps run their own condensers, which are 11 in. × 36 in. while the other condensers are independent of the pumps. The independent condensers are 12" and 20" by 18" and run at about 25 revolutions per minute, providing a vacuum of 20 to 22 inches when the barometer on the station stands at 25 inches. The condenser injection pipes are made about 18 inches shorter than the pump suction pipes. Then, if for any reason, the water in the sump should get low, the condensers will take air and the added back pressure on the exhaust will stop or slow down the pumps before any damage is done.

The system of steam piping used on these condensers is unusual. Instead of exhausting into the vacuum they exhaust into the low-pressure steam chests. This is not done to increase the economy of the pump, although it does so slightly, but to prevent excessive wear on the water valves and springs of the condenser. The pump cylinder never fills completely with water, and with steam pressure on one side of the piston and a vacuum on the other, the water piston moves back very quickly and strikes the water with sufficient force to wear out the water valves and springs rapidly. When exhausting into the low-pressure steam chests, however, the back pressure on the steam piston prevents this shock and considerably lengthens the life of the springs and valves.

Some features of the water columns are unique. Before installing the new pumps, two 8-inch pipe lines reached from an abandoned pump station on the 770-foot level to the surface. It was decided to extend them down to the new 1,000-foot level, join them at the bottom, and use them as one of the three necessary water columns. This junction was made by means

*629 N. Russell St., W. Lafayette, Ind.

of a peculiarly shaped special fitting which joins the columns, supports them, and receives the flow from a 12-inch pipe coming directly from one of the pumps. Twelve-inch pipe lines were then laid from pumps Nos. 1, 2, and 3 to the shaft, one emptying into the fitting mentioned above, as shown in Fig. 2, the other two extending to the surface. Each of these 12-inch column pipes had to receive the discharge from one other pump as there is not room in the shaft for a separate column from each. These lines had to join the first without tees to avoid the loss of head resulting from right-angled turns. Forty-five-degree **Y**'s with the branches turned down at 65 degrees and 32 minutes, shown in sectional view, in Fig. 2, and a 45-degree fitting bolted to each branch was put into each line. This arrangement allowed the pipes to be laid to Nos. 4, 5, and 6, and avoid sharp turns. The complete system of water columns and valves, shown in Fig. 2, will make plain how any column can be cut out and drained for repairs while the discharge continues through the other two. Check-valves are put into the discharge of each pump to obviate the necessity of opening and closing the 12-inch hydraulic valves near the roof.

No arrangement of valves or piping needs to be made on pumps Nos. 1, 2, and 3 for expansion, as the water ends of the pumps are anchored and the steam ends are on rollers. Owing to the shape of the crank and flywheel pumps, the steam ends are anchored and the provision for expansion, which is about ⅜ of an inch, is made at the water end. When pump No. 5 is installed, the valves and fittings for pumps Nos. 4, 5, and 6 will be arranged the same as for the direct-acting pumps in order to avoid the use of a cross at the junction with the water columns.

Hydraulic fittings tested to 2,000 pounds pressure are used throughout, while the extra heavy pipe is used from the pumps to a point 550 feet up the shaft, and standard pipe for the remaining 450 feet.

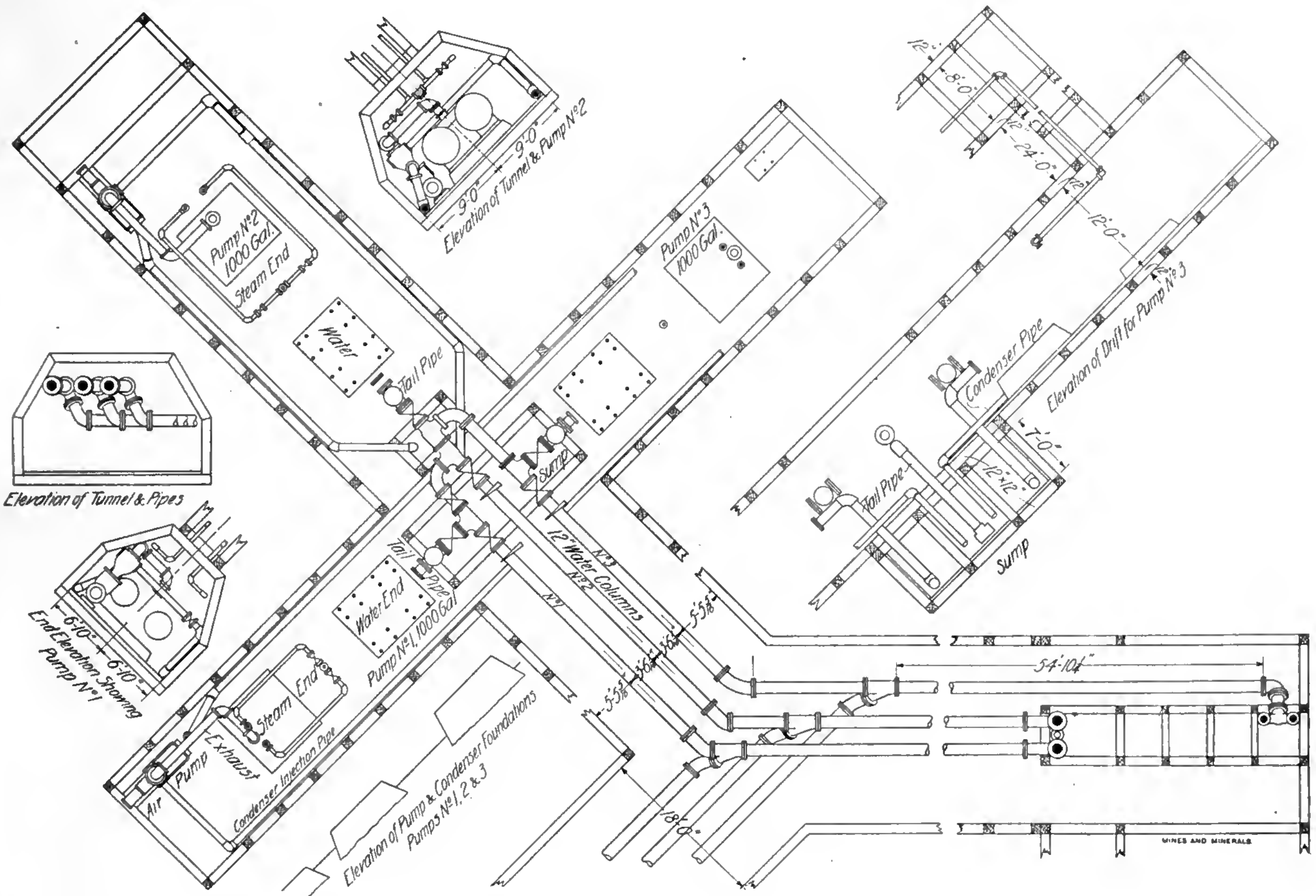


Fig. 2. Plan of Piping, Junction Shaft

While no unusual arrangement is used for supporting the water-column pipes in the shaft, the expansion and contraction of the steam pipes has required the installation of expansion joints. The latter are similar to those in use at the properties of the Oliver Iron Mining Co., of Duluth, Minn., and manufactured by the Imperial Iron Works, of Duluth. Fig. 3 shows the detail of the joint and the method of supporting it from the shaft timbers. We are indebted to Mr. H. J. Wessinger, chief engineer of the Oliver company for following data and illustration: "A joint is placed with the fixed end up so that each expansion joint supports and takes care of the expansion for the 200 feet of pipe above it. The first joint is placed 10 feet from the shaft collar to reduce the distance to the horizontal line leading to the boiler house. This arrangement is the fruit of experience in several bad breakdowns caused by omitting this joint and making the second joint do the work. The pipe expands 4 inches to 6 inches, causing the horizontal pipe to leave its supports. If blocking is then put under the pipe while it is hot there is no chance for the pipe to contract when steam is shut off, and

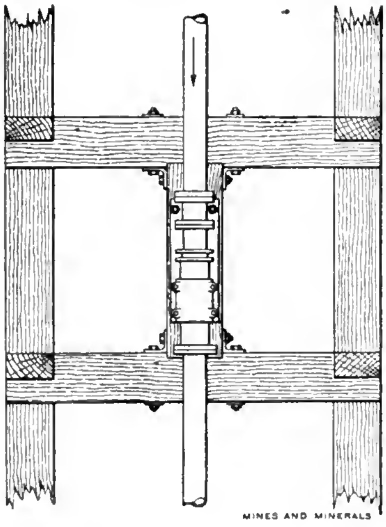


Fig. 3. Expansion Joint

it breaks usually causing the pumps to drown. By placing the top joint near the collar, the expansion to the horizontal line is reduced to less than ½ inch, making it impossible to block the pipe so that it will break when steam is shut off."

The station has been in complete operation since about the first of January, 1908, and is being watched with interest by all mining men who are confronted with the problem of handling large quantities of water.

OIL NOTES

State Mineralogist Lewis E. Aubury has, through the Mining Bureau of California, commenced to use the "Big Stick" on illegitimate oil companies, or those who strike oil by selling stock. Concerning this matter, he says: "We are determined that investors in California mines and oil wells shall be protected from the parasites which attach themselves to the industry. With this end in view, we propose to give the fakers all the publicity possible, and to cooperate with persons who have purchased stock in these companies, and to punish the offenders."

Mr. Aubury wishes to remind the purchasers of stock in these fake companies that he will not permit the State Mining Bureau to be made a collection agency—after they become aware that they have been "fleeced." The average person who has been duped is interested in only one phase of the question—the return of his money—not in the criminal prosecution of the faker.

"There is not one chance in a million," said Mr. Aubury, "of recovering a cent from these fakers, and there is only the satisfaction of placing them behind the bars, but this course remains for the person who has been swindled, and is one which is hardly ever adopted."

He calls attention to the Illinois Oil Bond Co., Chicago, the Paxton Gold Bond Co., Paxton, Ill., and the Wisconsin Gold Bond Oil Co. These were Arizona corporations, and were advertised as non-assessable, though it is well known that in California any corporation, foreign or domestic, may be assessed, and as a matter of fact, these, as well as the later corporations, were assessed out of existence. The Gold Bond part of the name refers to 3-per-cent. 30-year guarantee bonds which were offered to stockholders, but whether any of these bonds were called for, and if so what has become of them, cannot be determined. One of the companies supposed to issue these bonds cannot now be located; the other refuses any information.

"The Illinois Oil Bond Co. drilled a shallow hole said to be less than 300 feet deep and abandoned operations in this state."

"The Wisconsin drilled a hole to a depth of 1,335 feet and got a little heavy oil. This well was abandoned."

"The Paxton drilled a hole 1,500 feet, getting a trace of oil at 500 feet, but nothing below. This well was able to produce a few barrels of oil at considerable intervals."

Oil mining is about as uncertain as any other kind of mining, but like gold mining there are possibilities if one is fortunate. Where a person's investment is employed to develop the venture there is nothing to be said against promotion, but where there is intentional fraud and grafting they should be dealt with severely.

Trinidad Oil Fields.—Consul Franklin D. Hale furnishes the following information concerning oil seeking in the West Indian island of Trinidad:

"Oil men from England and the United States are working in the interest of their respective employers. The southern half of the island is being investigated, and all reports therefrom are favorable for a field of from 500 to 800 square miles of oil-bearing territory, some of the oil experts say that the field in all probability will extend all the way under the Gulf of Paria and westward to the region of the 1,200-acre pitch lake at Guanaco, which is operated by an American company, and still further westward to the Gulf of Maracaibo Should these surmises prove true, this is undoubtedly one of the largest oil fields in the world.

"A $1,500,000 company, floated in London, has taken over the Canadian company's 25,000-acre oil property in the southwestern section of the island, and a dozen wells, 800 to 1,400 feet in depth, have been bored with satisfactory results, some being excellent producers, although efforts are being made to check their flow until facilities for exportation are completed.

"An American company, which has long operated the pitch lake, has been exploring for oil for some time, and is constantly extending its operations. It has a number of wells at depths of 900 to 1,400 feet, and although it has two steel tanks of 84,000 gallons capacity each and land reservoirs of great storage proportions, many thousands of gallons of crude oil have run into the sea, which is only a few hundred feet from the field of operation.

"Many other companies are organized and being organized to develop the oil fields, and private capitalists are securing concessions and buying property and mining rights for prospective development or on speculation. A company located near the city of San Fernando has not only fair oil prospects, but has discovered coal of excellent quality in unknown quantity outcropping in its property, samples of which have been sent to England for examination and test.

"Exploration work is in prospect everywhere in the island; wells are being bored in new districts, while in the old districts everything is being rapidly enlarged. Under such circumstances reports of all sorts are in circulation, such as oil being struck a few feet from the surface in many places, etc. Regulations for acquiring rights in the oil fields and governing the prospecting for oil, transmitted by Consul Hale, are on file in the Bureau of Manufactures."

It is estimated that $210,000,000 are invested in productive oil properties in California, and to date $31,500,000 have been paid in dividends. The greatest oil well probably in the world is in the Midway field and is known as the Lake View. Its first day's production was 47,000 barrels, and it is thought to have attained a maximum flow of 90,000 barrels subsequently for a few days; at the end of 4 months it flowed 25,000 barrels per day. It was drilled to a depth of 2,265 feet. It is estimated that the output of California oil will be 80,000,000 barrels in 1910, making that state the largest producer of oil. At present the production is about 80,000 barrels more than the demand.

The steamboats Yale and Harvard, running between New York and Boston, are using oil as fuel. It is thought that the change from coal to oil, with the corresponding decrease in labor, will save $5,000 monthly.

Petroleum in New York State.—The oil district in the southwestern part of the state continues to afford a fairly large yield, though it has long since passed the high mark of productivity. The pools of Cattaraugus County were first tapped in 1865, and those in Allegany County about 1878, since which times they have been actively exploited. Many of the original wells that were drilled over 25 years ago still give a profitable return for pumping. No important discoveries have been reported in ecent years, yet by redrilling of territory once abandoned as worthless and by gradually extending the bounds of the known pools the natural decline has been checked, so that a long career of activity may be confidently expected for the future.

The productive area in Cattaraugus County is situated principally in Olean, Allegany, and Carrolton townships, embracing about 40 square miles. The oil is found at depths ranging from 600 to 1,800 feet. The larger pools are the Ricebrook, Chipmunk, Allegany, and Flagstone. They occur in the Chemung formation of the Upper Devonian.

In Allegany County are the Bolivar, Richburg, Andover, and Wirt pools, which extend across the southern townships and are tapped by wells averaging from 1,400 to 1,800 feet deep. The Andover pool lies partly in the town of West Union, Steuben County. A recent estimate placed the number of productive wells in Allegany County at 6,000.—*New York Museum Bulletin 142.*

TUNNELING ON LOS ANGELES AQUEDUCT

Written for Mines and Minerals, by R. L. Herrick

Conditions Found and Methods Used in Breaking All American Tunneling Records

In recording the great engineering feats of the twentieth century, the historian will certainly mention the Los Angeles aqueduct. Its claim to greatness will be based not so much upon the complexity of the engineering problems as upon the magnitude of the work and upon the speed and economy with which it was conducted. For a proper conception of its magnitude, nothing short of a trip over its 217 miles of length will suffice; but a fair idea may be obtained by reference to the map, Fig. 1. Other great enterprises of the future may be as successfully terminated, but thus far in the history of American engineering, the Los Angeles aqueduct stands without a peer among the various enterprises of similar character. This holds true, not only of those portions of the work involving mining problems, but also of the greater portion of the aqueduct involving civil engineering.

In pushing the work on more than 100 tunnels, aggregating a total length of about 28 miles, the best previous world's record for driving in soft rock has been exceeded, while the hard-rock record for the United States has been six times exceeded, and a new record established on each occasion.

The aqueduct when finished will deliver an ample water supply to the city of Los Angeles sufficient for the future, even should it attain to more than a million population; the surplus water will be used for irrigation and the proceeds will go to the city treasury. The water will generate hydroelectric power at three separate plants for transmission to the city and surrounding territory. The proceeds from the sale of this power will likewise go to the city treasury, so that with the income from the sale of water and power, the aqueduct is expected, not only to pay for itself within a few years, but to pay all expenses of the city government thereafter, thus relieving its inhabitants of all except nominal taxation.

The difference in elevation of the aqueduct line from the ntake at the Owens River to the city at approximately sea level, is 3,800 feet.

While the total length of the aqueduct as first surveyed was 238 miles, this original location has been several times changed in order to shorten the line and cheapen construction. The third annual report of the Bureau of the Los Angeles aqueduct gives the aggregate length as follows:

	Feet	Miles
Unlined canal	111,200	21.06
Lined canal	803,711	152.25
Tunnels in rock	91,019	17.24
Tunnels in earth	58,508	11.08
Siphons crossing cañon	74,847	14.19
Flumes	9,050	1.72
Total	1,148,335	217.56

Minor changes have been made in the line since this report, but the above may be taken as approximately correct.

Those familiar with the region traversed by the aqueduct know that the country to the east of the Coast Range is a desert; however, to others the accompanying views will doubtless prove interesting. In approximately level stretches there is sandy desert, where the heat is excessive, the alkali and dust-laden winds almost intolerable, good water scarce, and transportation very difficult. The same things may be said in general of those mountainous portions of the line traversed by the tunnels, with the addition that such portions are doubly inaccessible, so that road building has been no small item chargeable to the expense of tunnel driving.

West of the Coast Range, however, the character of the country traversed is the direct antithesis, much of it being under cultivation and favored by a plentiful rainfall.

The work has been divided into 11 divisions, each with its conveniently located construction camp, in charge of a division

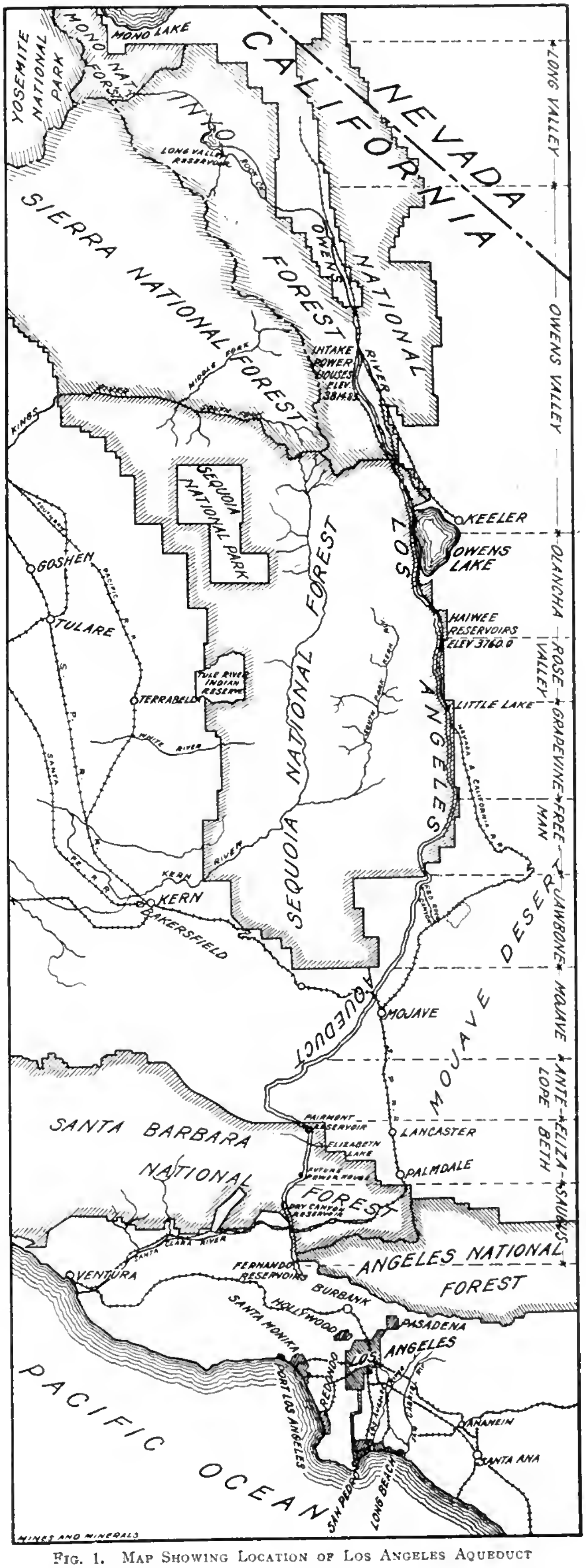


Fig. 1. Map Showing Location of Los Angeles Aqueduct

engineer. A railway has been built to parallel the aqueduct as closely as allowed by the contour of the country, and a complete telephone system connects all the camps and the main city office. Electric power transmission lines have also been built in to many of the camps, furnishing power for lighting and machinery. A pipe-line system likewise parallels the aqueduct, supplying water for the concrete work and domestic purposes. The cement for the concrete is supplied by three cement plants,

Fig. 2. Interior of Jawbone Tunnel

likewise constructed by the aqueduct commission, and they are sintering and grinding tuffs which outcrop at points adjacent to the plants. Four divisions of the aqueduct contain the majority of the tunnels. These, commencing with the most western, are the Saugus, the Elizabeth, the Jawbone, and the Grapevine.

Table 1 furnishes an idea of the magnitude of tunnel work.

Table 1

Division	Number Tunnels	Aggregate Length of Tunnels Feet	Longest Single Tunnel Feet	Kinds of Formation Penetrated	Boring Equipment	Status to Date
Saugus	30	54,040	San Fernando, 9,100	Compact sand and gravel formation free from water; also soft shale	Hand augers and hard drills	Completed
Elizabeth	2	26,860	Elizabeth Lakes, 26,860	Granite, schist, and gneiss	Machine drills	19,492 feet up to April, 1910, inclusive
Jawbone		59,448		Sand and gravel; cement; some granite largely decomposed	Hand work	Completed
Grapevine		40,350		Hard granite	Machine drills	

The tunnels were driven in hard rock only in the Elizabeth and the Grapevine divisions; in the others the formation penetrated was for the most part a sand and gravel cement with comparatively small extents of soft decomposed granite and soft black oil shales.

In soft-rock tunnels the world's best previous record for speedy driving, namely, 842 feet in 31 days, established at the Gunnison tunnel, has been exceeded.

The new record is 1,061.6 feet, established in the north portal of tunnel No. 17, Jawbone division, for the month of August, 1909, T. J. Flanigan, superintendent, and A. C. Hansen, division engineer in charge. An abstract of the official report in regard to this work shows there were 91 shifts with 9 men on a shift, and that these advanced the excavation 11.7 feet per shift through dry cemented sand and sandstone. The work was done by hand, using 4,450 pounds of 40-per-cent. dynamite and 2,372 pounds of black powder. The cost in dollars of the operation is given in Table 2.

Table 2

Classification	Labor	Animals	Supplies	Freight	Cost Per Linear Foot
Excavation.....	5,338.80	259.2	989.27	197.85	6.39
Timbering......			6.53*	1.30*	
Engineering	90.33				.06
Totals........	5,429.13	259.2	995.80	199.15	6.45

* Timbers under air shaft

This is ideal material in which to break records, for the ground can be rapidly drilled by hand augers, shoots fine, can be rapidly shoveled, and the walls stand without timbering for weeks, provided the roof is carefully arched. Fig. 7 gives a typical section of the tunnel driven in this division; Fig. 3 shows the kind of material excavated; Fig. 2 is a flashlight of the completed concrete-lined section; while Fig. 4 is a completed portal of the Saugus tunnel.

From this greatest speed record made in the most favorable material, we turn to examine into the records made in the next most favorable material—the black shale. Perhaps the best records in this material have been made in the second longest tunnel of the work, the San Fernando of the Saugus division, of which D. L. Reaburn is division engineer, and J. W. Henderson is superintendent. This tunnel, No. 104, has a total length of about 10,000 feet and was driven from the two opposite sides of the last barrier ridge of the Coast Range.

The first creditable record was made at the south portal under the charge of Foreman H. E. Warden, during the month of April, 1909. The arched section excavated had a width of 13 feet, and a height of 12 feet 3 inches. Using hand augers for drilling, and working 11 men per shift, 3 shifts per day, a total

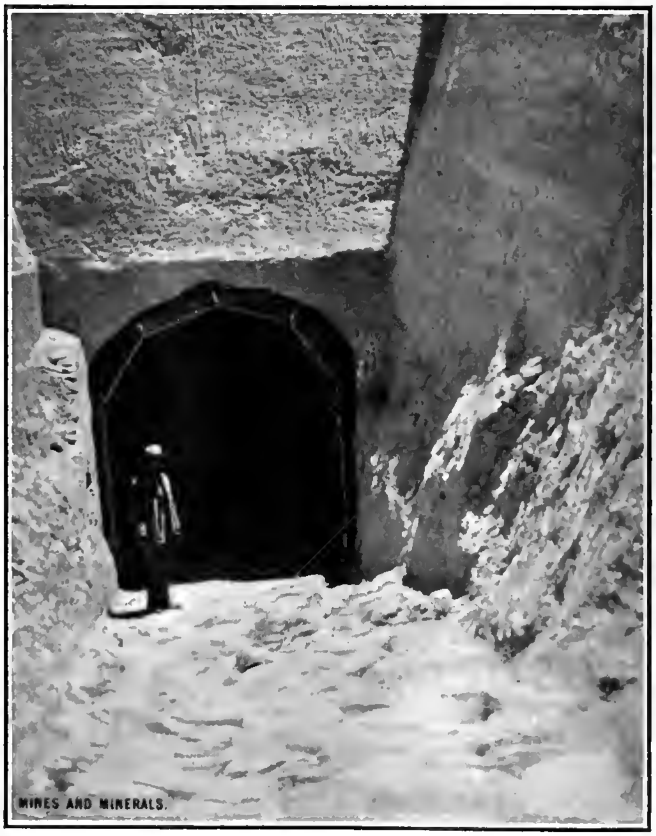


Fig. 3. South Portal, Tunnel No. 18, Jawbone Division

length of but 29 days, the progress made was 586.5 feet. The ground stood so well that the breast could be kept about 12 feet ahead of the timbers. The following were the total official costs per foot of advance, exclusive of concreting: Excavation, \$10.37; timbering, \$3.86; total, \$14.23.

In the following month of May, a record was made at north portal of this same tunnel, in charge of Foreman Sidney Dean, which everything considered is also very creditable. The ground at this end, however, was heavy and necessitated considerable retimbering as the breast was advanced. With other conditions similar to those already detailed at the south portal, the north portal heading was driven 579 feet in 31 days.

The effect of the heavy ground in increasing the costs over those of the south portal is seen in the following average costs per foot, exclusive of concreting: Excavation, \$14.11; timbering, \$5.45; total, \$19.56.

At both of these portals 40-per-cent. gelatin powder was used, of which about 10 pounds per foot of advance was the average consumption.

The crew at the faces on each shift consisted of a boss, 5 miners, and 5 shovelers.

Of the two aqueduct divisions containing hard-rock tunnels, namely, the Elizabeth and the Grapevine, the former has been the scene of all the record runs Up to October, 1908, the best previous American hard-rock record was held by the Gunnison tunnel—449 feet in granite for the month of January, 1908.

Table 3. Elizabeth Tunnel Records

No.	Month	Portal	Superintendent	Distance
1	October, 1908	South	Aston	466
2	March, 1909	South	Aston	474
3	September, 1909	North	Gray	488
4	October, 1909	North	Gray	547
5	April, 1910	North	Gray	561
6	April, 1910	South	Aston	604

Fig. 4. Portal of Finished Tunnel, Saugus Division

In October, 1908, however, the south portal of the Elizabeth tunnel established the new one month's record of 466 feet, and since that time this record as shown in Table 3 has been five times exceeded, till at the end of April, 1910, the record stood at 604 feet. It is noteworthy that this record now ranks fourth in the list of world's records, those ranking above it, all foreign records, being as follows:* Simplon tunnel, 755 feet; Arlberg tunnel, 641 feet; Albula tunnel, 607 feet.

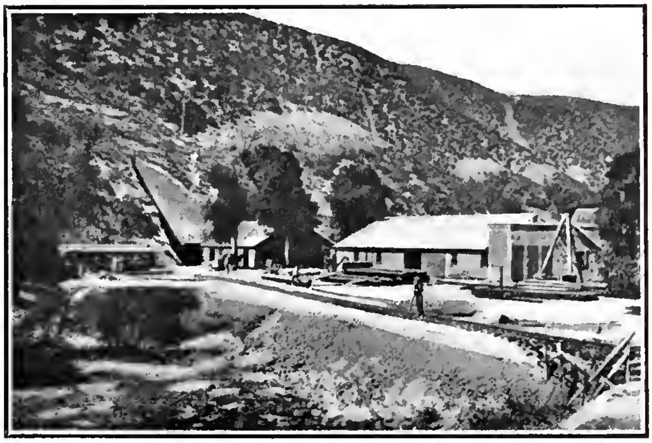


Fig. 5. South Portal, Elizabeth Tunnel

Construction work started on the tunnels of the Grapevine division early in 1909, and, everything considered, progress to date has been satisfactory. No records have as yet been established on this division, probably because the majority of the tunnels penetrate the hardest and toughest rock encountered on the aqueduct. This rock is very largely a tough seamless granite, very similar to the toughest of that encountered in the Cripple Creek drainage tunnel.

Fig. 9 gives a typical section of the excavation. Table 4 gives the costs in dollars of the best month's run up to the time of the writer's visit. In this tunnel the two shifts of 11 men

Table 4. Grapevine Tunnel. South Heading

Classification	Labor	Animals	Supplies	Power	Freight	Cost Per Linear Foot
Excavation	3,199.54	54.00	1,269.22	293.73	253.84	18.37
Engineering	43.64					.16
Timbering						
Totals	3,244.18	54.00	1,269.22	293.73	253.84	18.53

each succeeded in driving 9 feet per day, thus obtaining a bonus on 3 feet per day. There was 28.1 pounds of powder used per foot driven. While 270.5 feet is small progress as compared with the spectacular achievements on other portions of the line, everything considered, it is a very fair record for two shifts per day; besides, next to the Jawbone division, the Grapevine division work presents more difficulties than any other, due to its rugged contour, inaccessibility, etc. In the effort to ascertain ways of increasing progress on these tunnels, the division engineer, C. H. Richards, made the only carefully detailed study on the work which has yet been made on the aqueduct tunnels. The following is the report made by Mr. Richards on the work at the north portal of Tunnel No. 7.

As this is the only compilation of such detailed costs the writer has been able to secure, the report is published entire in the hope that, aside from the data given on aqueduct costs, it may serve as a model for other tunnel men:

*See "Tunnel Driving Records," Mines and Minerals, April, 1909 page 422.

Report on Tunnel Cost

The appended study covers unit costs of driving a timbered tunnel (North Portal, Tunnel No. 7) in the Little Lake subdivision of the Grapevine division of the Los Angeles aqueduct, Edward Comerer, foreman in charge; and the writer, C. H. Richards, engineer in charge of the division.

Ninety feet were driven in 15 eight-hour shifts, the period covered by detailed cost keeping.

The tunnel is approximately 10 ft. × 10 ft. in section; 3½ cubic yards in place per linear foot to pay line; overbreakage about 17 per cent., making a total of 6½ cubic yards of broken material per foot of tunnel.

The heading is in 800 feet, lighted by electricity at 110 volts, ventilated by a No. 3 Champion blower through 12-inch pipe, the heading being cleared in 15 minutes after shooting.

Drilling is done by one No. 7 Leyner drill, water being forced through hollow steel; drill uses approximately 66 cubic feet free air per minute at 83 pounds pressure per square inch, drilling holes to 10 feet in depth.

Mucking is accomplished by use of steel sheets laid down before shooting; No. 3 D-handle, square-point, shovels, and 32-cubic-feet rocker dump cars pulled by a 3½-ton locomotive, running on a 24-inch gauge single track laid with 25-pound steel.

The rock is a close-grained, hard, gray granite with numerous seams, causing the drill to run from alinement, but breaks well. The seams and water combined make it necessary to timber all this ground. The ground carries enough water to make disagreeable mucking, and has to be pumped out.

Timbers are of 6″ × 8″ Oregon pine, spaced 5 to 8 feet apart, as ground permits, lagged with 2″ × 6″ plank. Sets of timbers consist of two vertical posts and a four-segment arch.

Table 5. Labor Costs

Class of Work	Total Hours Labor	Total Labor Costs	Cost Per Foot of Tunnel
Inside Labor:			
Squaring heading	23.50	$ 9.03	$ .100
Setting up and tearing down machine	36.00	16.59	.184
Drilling, one No. 7 Leyner; including shift boss' time	55.33	43.21	.480
Number of holes drilled 150			
Total footage of holes 1,202.30			
Feet drilled per hour, including lost time 15.84			
Feet drilled per hour, actual drilling time 21 74			
Average depth of holes, feet .. 8.00			
Cost per foot of hole, cents ... 3.60			
Fastest hole 9′ 6″ in 10 minutes			
Slowest hole 8′ 6″ in 1 hour 18 minutes			
Average hole 8′ in 22 minutes			
Blowing out holes	5 75	4.15	.046
Loading and shooting	56 25	22.31	.248
Mucking, 412 cars	835.00	268.44	2.980
Trimming, stulling, caves, etc.	102.00	39.24	.436
Timbering (cost per M′ = $11.32)	107.25	40.82	.453
Lost time	40.75	15 66	.174
Bonus, 30 cents per man per foot in excess of 2.3 feet per shift		112.08	1.240
Repairs, to trolley, pump, etc	3.25	1.20	.013
Totals, inside labor	1,265.08	572.73	6 354
Outside Labor:			
Sharpening steel, with Leyner No 2 machine	44 00	17.83	.198
Repairing drill	7.50	2.88	.032
Framing timbers, at shop, pr. M′ = 2.42		8.75	.097
Light and power	90 00	33.75	.375
Totals, outside labor	141.50	63.21	.702
Auxiliary Labor:			
Laying track, 90 feet	31.50	12.75	.141
Drainage line, 90 feet	43.50	14 33	.160
Ventilation line, 72 feet	11.00	3.41	.048
Trolley line, 95 feet	18.00	6.35	.067
Air line, 80 feet	2.50	.80	.010
Water line, 80 feet	2.50	.79	.010
Lights line, 90 feet	8 00	2.86	.032
Totals, auxiliary labor	117 00	41 32	.468
Local Administration and Engineering:			
Proportion of division engineer and assistant's time		50 40	.560
Total labor costs	1,523.58	$727 66	$8 094

The crew consisted of 1 shift boss at $3.50 per day; 4 miners at $3.50; 5 muckers at $2.50, and 1 trammer at $2.50. The blacksmith doing repair work was paid $4 per day.

The four miners worked on day shift drilling the ground, timbering and shooting, the muckers following on night shift, resulting in a clean heading for the drill crews, and nothing interfering with the mucking crew.

Tables 5, 6 and 7 give the amounts and costs of the different items used.

Table 6. Costs of Materials and Supplies

	Total Material Costs	Cost Per Foot of Tunnel
Construction Materials and Supplies:		
Drill repairs, 2 side rods, $1.40; 1 chuck, $15; 2 rings, $2.02; 1 oil can, .15; 1 belt, .16; 20 per cent. freight, $3.74	$ 22.47	$ .250
Cost per foot of hole = .018		
Drill supplies, machine oil, .58; drill steel, 45 inches, $4.12; 412 lb. blacksmith coal, $3.14; 20 per cent. freight, $1.57	9.41	.104
Cost per foot of hole = .008.		
Mucking supplies, car oil, .20; pick handle, .26; hammer handle, .15; 20 per cent. freight, .12	.73	.008
Power, machine, 2,052 K. W. H.; blower and lights, 355 K. W. H.; locomotive, 1,800 K. W. H. = 4,207 K. W. hours. at 1.7c.	71.52	.795
Explosives, tamping stick, .40; 2,700 feet fuse, $11.54; 306 15 gr. Lion caps, $2.08; 650 lb. 1¼ inch, 40-per-cent. gelatin powder, $69.88; 250 lb. 1 inch, 40-per-cent. gelatin powder, $26.88; 150 lb. 1 inch, 60-per-cent. gelatin powder, $20.63; 20 per cent. freight, $26.28	157.69	1.752
1,050 lb. powder = 11.66 lb. per foot of tunnel.		
1,050 lb. powder = 3.3 lb. per cubic yard in place.		
Explosive cost = $.050 per cubic yard in place.		
Explosive cost = $.27 per cubic yard broken.		
Timbers, 3,597 feet B. M. lumber, $59.24; freight on same $55.01; 775 wedges, $5.43; 50 dowel pins, .42; nails, .73; freight, $1.33	122.16	1.360
Timber per foot of tunnel, B. M. = 40.		
Lighting, candles, $7.15; 14, 16, and 32 candle-power globes, $2.58; 20 per cent. freight, $1.94	11.67	.130
Totals for construction materials	395.65	4.399
Auxiliary Material:		
Trackage, 180 feet, 25 lb. rail, $15; splices and bolts, .83; spikes, .18; ties, $1.42; 20 per cent. freight, $3.68	21.11	.235
Drainage 90 feet, 250 feet wire, $2.13; knobs, .07; 2-inch pipe, $4.36; 20 per cent. freight, $1.31	7.87	.087
Ventilation 72 feet, 12-inch pipe, $24.34; 20 per cent. freight, $4.87	29.21	.405
Trolley 95 feet, wire, $6.32; lumber, .37; fittings, $1.42; 20 per cent. freight, $1.62	9.73	.102
Air line 80 feet, pipe, $5.28; fittings, .76; 20 per cent. freight, $1.21	7.25	.090
Water line, 80 feet, pipe, $5.28; fittings, .76; 20 per cent. freight, $1.21	7.25	.090
Light line, 90 feet, wire, $1.48; fittings, .37; 20 per cent. freight, .37	2.22	.025
Total auxiliary materials	84.64	1.034
Total material costs	480.29	5.433
Live Stock:		
Mule 15 days at 90 cents $13.50		.150
Total direct and auxiliary field charges $1,523.58 $727.61	$480.29	$13.667

Table 7. Recapitulation. Costs Per Foot of Tunnel

Labor, direct charge		$ 7.056
Material and supplies, direct charge		4.399
Local administration and engineering		.560
Stock service		.150
		12.165
Labor on tracks, etc.	.468	
Material for tracks, etc.	1.034	
	1.502	
As this work will salvage at about 66 per cent., we deduct	.690	
Net charge for auxiliary work		.812
Estimated proportion of charge for roads and trails on division		1.500
Estimated proportion of charge for buildings on division		.200
Estimated proportion of charge for water supply on division		.220
Estimated proportion of charge for machinery and tools		1.060
Total field charges		15.957
Add 3 per cent. to cost for executive office administration		.475
Total cost of tunnel ready for lining		$16.432

ELIZABETH DIVISION TUNNEL

The greatest obstacle to the success of the aqueduct as a whole was believed at the start of work to be the highest barrier ridge of the Coast Range which is penetrated by the Elizabeth tunnel whose length when completed will be 26,860 feet. The rock composing the ridge is mainly granite, which in places shades into both gneiss and schist. The granite is composed mainly of feldspars and biotite with only a small amount of quartz usually present. Its texture varies at different points along the tunnel, the finely crystalline rock usually being without joints or seams and the coarsely crystalline rock usually being full of seams. The finer crystalline rock, therefore, allows much the slower progress in tunnel driving, while the blocky ground gives about as near ideal conditions for record-breaking drives as one is apt to find.

Table 8. Progress of Elizabeth Tunnel

	North Portal								South Portal		Total	
	Portal Heading		North Shaft Heading		South Shaft Heading		Total					
	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering	Excavation	Timbering
									72	72	72	72
November, 1907	165	158					165	158	117	102	282	260
December, 1907	142	130					142	130	91	75	233	205
January, 1908	115	98					115	98	85	70	200	168
February, 1908	162	71					162	71	115		277	71
March, 1908	135	140					135	140	110	33	245	173
April, 1908	42	79					42	79	156		198	79
May, 1908	100	98					100	98	244		344	98
June, 1908	132	99					132	99	237		369	99
July, 1908	41	128					41	128	285		326	128
August, 1908, {	83											
August, 1908, { 8′×8′	18	100					101	100	288		389	100
September, 1908		143						143	242		242	143
October, 1908, {	59											
October, 1908, { 8′×8′	122	30			86		267	30	a 466	228	733	258
November, 1908	164	159	41	21	62	87	267	267	395	6	662	273
December, 1908	138	95	96	78	166	171	400	344	206	93	606	437
January, 1909	121	101	141	106	206	186	468	393	383	45	851	438
			5									
February, 1909	226	151	90	78	271	150	592	379	319	91	911	470
March, 1909 {	60											
March, 1909 { 8′×8′	185	115	230		246	313	721	428	b 474	151	1,195	579
April, 1909	85	232	105	153	287	228	477	613	453	111	930	724
May, 1909	332	83		77		232	332	392	349	117	681	509
June, 1909	229	590					229	590	251	134	480	724
July, 1909	363	210					363	210	339	59	702	269
August, 1909	404	460					404	460	282		686	460
September, 1909							c 488	440	355		843	440
October, 1909							547	520	413	159	960	679
November, 1909							471	545	378	85	849	630
December, 1909							476	460	480	50	956	510
January, 1910							481	445	515	104	996	549
February, 1910							529	527	516	188	1,045	715
March, 1910							546	603	518	12	1,064	615
April, 1910							e 561	560	f 604		1,165	560
Totals	3623	3,390	708	513	1,324	1,367	9,754	9,450	9,738	1,985	19,492	11,435

a, b, c, d, e, f, Records broken.

Table 9. Progress at North Portal of Elizabeth Tunnel, April, 1910

Classification	Labor	Live Stock	Material and Supplies	Transfers: Labor	Transfers: Stock	Freight and Handling	Total	Unit Costs: Linear Foot
Month Cost and Progress								
(A) Engineer and superintendent	$ 150.00		$ .14	$132.50	$ 27.00	$ .03	$ 309.67	$ .55
(B) Excavation	3,123.40	$ 37.80	1,798.17			359.63	5,319.00	9.48
(C) Mucking	4,572.97	32.40	110.73			22.14	4,738.24	8.44
(E) Drainage	55.00		25.95			5.19	86.14	.15
(F) Ventilation	52.50						52.50	.09
(G) Light and power	596.92		308.30	Energy	1,380.51	61.66	2,347.39	4.17
								22.88
(D) Timbering	2,766.88	27.90	812.22			162.44	3,769.44	6.73
Totals	$ 11,317.67	$ 98.10	$ 3,055.51	$132.50	$1,407.51	$ 611.09	$ 16,622.38	$29.61
Total Cost and Progress								
(A) Engineer and superintendent	$ 12,296.00	$ 713.50	459.91			74.51	13,543.92	$ 1.38
(B) Excavation	85,531.20	775.90	26,969.03			5,117.02	118,393.15	12.13
(C) Mucking	113,492.20	490.17	2,011.47			327.00	116,321.06	11.93
(D) Timbering	41,999.31	238.30	16,520.31			2,983.54	61,741.46	6.53
(E) Drainage	3,668.42		502.64			86.00	4,257.06	.43
(F) Ventilation	1,000.77		1,284.75			194.89	2,480.41	.25
(G) Light and power	14,128.23		33,003.25			1,756.94	48,888.42	5.01
(H) Concreting	1,105.75	343.80	30.71			3.84	1,484.10	
Backfill	539.21						539.21	
(I) Portal cut	1,677.61	84.30	24.06			2.64	1,788.61	
(J) North shaft	8,345.08		3,468.50			629.62	12,443.20	
(E) Equipment	11,750.36	32.40	69,978.24			11,831.13	93,592.13	
Totals	$295,534.36	$2,678.37	$154,252.87			$23,007.13	$475,472.73	$37.66

Powder 16.8 pounds per foot. Estimated cost $763,420. Cost for 9,754 feet, $475,472.—E. J. Sharpe, Superintendent.

While originally the rocks composing both sides of the ridge were doubtless similar, today they are widely dissimilar in physical character, which necessitates two widely differing methods of mining attack.

The rock so far penetrated by the south heading is hard, unaltered, and in general blocky enough to afford fine breaking qualities. It requires no timbering except one or two sets at long intervals where the roof is heavy, and there is comparatively little water encountered.

In the north heading, the rock has been shattered by scores of fissures, and in general is so kaolinized by percolating

Fig. 6. Interior of North Portal, Elizabeth Tunnel

waters that it is for the most part soft, friable and frequently pasty. Considerable water enters the tunnel, and the ground is so heavy it requires timbering. except at one or two places in the tunnel of small extent where hard unaltered rock was penetrated. Long extents of heavy, treacherous ground are continually encountered, so that the timbering must be kept well up to the face.

As a result of these widely varying conditions neither the costs, nor the record performances of work at one heading may be fairly taken as a criterion of the work at the other heading of this same tunnel, a fact the aqueduct supervisors would do well to recognize.

In spite of this diversity of work, however, the progress made at each heading has been surprisingly uniform as shown in Table 8, until at the end of April, 1910, 30 months after starting work, the north heading has penetrated 9,754 feet and the south heading 9,738 feet, totaling 19,492 feet, or nearly 72 per cent. of the entire length. At this rate of progress the tunnel will be finished by April, 1911, or 18 months before it was estimated at the start of work that the tunnel could be completed.

For the accomplishment of this work, destined long to mark the top notch of the American tunnel driving art, two former Colorado men have earned a large part of the credit. They are C. W. Aston, superintendent of the south portal, and John Gray, superintendent of the north portal. Between these two and their crews has sprung up a friendly rivalry resulting in the shifting of the championship of America from one side to the other as each in turn has exceeded the record month's progress of the other, as shown in Table 8.

North Portal Tunnel Driving Methods.—The north portal of the Elizabeth tunnel is located on the desert side of ridge and the effect of the radically different climate upon the vegetation and surroundings is made clearly apparent by contrasting Figs. 5 and 10.

Tunnel driving on the north side of the ridge is an entirely different proposition from that on the south side. For almost the entire half of the tunnel the rock is shattered by hundreds of faults which have rendered the ground both blocky, and usually very heavy, necessitating the timbering of all but 300 odd feet up to the time of the writer's visit. As this side of the tunnel has a down grade, the water which accumulates in considerable quantity, Fig. 6; has to be pumped to the outside. Owing to the soft, blocky ground, which is easily shattered by comparatively small amounts of explosive, the average round at this end includes only 16 holes instead of 25, and they are drilled by two Leyner machines instead of three as at the south heading.

The heading crew, exclusive of the shift boss on each shift, is as follows: Two machine men and helpers (4), 6 shovelers, 2 trammers, 8 timbermen, and from 6 to 8 trackmen, making a maximum total of say 28. To this must be added the train motorman and outside dump man. Instead of hauling the loaded cars from the heading back to the switch where the

Table 10. Progress Sheet of South Portal, Elizabeth Tunnel, April, 1910

Month Cost and Progress								Unit Costs
Classification	Labor	Live Stock	Material and Supplies	Transfer	Energy	Freight and Handling	Total	Linear Foot
(A) Engineer and superintendent	$ 70.00		$ 2.83	$89.50		$ .56	$ 162.89	$ .27
(B) Excavation	3,729.68		2,861.35			572.27	7,163.30	11.86
(C) Mucking	5,136.49	$ 81.00	48.58			9.71	5,275.78	8.73
(E) Drainage	44.49						44.49	.07
(F) Ventilation	85.10						85.10	.14
(G) Light and power	452.04		172.91		$1,450.02	34.58	2,109.55	3.49
	9,517.80		3,085.67			617.12	14,841.11	24.56
(K) Back trimming	416.50						416.50	.69
Timbering	12.00		43.83			34.83	90.66	
Totals	$ 9,946.30	$ 81.00	$ 3,129.50	$89.50	$ 1,450.02	$651.95	$ 153,48.27	$25.25
Total Cost and Progress								
					Freight			
(A) Engineer and superintendent	$ 9,600.96	$ 149.90	$ 327.84		$ 42.69		$ 10,121.39	$ 1.04
(B) Excavation	94,468.43		50,841.14		9,766.20		155,075.77	15.92
(C) Mucking	105,744.49	1,421.30	3,345.07		589.65		111,100.51	11.41
(D) Timbering	6,017.48		4,209.87		1,085.93		11,313.28	5.70
(E) Drainage	1,393.03		13.03		1.56		1,407.62	.14
(F) Ventilation	1,481.30		3,097.86		530.27		5,109.43	.52
(G) Light and power	13,495.78		32,669.59		2,100.22		48,265.59	4.95
(I) Portal cut	720.49						720.49	
Totals	$232,921.96	$1,571.20	$94,504.40		$14,116.52		$343,114.08	$39.68

Powder 19.15 pounds per foot. Feet to be driven, 13,430. Estimated cost, $750,570. Feet so far driven, 11,693. Actual cost so far, $343,114. —W. C. Aston, Superintendent.

trains are made up by a mule, the adverse grade is overcome by rope haulage, a small compressed-air hoist located a short distance back from the face being employed in the work. Of the two trammers mentioned, one runs this hoist and the other manipulates the rope. In this heading from 6 to 8 timbermen find steady employment and the work they perform is one of the main features of interest.

A reference to Fig. 8 will show the section of the finished tunnel and its dimensions, while a careful study of Fig. 11 will make clear the steps by which that condition is arrived at.

For convenience, assume the time of inspection to be at the end of a shift when the drills have been cleared away from the breast preparatory to shooting the round. Arrived at a point 100 feet back from the breast the transition of method in roof support is found similar to that shown in Fig. 11 (*a*). The finished sets are best seen in the end sectional view through $E\ F$, in front of which stand the temporary supports or "false sets." These false sets support the heavy roof by longitudinal timbers resting upon them after the manner of the ordinary forepoling method, the roof being carefully supported on blocks, timber ends, etc., all carefully and tightly wedged. The situation at the breast just before shooting is shown in Fig. 11 (*b*). It will be noted that the last false set has been placed within 3 or 4 feet of the face about to be blasted, carefully braced with timbers and the roof wedged up as tightly as possible.

Shortly following the blasting the timbermen go into the heading as quickly as possible and shore up the roof temporarily from the top of the broken material, as shown in Fig. 11 (*c*).

The posts of the permanent sets are 8″×8″ squared timber, spaced irregularly from 2 to 8 feet apart depending on the weight of the roof; those shown in the illustration are spaced on 8-foot centers. The sets are ordinarily spread by lengths of 2″×6″ plank whose ends are supported for convenience on either wedges or timber ends spiked to the posts. The posts are 8 feet 6 inches in length and set 10 feet 2 inches in the clear while the heading is broken as nearly as possible to a width of about 12 feet. Over both sides and top arch timbers is placed 3-inch timber lagging.

In ground that stands fairly well for a day or two, this full width of 12 feet is blasted by the rounds and the placing of false sets is, of course, an unnecessary preliminary. In the heavy ground, the width of the heading first blasted varies from 6 to 8 feet, or whatever it happens to break, and the rock is so often be crushed. For this reason the timbering in the heavy ground is of vital importance to avoid delays, which mean a loss of bonus to the crews. The only time when the roof is not kept constantly supported, therefore, is the interval of from

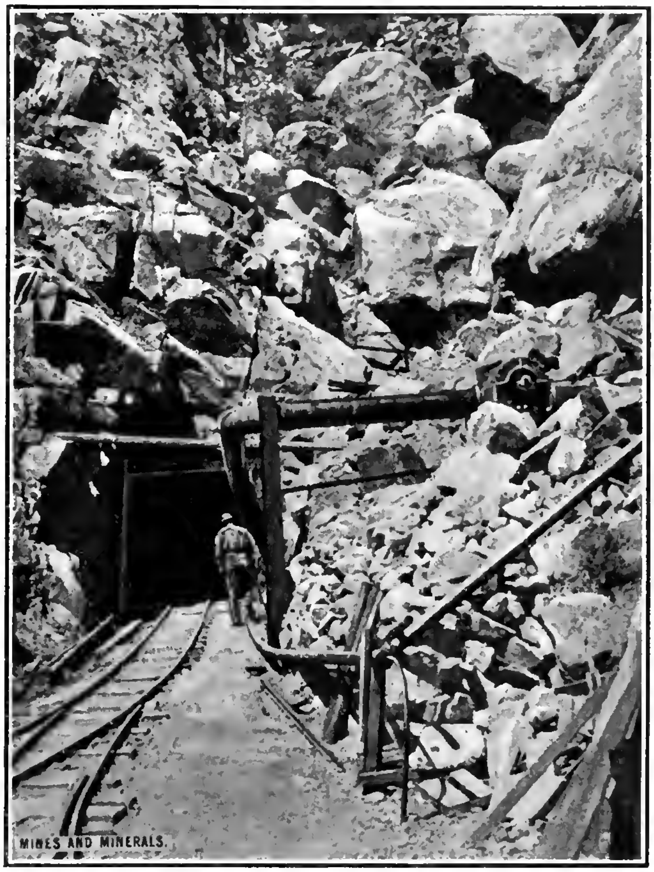


FIG. 10. TUNNEL ADIT IN BOULDER PEAK REGION

15 to 20 minutes following the blasting during which the gases are being cleared out. The first work of the timbermen on the new shift is to shore up the roof temporarily from the top of the rock pile. By placing two longitudinal timbers, Fig. 11 (*c*)

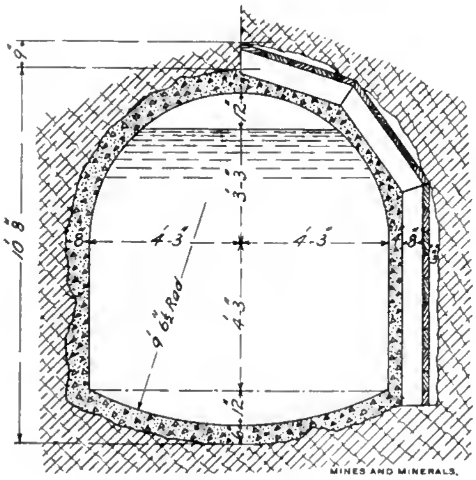


FIG. 7. SECTION OF TUNNEL NO. 17, JAWBONE DIVISION

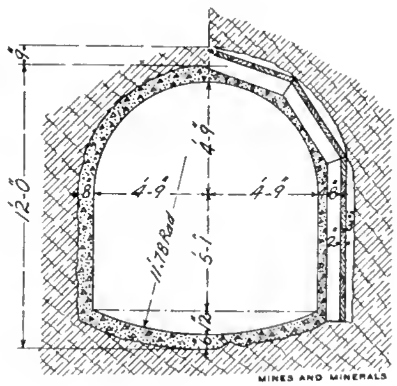


FIG. 8

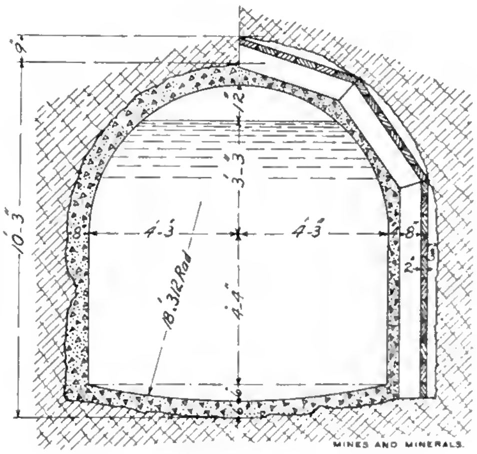


FIG. 9. SECTION OF TUNNEL, GRAPEVINE DIVISION

badly shattered that the subsequent enlarging to the full section is only a matter of judicious pick work. Experience has proved that if the roof is at all times kept well supported on the timbers, so that it does not start to fall, little retimbering will be necessary, while if it once breaks loose, heavy close-set timbering will supported from the broken rock close to the side walls, and shoring up the shattered roof by means of transverse timbers and blocking resting upon them, the timbermen interfere but little with the shoveling back from the face to allow the placing of the drills. This done, the broken rock is thrown aside to allow

placing a false post on solid bottom, which post from here on takes the weight of the roof, while the simultaneous shoveling and drilling for the next round go on. The false posts used in this work are later used as permanent posts, and as their upper ends are beveled to support the roof arch, these beveled ends are placed next the floor, as shown in sectional views *AB* and *CD*, while their squared ends support the caps which consist likewise of the timbers already cut to length, which are later used as permanent posts. In this way there is no handling of heavy timbers not intended for permanent use. With the false sets to the number of six or more, placed for a distance of about 50 feet back of the breast, plenty of leeway is allowed for the ground to work and settle for several days before the permanent sets replace them. The posts of these false sets are placed no fixed distance apart as the width of section first broken varies considerably.

The section *A B*, Fig. 11, contrasted with that of *C D*, shows the situation before and after widening out the tunnel to its proper dimensions by means of pick work. This widening-out process starts next the permanent set and proceeds toward the face, the permanent posts *p*, shown in (*a*), being placed as fast as the section is widened. Transverse spreader timbers shown in section *C D*, are now supported with their bottoms 14 inches below the top of the posts, resting on timber ends spiked to the posts. Across these spreaders are laid two layers of 2-inch plank forming a flooring 4 inches thick. This floor is some 2 or 3 inches below the bottom of the caps resting on the posts of the false sets, so that it is easily laid while the false sets continue to hold the weight of the roof. Working now from the end of this floor, the wedges and blocking transmitting the roof weight to the false sets are carefully knocked out and the shattered roof rock picked down on to the floor. Later on, cars are run beneath this floor, a hole is made in it by pulling out some planks, and muck is allowed to run or is shoveled into the cars.

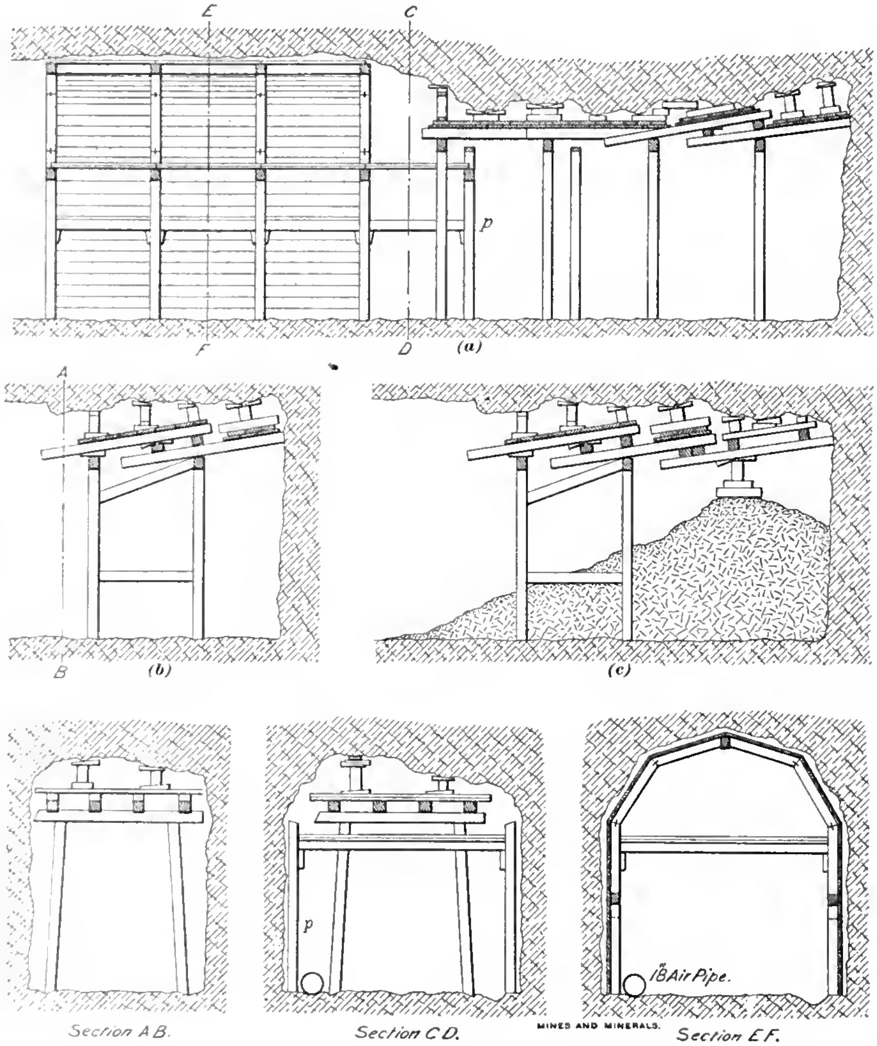


Fig. 11

By advancing the permanent posts *p*, a foot or more forward from the false set posts as in (*a*), the arch timbers of the permanent sets may be placed as fast as the removal of the roof rock exposes their upper beveled ends. Lagging and wedging quickly follow so that the roof weight is taken up by the permanent sets shortly after it is removed from the false sets, after which the posts of the latter are knocked out and carried forward for reuse or are placed as permanent posts.

As the heading is advanced at the rate of 16 to 18 feet per day, 3 shifts, the timbering must be placed and removed by each shift, which keeps every one busy, although system reduces disorder in the heading to a minimum. It is complimentary to the methods adopted by Superintendent Gray that in spite of the hurry with which everything is pushed by the timber crew, up to this writing there has been no fatality in the heading attributable to a roof fall.

To those interested in the rivalry between the crews of the north and south Elizabeth tunnel headings, the accompanying Tables 9 and 10, will be replete with interest as well as with valuable data. These give the official costs for both headings during the month of April, 1910, when the south portal advanced the American record to 604 feet, while the north portal progressed 561 feet in spite of exceptionally heavy ground. It will be noted that, as might be expected, the cost of explosives was the heavier for the south heading, while the cost of timbering is mainly responsible for increasing the cost per foot of the north heading to $4.36 more than the $25.25 of the south heading. It is also significant that the cost of mucking in the two headings is very nearly the same.

And now why is it that in spite of widely diversified methods of attack, of different hardness of rock and many other conditions, the average progress of one heading is about the same as the other? This is surely a fair question whose solution may throw considerable light upon the reason for continued supremacy of foreign records, if indeed they can long continue supreme.

As regards the matter of drills and drilling the writer hazards the opinion that for speedy work the water Leyner has no peer. The fact that both headings maintain about the same speed simply shows that the actual hardness and toughness of rock has but little to do with the problem, at least in this case. It does show that the Leyner drill, owing to its rapidity of stroke, is capable of holing in the soft seamy ground of the north heading, where some drills would be continually fitchering, as rapidly as in the hard fine-cutting granite of the south heading.

Another fact worthy of note is that continuous timbering can no longer be regarded as a bar against speedy work, except where quicksand is encountered, for what time is lost in timbering is balanced by the easier breaking of ground.

The writer is now inclined to the belief that the breaking of records has proceeded to the point where it is a shoveling

problem rather than a ground breaking one. Given an American loading system capable of offsetting the cheap labor handicap of foreign record breakers, the results may soon read differently.

The visitor to the various camps along the line of the aqueduct has only words of praise for every department save one, and that is the commissary department. In spite of the determined and persistent efforts of the men to secure wholesome food they have so far been unsuccessful. If the commissioners would unexpectedly visit the camps under an alias we believe they would quickly change conditions from what at the time of the writer's visit he can only compare to the fertilizer department of a glue factory.

In conclusion the writer extends his hearty thanks to the many executives of the aqueduct work who have furnished data and extended asistance to him, an enumeration of which would read like a directory of the force. May the above remarks on the commissary department result in action that will prove some return for the many courtesies extended. Special acknowledgment, however, is due to Mr. J. B. Lippincott, assistant chief engineer, to whose courtesy the writer is indebted for an automobile trip over the line and to whose kindness he is indebted for much of the tabulated cost data and many of the accompanying illustrations.

KENNETT SMELTER FUME ABATEMENT

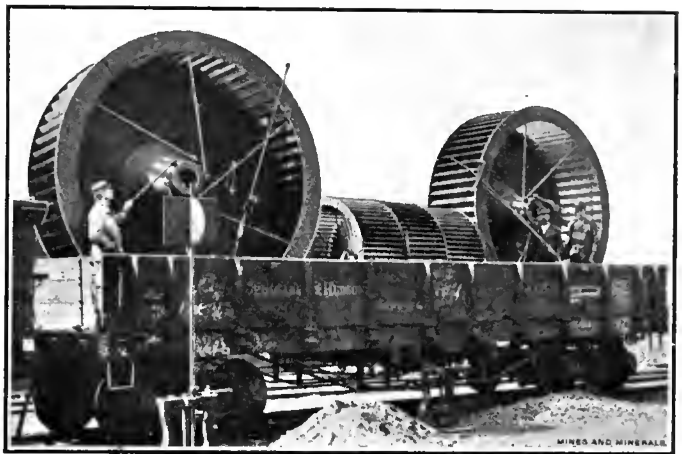
Written for Mines and Minerals, by E. B. W.

The Mammoth Copper Mining Co., controlled, through the ownership of the entire stock issue, by the United States Smelting, Refining and Mining Co., is located 3½ miles from Kennett, on Little Backbone Creek, in Big Backbone district, Cal. The smelter, ½ mile from Kennett, 2½ miles from the mines and 2,200 feet lower, is the largest in California.

The Kennett smelter, which originally had a capacity of 1,000 tons daily, was enlarged in 1907 and the capacity increased to more than 1,500 tons daily. The buildings are of steel, and all material is handled by gravity. Both the mine and the smelter machinery is operated by electric power obtained from the Northern California Power Co. at a reasonable rate.

The smelter plant occupies 80 acres and is equipped as follows: Sampling mill; ore bin house 214 feet long; blast furnace building 72 ft. × 222 ft., which houses three 42″ × 180″, and two 56″ × 180″ furnaces; converter building 92 ft. × 204 ft. with 2 hydraulic stands, 8 converter shells, and 50-ton electric crane; power house 250 feet long, containing seven 15,000 cubic feet per minute blowers, driven by three 200-horsepower and four 250-horsepower motors; boiler house 43 ft. × 176 ft., out of commission, but held in reserve for emergencies; shop building 41 ft. × 432 ft., divided into machine shop; blacksmith shop; shed; warehouse; boiler shop and locomotive stable; office and laboratory buildings complete the plant, so far as buildings are concerned. The equipment in the buildings would read like a stock inventory and in any case could only be appreciated by the experienced, who know that the cost of furnaces is merely a drop in the bucket, to the cost of machinery, apparatus, etc. needed to run them.

When the furnaces were in full blast, about 5 tons of arsenic, 7 tons of lead, and many tons of sulphur escaped from the furnace stack as fume. The farmers in the vicinity declared the fume killed their crops and served an injunction that prevented the company using this plant. The method adopted for overcoming the fume difficulty at the United States Smelting Co.'s Midvale smelter, in Utah, was described in the March, 1910, issue of Mines and Minerals, while the arsenic plant was described in the June, 1910, issue. The United States Smelting Co. is now installing their patented system of neutralizing the noxious fume in the Kennett plant, after which the injunction will be removed in California, the same as it was in Utah in the case of the Midvale furnace. The blast furnaee gases which have a temperature of approximately 700° F. are cooled down to nearly 200° F. and filtered through bags in the bag house. For this purpose powerful fans which will create a partial vacuum equivalent to from 1.5 to 2 inches of water gauge are needed, owing to the friction involved in passing the gases through cooling pipes, flues, chambers, and eventually delivering the smoke to the stack. The two single-inlet Sirocco fans, each 132 inches in diameter shown at the ends of the car in Fig. 1, are destined for the Kennett plant. Each fan is to circulate



Sirocco Fans for Circulating Gas

245,000 cubic feet of gas per minute that has a temperature of 450° F; owing to this temperature and to the altitude, which is 1,000 feet above sea level, the fan must make 235 revolutions per minute and absorb not over 335 horsepower for 5-ineh water gauge.

WET ASSAY FOR GOLD

Written for Mines and Minerals

Dr. F. Jerome Davis, of Raymond, Cal., furnishes the following method of making a wet assay for gold:

The material is pulverized to pass through a 100-mesh screen. It is then soaked in a solution composed of two parts of potassium, four parts of potassium iodide, and 100 parts of water. From 40 to 50 cubic centimeters of this solution is taken to each assay ton of ore. If the solution fades, more solution is to be taken, as there is a reducing. element in the ore. Grind the pulp wet with solution in a Wedgewood mortar at intervals for 2 hours, giving a few rubs each time, then filter off and wash the ore, putting filtrate in a flask or large test tube. Next add from 40 to 60 grams of zinc amalgam, agitate freely, add nitric acid, and boil until a porous button of gold is present. It is necessary to boil slowly or the button will be broken apart. Weigh as in the case of fire assay. In the case of silver, ½ assay ton may be used, the silver precipitated from the silver nitrate solution with hydrochloric acid, and then weighed as silver chloride. In some cases pure mercury is used instead of zinc amalgam with as favorable results.

Another method which is advanced by W. H. Seaman in his recent book on "Manual for Assayers and Chemists," is as follows: A suitable quantity from 10 to 1,000 grams of finely powdered mineral is placed in a large bottle thoroughly wetted with water and then treated with 500 cubic centimeters of bromine water for a period of several days. The bromine forms tribromide of gold $AuBr_3$. With portions of other metals the solution is filtered off, the residue washed thoroughly, and sodium peroxide added to the solution, which precipitates the gold according to the following equation:

$$2AuBr_3 + 3Na_2O_2 = 2Au + 6NaBr + 6O.$$

The precipitates are then taken up with hydrochloric acid, and after the gold is settled the liquid is decanted off and the fine gold caught on a small filter, after which it is dried, ignited, and weighed as a metallic gold.

TIMBERING IN THE JOPLIN DISTRICT

Written for Mines and Minerals, by Lucius L. Wittich

To support the drifts in the soft-ground zinc and lead mines, of the Missouri-Kansas-Oklahoma district, heavy timbering is required. The process is expensive, but failure to timber a drift properly may mean a cave-in entailing damage to property, possibly loss of life.

Conditions that Require Large Quantities of Timber. Method of Placing. Costs

That disastrous cave-ins are not impossible is shown by Fig. 1, this view having been taken when the roof of the Morning Hour Mine, west of Joplin, Mo., dropped without an instant's warning, the track of the St. Louis & San Francisco Railroad being left suspended like a sagging string across the chasm. Other cave-ins have occurred since in this immediate vicinity that stopped traffic on the railroad on several occasions.

Owing to the pockety formations of the zinc and lead ore deposits the drifts of a necessity must conform to the dimensions of the ore bodies if mining is to be conducted at a profit, for which reason it is necessary to make some excavations wide and high while others may be low and narrow. In Fig. 2 is seen a diagram of the typical pocket formation and the method of working it by underhand stoping which is followed in the majority of the mines. In this system the ore is taken up from below, a face about 6 or 8 feet in height being worked first, then, if ore still exists in the floor, a stope is taken up. This process is carried deeper and deeper until the bottom of the mineral has been reached. In some of the mines of the Joplin district stopes 80 feet in height are worked, but such instances are found only in substantial ground where the placing of little or no timbering is required.

Fig. 1. Cave In at Morning Hour Mine

Timber suitable for mine purposes is plentiful throughout the district, but the supply is diminishing rapidly; especially since operators have displayed a tendency to return to the development of the shallower ore deposits. Many of the deep, hard ground operations in sheet ore have been suspended temporarily on this account.

In following an ore body a drift will twist and curve, but soft-ground deposits as a rule are sufficiently irregular to permit of pillars of barren rock being left as supports for the roof. Occasionally, though, it becomes necessary to leave pillars of ore that contain enough mineral to warrant the operators returning to them eventually and removing them at the risk of a cave-in. It is a common practice to work out an ore deposit, then return to the outlying pillars first and take them down one at a time until the roof of the mine is ready to drop without warning. Timbers seldom are employed as permanent supports, as no tree trunk is sufficiently massive and strong to withstand the weight of earth and rock above it; pillars of rock are a necessity, timbers being employed only to support the walls of the excavation temporarily. In some of the softer mines it is necessary to place large timbers side by side as close as it is possible to get them, and even these cannot hold up the roof when it commences to crush. Heedless, apparently, of impending danger, the Joplin miner will labor day in and day out beneath the splintered timbers, for it is common for massive supports to fracture and yet keep the overhanging tons of rock from crushing down. Even when the strain becomes so great that the timbers creak and groan and crack like rifles the miners realize that the collapse will be slow and with a watchful eye they continue at work. New and even heavier timbers are installed if occasion demands and thus the strain is removed from the ones already broken.

Without timbering the development of a soft-ground mine would be almost impossible. By diverging the advancing timbering the size of the drift may be varied; however, it rarely exceeds 18 feet in width by 30 feet in height, while much smaller drifts are the rule. The cover above the excavations is so heavy that long timbers cannot withstand the strain; therefore, they are seldom used.

In Fig. 3 is shown the common method of timbering soft ground. Where cavities occur back of the timbers, lagging is installed, and by the aid of wedges is forced up against the roof. Behind the lagging, nailed to the legs, short pieces of wood varying in length from a few inches to several feet, are thrown in a conglomerate mass to prevent loose dirt from the sides falling into the drift.

Drifting is accomplished by starting a tunnel from a shaft toward the ore, as it is poor policy in this kind of ground to sink a shaft directly into the mineral deposit. Therefore it sometimes occurs that the bottom of the shaft may be some distance from the ore deposit, necessitating the driving of tunnels through barren ground.

When the drift has reached a length hinting of possible disaster unless precautions are observed, two posts, and possibly a mud-sill and a cap are installed. As the tunnel is lengthened, additional sets of timbers are put in place. Stakes, or poling boards, consisting of split green poles about 5 inches in diameter and known as "spiles" are driven forward over the cap of the set farthest in advance. The softness of the ground will determine how many stakes will be required. In extremely soft ground it will be found necessary to observe extra precaution. The dirt is removed from beneath the stakes, temporary props being employed if necessary until the drift has been advanced to a point warranting the installation of another set of timbers.

An excellent plan is to bevel the "spiles," then drive them forward with the bevel downward. This has a tendency to cause the stake to take an upward course, which is desirable. As the earth is removed from beneath, the pressure from above will cause the spiles to settle. Without proper elevation, on being driven into the face the weight would cause the spiles to settle so that it would be impossible to maintain the regular size of the timber sets and the drift gradually would become smaller and smaller. In addition to the rise resulting from the bevel, the spile should be pointed upward at an angle of about 6 degrees. Where the roof is soft the angle should be increased. As the timber sets are placed the spiles settle down on the cap and form part of the timbered set.

In the oozy ground sometimes encountered, extra precautions are needed when the roof spiling will not hold the mass of earth and rock. It then becomes necessary to advance a cap timber by excavating near the roof of the drift and pushing steadily into the face. Small cuts at either side of the drift are then extended until it is possible to support the cap with posts. This accomplished, the earth in the center of the drift can be removed in comparative safety.

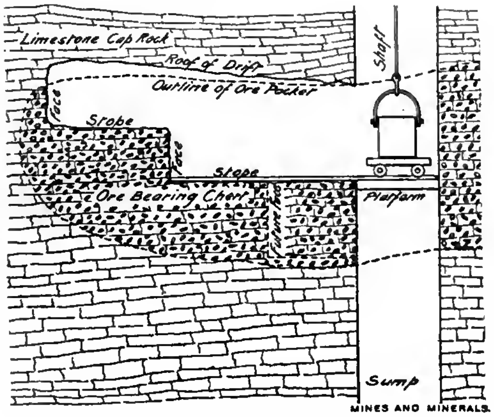


Fig. 2. Method of Mining

Where the ground is fairly substantial it is possible to do away with the lengthwise timbering. the main sets being able to hold the roofs and sides. The principal danger in these mines lies not in the possibility of a falling boulder or two, but in the probability of a disastrous cave-in that will extend to the surface and carry with it loss of life and great property damage.

Where the ore is richest, timbering almost invariably is required. The money-making mines have not all been located in soft ground, but where the percentage of ore is exceptionally high it is found as a rule in ground that is easily worked. Frequently nature has done the work for the operators, by leaving a cap rock of solid limestone over the ore deposit. Where this is the case no timbering is needed. Soft-ore deposits occurring between beds of solid lime have been discovered and such conditions are ideal for mining in the Joplin district, because mining requires small amounts of money for explosives or timbering.

In the Joplin district round sticks are used for timbering purposes, as timbers in this form are more substantial than if dressed.

The life of the average soft-ground mine is short; seldom does the typical pocket producer prove a bonanza for more than 2 or 3 years, although there are examples where such mines have steadily produced for 10 years. Occasionally the ore pockets are closely connected, being joined with thin runs of "shines" that lead the operators to extend prospect drifts into fresh deposits. Some of the Joplin district mines have succeeded in developing pocket after pocket, the deposits sometimes occurring in circular form, while at others they are found in runs.

Where it is necessary to stope in soft ground the operators will save themselves future trouble by using mud-sills in the original drifts, as eventually the floor of the first drift must be taken up. How to accomplish this when roofs and sides are braced and supported, and are none too secure at the best, is a problem. To go beneath these timbers, hold them in place by equally heavy or even heavier timbers, is a necessity if the ore is to be handled. The advisability of employing the mud-sills in the original drift therefore can be appreciated. Unless the mud-sills have been used, the space of the first drift must be cribbed and retimbered, comparatively light timbering from 4 to 8 inches in diameter being sufficiently heavy to answer the purpose. Securely wedged at the top, the cribbing is ready to rest upon the caps of the sets in the drifts to be driven underneath. The method of advancing the stope beneath the old drift is much the same as that employed in driving the original drift. Care is taken to prevent the timbers from above coming down. Overhead stoping sometimes is adopted, the ore deposit where unusually thick being worked from the bottom after the initial drift has been driven. In this type of development the caps of the first drift act as supporters for the platform of the second drift to be driven above the former workings.

An enormous amount of timber also is employed in lining shafts. While the size of this material is less than that used in the drifts, the amount used possibly is as great as that employed underground. Unless sunk through absolutely solid ground, practically all the shafts are lined from top to bottom.

The cost of timbers is regulated by length and diameter, the smaller end always being used in the latter measurement. A log 10 feet in length costs 8 cents per inch of the diameter at the smaller end, unless the log be 10 inches or more in diameter, in which case the cost is 10 cents per inch.

The following gives a fair idea of the cost of timbers in the Joplin district:

Twelve-foot logs, 9 cents per inch up to a 9-inch diameter; above 9 inches, 12 cents per inch; 14-foot logs, 11 cents per inch up to 11 inches in diameter; above 11 inches, 14 cents per inch; 16-foot logs, 12 cents per inch up to 12 inches in diameter; above 12 inches, 16 cents per inch; 20-foot logs, 20 cents per inch, regardless of diameter

The price charged for lagging is about as follows: 3 to 4 inches in diameter, 10 or 12 feet long, 15 cents each; 4 to 5 inches in diameter, 20 cents each; 5 to 6 inches in diameter, 30 cents each. Cord wood, delivered at the mine, $5 per cord; spiles, 40 cents each; collar braces, 10 to 15 cents each.

The cost of setting timbers in the drifts is about equal to the cost of the timbers, although this is only an approximate deduction arrived at by half a dozen of the larger operators in the soft-ground districts. The cost of excavation varies. In an extremely soft sheet ground mine in West Joplin the cost of excavating a drift 12 feet wide, 12 feet high, and 12 feet long was only $25. No powder was required.

The average cost of lining a $4' \times 5'$ shaft with $2'' \times 4''$ plank, at $15 per 1,000, ranges from 95 to 99 cents; a $5' \times 7'$ shaft will cost about $1.25. Rough timber of scrub oak costs from ½ cent to 1½ cents per foot, making a 10-foot length cost about 8 cents

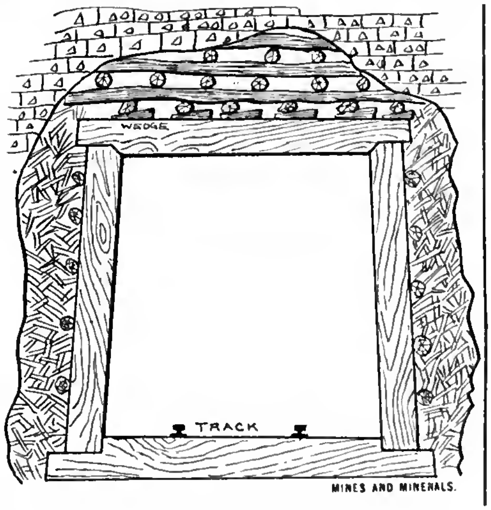


Fig. 3. Timbering in Soft Ground

on an average. These timbers are sawed in two lengthwise, and thus two cribbing timbers are secured from each pole.

Concrete, made partly from the tailing of the Joplin mines, is proving successful as cribbing, while there is much talk of experimenting with this material in bracing the roofs of drifts. On the Granby land at Joplin, and at the Eureka Mine, north of Galena, Kans., concrete has been found effective in walling in the shafts. It is superior to timber in that it prevents the leakage of surface waters.

OXYGEN PROCESS FOR MELTING OF IRON

Written for Mines and Minerals, by Dr. Alfred Gradenwitz

Method by Which Large Masses of Iron or Steel May be Quickly Burued Through

The melting process invented by Dr. E. Menn, engineer to the Koln-Muesener Bergwerks-Aktien-Verein, at Creuzthal, Westphalia, enables iron and steel blocks of considerable thickness to be pierced rapidly in a horizontal or vertical direction, and has been found especially suitable for removing of blast and smelting furnace obstructions. The process is as follows:

The iron or steel masses having been heated by some means or other, preferably with an oxy-hydrogen flame, up to the ignition temperature of their combustible components, oxygen is forced against them at a high pressure (upwards of 30 atmospheres). Though this stream of oxygen exerts some cooling effect, or even puts out the flame entirely, the heat of combustion of the iron causes the metal masses to melt, while the high pressure throws the oxidized and molten material out of the hole thus formed, so that the oxygen always finds renewed points of attack in the combustible material, and cannot exert any cooling effect. Oxygen, in virtue of the well-known law of expanding gases, will cool down considerably below the freezing point of water, and the fact that this gas allows effects to be obtained which are impossible in the case of the hottest of all flames, the oxy-hydrogen flame, at first sight seems paradoxical. The apparent contradiction is accounted for when it is considered that each volume unit of burning iron develops about 5,000 times more heat than an equal volume unit of hydrogen.

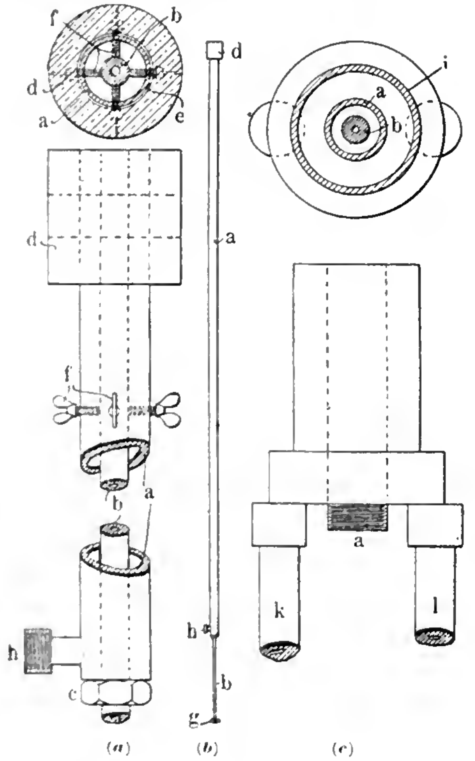


Fig. 1

It is true that the hottest parts of the oxy-hydrogen flame are heated above 2,000° C., so that an ordinary Bunsen burner fed with air will melt thin platinum wires. Any thicker wires will at most become red hot, as the heat losses in this more extensive mass exceed the heat supply. In a similar manner, an oxy-hydrogen flame allows thin sheets to be pierced; as soon, however, as the mass considerably exceeds the flame volume, the heat supply is distributed over the mass of the block, and only exerts far more moderate effects.

The oxygen process, however, is quite independent of heat losses, and allows the largest blocks to be pierced with the smallest flames, owing to the extraordinarily high pressure which throws away the molten molecules more rapidly than heat dispersion is allowed to come into play.

The apparatus is simple and inexpensive. It consists of two steel flasks, each with a reducing valve, and of a few yards of iron tubes and armored hose.

The design of a suitable burner at first offered some difficulty. As this burner, on being introduced into deepening holes, is surrounded by the recoiling flames and the oxygen, its point must, in fact, be protected efficiently lest it become melted more quickly than the block to be pierced. All the usual burner safeguards (platinum, soapstone, etc.), proved ineffective in this connection, whereas heads made of compressed coal, coke, graphite, retort carbon, magnesite, and especially carborundum, were found sufficiently resisting.

Fig. 1 (*b*) represents the burner with its head and sleeve, and the connection tubes for supplying combustion gas and oxygen, and (*a*) details of the burner itself.

The tube *a*, supplying the combustion gas, is arranged outside the tube *b* supplying the oxygen, and which, at its rear end, is tightened by a stuffingbox *c*. The sleeve surrounding the burner head, made of compressed carbon or the like, is shown at *d*. Between *a* and *d* there is provided an interval *e* filled with yielding material, such as asbestos paper, for preventing the sleeve *d* from breaking as the tube *a* is expanded by heat. The setscrews *f* are holding the tube *b* in position while *g* and *h* are fittings for connecting the hose. The sleeve can be made of several rings so that in case of rupture of the foremost ring the burner point only can melt down to the next.

Beside these sleeves of refractory material, water-cooled burner heads, as represented in Fig. 1 (*c*), have likewise given good results. The hollow burner head *i* is made of metal, and provided with a water cooler *k l*. Burner heads have been made of copper provided at the outside with grooves 3 to 4 millimeters in depth which are smeared with loam so as to damp the heat of the flame and the projected iron by a bad conductor of heat. This arrangement prevents the cooling water from being heated to any considerable degree. The burner is preferably some yards in length.

This simple appliance is worked as follows:

The hydrogen is allowed to escape first from one of the two tanks shown in Fig. 2, and is lighted, after which the stream of oxygen from the other tank is turned on. The pressure of both gases is first kept low, but gradually raised and regulated in such a way as to give a very hot flame, which heats the spot upon which it impinges to a white heat, as in Fig. 3. The pressure of the oxygen is then raised to such an extent that the iron commences to burn, which is shown by sparks being thrown about, as in Fig. 4, taken half minute after Fig. 3.

The pressure of the oxygen is now further raised to 30 atmospheres and above, whilst the supply of hydrogen is entirely stopped. It is now the iron alone which burns, thus replacing the hydrogen as a combustible, whereby a degree of heat is developed which far surpasses that produced by means of oxy-hydrogen gas. The high pressure of the escaping oxygen serves to force out all the molten iron, thus keeping the hole perfectly clean throughout the operation. In this way a solid block of cold iron or steel, say 16 inches thick, can be pierced within 1 or 2 minutes.

The heating effect of the oxy-hydrogen flame alone is far too low to serve the purpose of opening closed-up tap holes or tuyeres. This is principally due to the fact that (on account of the comparatively great volume taken up by hydrogen) the loss of heat of formation is much too large, as compared with iron, taking both as combustible matters into account.

The oxygen process, however, has been successfully applied for opening tap holes closed up by solid iron or steel. It is of great importance that no delay should occur in tapping an open-

hearth furnace as soon as the liquid metal has reached the desired degree of decarbonization, as otherwise its composition might change to such an extent as to render it useless. The new process is extremely rapid in working, a few minutes being sufficient to perform the operation. Fig. 5 was taken half minute later than Fig. 4: the tap hole is being opened, the iron pouring out.

The same process is also used for improving steel ingots by removing the "pipe" caused in the upper portion owing to the shrinkage during cooling. The crust of solid steel or iron above the pipe is burnt through by means of the outfit above described within less than a minute, and superheated liquid metal of the same quality as the rest of the ingot poured in so as to thoroughly fill up the pipe.

Another successful application has been made for removing dead heads or runners on steel castings, which is of special value in cases where the hardness of the steel casting is such as to resist the action of the cutting tool.

The oxygen process can be employed also in rolling mills, where work can be discontinued only at considerable expense, for melting through and quickly removing any broken shaft or axletree from couplings, flywheels, and the like. It has also been advantageously used for piercing armor plates for war ships, and armored turrets, which is of special importance in case such plates are of hard metal. A hole through such a plate of say 9-inch thickness would require 2 or 3 hours if drilled in the ordinary way, whereas the same work can be done with compressed oxygen within 15 or 20 seconds. In this case the electric current has been found more convenient than oxy-hydrogen flames for producing the necessary initial heat, a current of from 4 to 6 volts, and about 200 amperes, being quite sufficient.

FIG. 2. OPENING A TAP HOLE

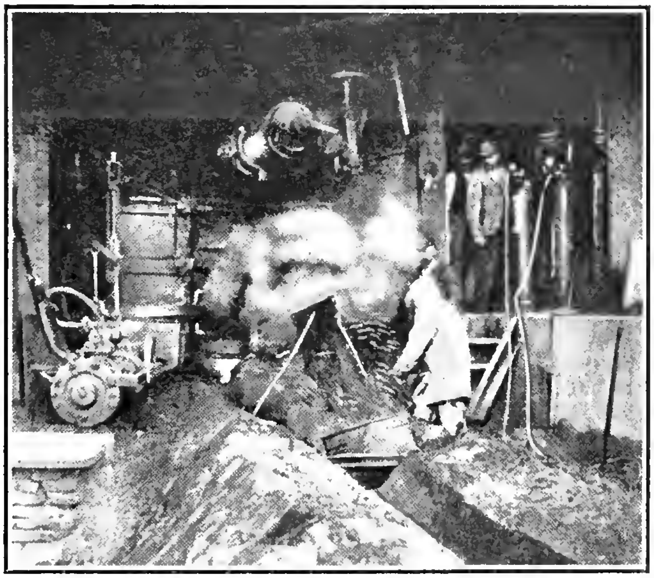


FIG. 3. PREHEATING

FIG. 4. ONE-HALF MINUTE AFTER FIG. 3

FIG. 5. IRON POURING OUT ONE-HALF MINUTE AFTER FIG. 4

The principal outlay is for the oxygen gas, which costs about 75 cents for 35 cubic feet, so that the working expenses are far from being prohibitory.

The Yukon placers yielded their largest annual output in 1909. Practically all the Yukon camps have increased production, those of the Tanana Valley leading with about $10,150,000. Of this amount the Fairbanks district produced $9,650,000. A further increase of production is recorded for the Koyukuk district and an important feature of the year's mining was the continued success of dredging in the Fortymile district.

HISTORY OF THE WATER LEYNER DRILL

Written for Mines and Minerals, by Chas. A. Hirschberg

The Development of Hammer Type of Rock Drill Delivering Air and Water at Point of Steel

Any "History of the Rock Drill" would be incomplete without mention of the Water Leyner drill, which is in general use wherever mining or tunneling is in progress, and marks a stage of advancement in rock drilling, the same as the electrified street railways of today have superseded the old cable tram lines.

Mr. J. Geo. Leyner is of German descent and was born in Boulder County, on August 26, 1860, being the second white male child born in what is now the state of Colorado. His father was a ranchman, but Mr. Leyner very early developed a genius for things mechanical and was not content to stay and take up the work upon his father's ranch.

He spent a number of years in active mining, both in the mechanical and in the underground departments. In 1893 he located in Denver, opening a shop for the repair of mining machinery. In 1895 he brought out his first rock drill, which was of the reciprocating piston type; 1897 saw his first hammer drill and 1898 the first Water Leyner drill.

There is no questioning the debt the mining industry owes to the invention of the reciprocating or piston type of rock drill. It has been the means of increasing the world's mineral production, more than any one thing, through placing the operator in a position to work the low-grade ores that previous to the advent of power drilling were in the prohibited class, due to the cost of hand drilling absorbing the full value of the mineral contained in the rock.

But, from the advent of the first reciprocating piston drill up to the present day, there has been but little material improvement made upon them along the lines of greater drilling efficiency or tending to abate the very serious dust nuisance, which has been the bane of the rock-drill operator.

It remained for a western man, born and raised among the hills of Colorado, to raise the standard of efficiency in rock drilling machines and also bring about a healthier underground condition, by the invention of a water device which allays the dust and prevents that dread disease, miners' phthisis, which has always been associated with drilling, except where this device is used.

The first drill brought out by Mr. Leyner, in 1895, was of the reciprocating piston type, similar to those on the market at the present time. Being, however, of a progressive and inventive turn of mind, he was not satisfied with this type of drill, which had then been in use practically 40 years. He began experimenting with drills involving the principle of the hand hammer blow and soon became convinced that this was the ideal one for rock drilling, due to the fact that the whole energy of the blow would be imparted to and through the drill steel to cut the rock and would not be wasted in overcoming inertia and friction. This is the case with piston drills in which the steel is rigidly fastened to the piston and both move back and forth in the drill hole several inches, whereas, in the Leyner drill the only moving part is the hammer. This construction also gives a weight that is light and constant at all times, permitting lighter construction and greater drilling speed.

The correctness of this principle is generally admitted by the mining fraternity and by manufacturers, who, until recently, advocated the reciprocating piston type of drill, exclusively.

Other inventors had previously experimented with drills of the hammer type, none of which, however, proved successful commercially. Mr. Leyner is, therefore, the pioneer in the advocacy, invention, manufacture and sale of drills constructed on this principle, which has proved successful.

The first hammer drill which he produced did not embody the water feature, but, realizing the value of cleaning the holes of the rock cuttings while drilling, and, as the dust created by passing air alone through hollow drill steel to the cutting point prohibited the use of the drill as then constructed, he conceived the idea of passing air and water combined through the drill steel to the cutting point.

Passing air and water through the drill and the drill steel is very important to the mine manager, for the reason that it ejects the rock cuttings from the hole during the process of drilling so that the steel is cutting virgin rock with every blow. Further, the fact that the dust is allayed at the same time, improving underground sanitary conditions, is receiving the attention of governments, notably that of the British government, particularly in South Africa and in Australia.

The first model of the Water Leyner drill was brought out in 1898 and demonstrated the value of the hammer principle as well as the water feature. The attention of Mr. Leyner has since been devoted to simplifying and strengthening the means of utilizing these two features, with the result that now the Leyner rock drill consists of the fewest possible number of parts, each of ample strength to constitute a rock-drilling machine of the highest efficiency and economy yet attained. In late models, new appliances of utility and convenience have been added, one of which is an automatic oiling device which consists of a pocket or oil chamber cast on the side of the drill cylinder, from which ports lead into the cylinder and are covered by the hammer, which at every movement, wipes a light film of oil sufficient to oil the machine throughout. The importance of this is at once apparent, for rock drills, more than any other class of machinery, are subjected, not only to abuse, but to passive inattention and neglect as well. This device insures proper, automatic, continuous lubrication of the drill without any attention from the operator.

The air throttle, unlike that of other drills, is located in a taper seat formed in the drill cylinder casting. This is automatically pressure packed and consists of two parts, a handle and valve, while its location is both convenient and safe.

Another important feature of the Water Leyner drill is that it permits of the use of an improved system of pointing drill holes, which, when charged with explosives and fired, break the ground more effectually than by other systems. The term "Leyner cut"* as applied to a round of holes has come into general use on account of the many upper or dry holes involved which can be drilled to advantage only with Water Leyner drills. From the fact that the drill holes can be pointed exactly as they should be, without regard to angle, direction, or condition of the face and are not limited in position by the unwieldiness of the machine, as is the case with piston drills, much less powder is required and the same breaking effect can be obtained with fewer drill holes of smaller diameter.

It is possible with Leyner drills to carry out accurately every time, any system of holes that may be devised for breaking the rock to the best advantage. In other words, a hole can be started wherever desired and bottomed accurately at a predetermined point, which is an advantage especially valuable in tunnel driving. In fact, the Leyner drill has reached such a point of efficiency that today it holds the three best American tunnel-driving records, not alone for drilling speed, but on a basis of cost per foot of advance, for maintenance, power, and labor.

Owing to the rapid growth of Mr. Leyner's business it was necessary to move to larger quarters, but in 1902 it was found that still larger quarters would have to be provided, and, consequently, in September, 1902, he formed a stock company, incorporating under the name of the J. Geo. Leyner Engineering Works Co. Twenty-six acres of land were arranged for just north of Littleton, Colo., a suburb of Denver; new shops were erected and October, 1905, found the company occupying its present home.

The shops contain 107,350 square feet of floor space and are equipped with the most modern machine tools, all operated

* See Mines and Minerals, June, 1910, page 652.

by electric motor, current being generated by their own plant, making the combination one of the most compact modern manufacturing plants west of the Mississippi River. Although when built it was expected to be ample for the company's requirements for many years to come, the need for more room is becoming very apparent.

LAKE SUPERIOR COPPER ORES

The persistent rumors that the Calumet & Hecla Mining Co. is planning to curtail still further its production of copper are not borne out by indications at various shafts and mines controlled by the big company. The Allouez is increasing its ore reserves and doing much development work underground on the Kearsarge lode. No indication that the rock shipments are to be cut are to be seen. At the Red Jacket shaft it is said that the rock shipments are to be kept as large as possible and at the same time allow for the repairs that have been planned and are considered absolutely necessary there. The repair work at this shaft started the first of August and probably will continue a month and a half. At none of the other shafts are there any indications that the product is to be further curtailed at the present or any time during the summer. This leads local investors to believe that the mining company officials see more than a mere gleam of hope for a betterment of the copper metal situation in the near future. Buying of copper shares is a little stronger and the tone of the market of late has been much better than for some time past.

More encouragement is being met at the Seneca as the work of development continues. Nothing but disappointment was experienced for a long time but now some copper is being cut in the south drift at the third level. The shaft is down 920 feet and the station is being cut, preparatory to cross-cutting. The shaft is in the foot-wall of the Kearsarge lode.

At the New Arcadian the pit is down 16 feet in the overburden. This is being sunk to uncover the copper bearing amygdaloid cut some time ago during the diamond-drill operations.

By utilizing exhaust steam to drive a turbine electric generator, the Quincy Mining Co. hopes to secure 50 per cent. more power for the amount of fuel consumed, thereby reducing the operating costs. Waste steam from the No. 2 hoist will be employed and from one turbine the company hopes to secure about 500 additional horsepower, all without extra expense. The first turbine has been placed in operation and attracted considerable attention from mine superintendents from all parts of the district. Other turbines will be placed in operation at other parts of the mine, and also at the stamp mill. The power generated by the one turbine is believed to be sufficient to operate all underground electric locomotives, rock-house machinery, shops, and all places where electricity can now be used. The coal used in mining and stamping 1 ton of Lake Superior rock is estimated to cost from 14 to 24 cents, according to the hardness of the rock, the depth and haulage, and it is hoped to reduce the cost of coal from one-third to one-half by utilizing the exhaust steam. The idea is that of General Manager Lawton of the Quincy Mining Co., who braved the skepticism of mining men for about two years while he was planning the details. There is no doubt that the arrangement, which so far has worked all right, will help the mines of this district to economize.

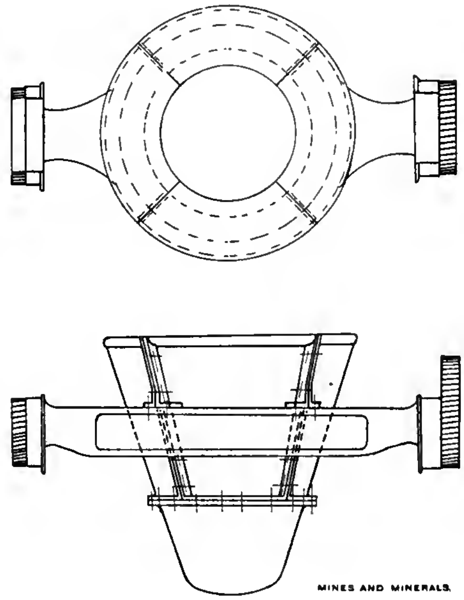
A notable change in policy is to be put into effect at the property of the Lake Copper Co. by the new management, which recently assumed control. Rock shipments will soon be stopped, it is said, and the entire attention of the management centered on development work, the opening up of large ground reserves, and proving up of the property so as to leave no doubt as to its future. The new management believes it will save money by preparing the property for operation on a large scale, and will inaugurate a systematic and broad-gauge plan of development work in every department of the mine's operations. The company's new compressor and its 50 drills will be in operation next month. No attention will be paid to production, but all efforts aimed at making the property one of the big and profitable producers of this region when sufficiently opened up.

SECTIONAL SLAG POT

Written for Mines and Minerals, by E. C. Reeder

Among all smelters handling furnace slag from the furnaces to the dump in large pots drawn by locomotives, there is always more or less trouble with the bowls of the pots from cracking of the material, which, of course, leads to breaks which destroy the usefulness of the pot. These breaks are usually patched up to prolong the life of the pot, but a new bowl is the final solution.

At the works of the Canadian Copper Co., at Copper Cliff, Ont., they use large pots which run on standard-gauge trucks and hold about 20 tons of slag. Originally these pots were made of cast iron in one piece, but after comparatively short service they went to pieces. Cast steel was then tried without much



better results. Finally a scheme to make the pot sectional was hit upon and it was worked out very successfully.

The bowl is made in five pieces, as shown in the sketch, the various sections being held together by bolts which pass through flanges on the edges of the sections. Asbestos gaskets are used to take up the inequalities of the castings which are not machined. All bolt holes are cored. In actual use the top sections crack first, but it is a very simple matter to take out the broken section and replace it with a new one.

With this form of construction the evils of pouring hot slag over a joint are done away with and the expansion of the bowl automatically adjusts itself without strain.

This device has been in use at the Canadian Copper Co.'s smelter for some 4 years or more, and all their slag cars are so equipped.

PROOF ASSAY FOR COPPER

In the proof assay for copper, instead of adding hydrochloric acid and filtering, it is possible to allow for the silver present, and subtract this amount from the result of titration. For every 100 ounces of silver present, subtract .10 per cent. copper. Thus for 600 ounces of silver per ton, subtract .6 per cent. copper. This simplifies the operation.

UTAH CONSOLIDATED AERIAL TRAMWAY

Written for Mines and Minerals, by Leroy A. Palmer

Method of Cheaply Transporting a Large Tonnage of Ore in a Rough Region

When the International Smelting and Refining Co. located its plant at the mouth of Pine cañon, at Tooele, 6 miles from the nearest railroad, the question of ore transportation became one of importance. In order to get supplies, the company built the Tooele Valley Railroad which serves very well the shippers from the south and west, but a large part of the ore supply is to come from Bingham, and this called for some special provision. While the plant is only a few miles over the mountain from Bingham, the country between the two places is impractical for railroad construction, and in order to ship by rail the ore must be hauled out of Bingham around the mountains to Garfield, over the Denver and Rio Grande, thence by the Salt Lake route to Tooele, and by the Tooele Valley to the smelter, a distance of 48 miles from the loading stations at Bingham, which are 1½ to 3 miles from the largest shippers. To overcome this roundabout and costly way of transportation two enterprises were projected, both of them to serve individual corporations, but one of which will doubtless serve other of the Bingham companies than that for which it is being constructed. One of these is the aerial tramway of the Utah Consolidated and the other is the tunnel of the Utah Metals Co. The Tooele Valley Railroad does not present any features which are not found in many ore roads but the other two projects are worthy of some consideration.

The new aerial tramway of the Utah Consolidated starts from a loading station near the mouth of the mine tunnel and ends about 500 feet from the sampler of the International

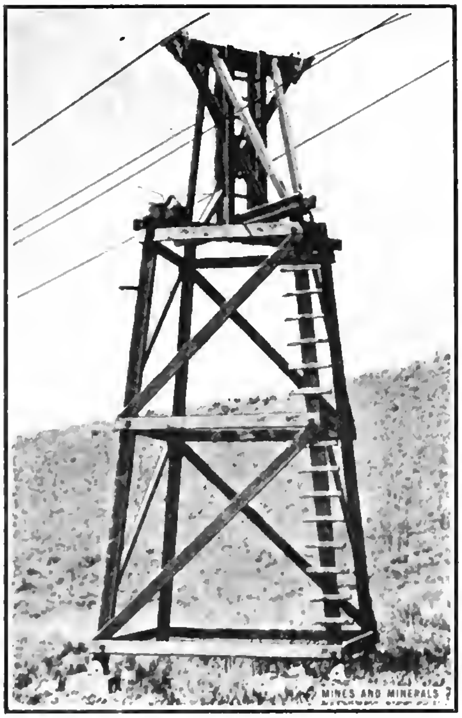


Fig. 1. Tramway Tower

smelter, a horizontal distance of 21,040 feet, slightly less than 4 miles. It is the Bleichert wire-rope tramway system of the Trenton Iron Co., and erected by the Utah Consolidated under the supervision of the Trenton Iron Co.'s engineers. The capacity is 100 tons per hour with a rope speed of 600 feet per minute. Leaving the loading station at an altitude of 6,824 feet the line goes directly up the side of a mountain to a tension station from which it crosses a gulch with its longest span, 1,145 feet, to control station No. 1 at an altitude of 8,250 feet, the highest point on the line, where the first traction rope ends. Thence down the side of a mountain to control station No. 2 in a gulch, where the second traction rope ends. From control station No. 2 it goes up a second mountain side, then down the opposite side and across a flat to the smelter, the ter-

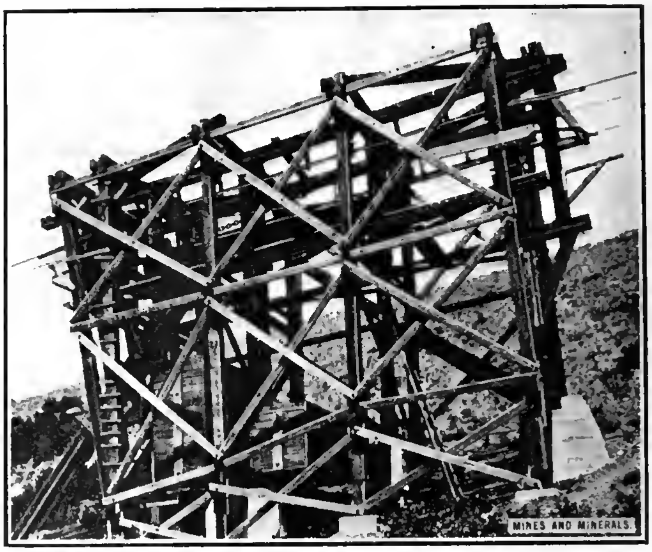


Fig. 2. Tension Station

minal at this point having an altitude of 5,507 feet. The rise to the summit is 1,426 feet in 4,400 and the drop is 2,743 feet in 16,600 so that the line beyond the summit not only has 1,300 more feet of drop than the portion to the summit but also carries 3¾ times as many loaded buckets which gives an excess of power which is utilized for doing work.

These figures will give some idea of the topography of the country, and when it is considered that the line runs directly over two mountains and through a heavy growth of scrub oak for a considerable portion of the way it will be understood that the construction was attended by considerable difficulty. A wagon road winds back and forth across the line as it goes up the first slope, but even so a considerable amount of packing was required. Over the mountain the only method of transportation by which the materials could be gotten into place was by means of burros, and all timber, cement, sand, gravel, and even water, was hauled in this way. For this reason the company found difficulty in letting a contract for the construction of this tramway and finally undertook the work themselves. Notwithstanding the fact that the cost of construction was high, the tramway was completed at less cost than the tramway company's engineers placed as a minimum.

The ore is hauled from the mine in trains drawn by an electric locomotive and is dumped to the bins from which it is unloaded through hand-operated rack-and-pinion gates to the buckets which have a volume of 8 cubic feet and a capacity of 1,280 pounds. Inside the loading and terminal stations, and all others where the traction rope is led around sheaves, the buckets are detached from the stationary cable and run along a track to the other cable or to the return side of the same cable, as the case may be. The detaching and attaching is done automatically by a board which lies alongside and engages a lug on the grip. The stationary cable on the load side is a 1½-inch patent lock-coil and on the empty side a 1-inch of the same type. In the lock-coil cable the two outer rows of wires are made with flat surfaces and the outer row cut so as to interlock. This gives a perfectly smooth surface which has proved very efficient in withstanding the continual wear to which it is subjected. The moving rope, which is $1\frac{1}{16}$-inch, 6-strand, 19-wire Lang-lay, cast-steel, passes

around an 8-foot terminal sheave mounted on a tension car connected by cable to a box containing 3,000 pounds of scrap iron, the tension at the sheave is 1,500 pounds as the weight is suspended from a pulley.

On the line there is a total of 78 towers, 7 tension stations, and 2 control stations. The towers, which are of the latest model adopted by the manufacturers, as shown in Fig. 1, vary

Fig. 3. Control Station No. 1

in height from 10 to 80 feet. They consist of 4 posts of 8"×8" timbers with a batter of 1 in 10 each way, so that the slant of the post is about 1 in 7. These posts are braced by 3"×8" timbers every 10 feet of height. The cap, which is 10 feet above the tower proper, is of 10"×12" timber substantially supported by posts and side braces, also by floor beams, front and rear braces. The towers are set so that the diagonal coincides with the center line of the system. Each post rests on concrete which has been carried to a firm foundation, at least to 4 feet unless solid rock was encountered at less depth. Bolts are embedded in the concrete of each pier and to these a square cast-iron socket is attached. The post sets in this socket and is securely bolted to it.

The tension stations are built of 10"×10" timbers set on concrete piers as in the case of the towers. The end posts are slanted so as to give stability and rest against buttresses on the piers. There is, perhaps, more timber in these stations than is really necessary, but the line is built for high duty and high speed and it is doubtless that any error in the design of these stations lies on the side of safety. One end of each rope is anchored to a deadman, and the other ends are attached to boxes which contain 42,000 pounds of rock on the loaded rope and 21,000 pounds on the empty rope. The boxes are shown in Fig. 2. The braces of the tension stations are of 3"×8" timbers.

One of the two control stations, Fig. 3, is situated at the highest point of the line and the other is in the bottom of a deep gulch beyond. That portion of the line from the loading station to control station No. 2 works almost in balance. A 100-horsepower General Electric induction motor at control station No. 1 is used to start the line, but once in operation, only 15 horsepower is required to maintain it, this amount being used in overcoming friction. The motor is belted to a pulley which is clutched to a 6-inch shaft. On this shaft is a pinion which drives an 8-foot gear mounted between two grip sheaves carrying the traction ropes. The sheaves are controlled by 10-inch strap brakes which are applied by wheels running on threaded reach rods.

Power is being taken temporarily from a stub line of the Telluride Power Co., but will be supplied by the International smelter when that plant is in operation. It comes from the line at 5,000 volts and is stepped down to 2,200 at control station No. 1 in three 37½ kilowatt General Electric transformers and part sent to control station No. 2 at the lower voltage.

Control station No. 2 is similar to No. 1 but is equipped so that the motor can be used as a generator. This portion of the line requires about 100 horsepower to start, but once in operation, the long drop on the farther side of the mountain develops 100 excess horsepower which is returned to control station No. 1 and to the mine, the only power drawn from the supply line being an amount sufficient to excite the field of the generator.

The lower terminal of the line, shown in Fig. 4, is about 500 feet from the sampler of the smelter. The general arrangement is similar to that of the loading station. The buckets are detached automatically and are dumped by hand to the bins from which the contents are unloaded to standard-gauge railroad cars and hauled to the sampler.

According to the annual reports of the company, shipment to the Garfield smelter by means of the old tramway and the Denver & Rio Grande Railroad cost in round numbers, 50 cents per ton, 8 to 10 cents on the tram and 40 cents on the railroad. The old line was something over 2 miles long, so, while it is too early to make any definite statements, it would seem that transportation by the new line should be accomplished for 15 cents per ton and perhaps less in time. The contract with the International smelter allows a maximum of 1,200 tons per day but it is probable that the Utah Consolidated, for the time being at least, will maintain its present output of 800 tons per day. They had a 40-cent per ton rate from Tooele to Garfield, and shipped that way, while trying out the tram, a sufficient amount of ore to close the contract with the Garfield smelter, which was on a tonnage basis. To ship from Bingham to Tooele by rail would require transfers at Garfield and Tooele, so that it is unlikely that a rail freight rate of less than 80 cents per ton could be obtained, to which must be added the 8 to 10 cents cost over the old tram. The tramway should thus effect a saving in freight of 75 cents per ton or $600 a day, practically $220,000 per year, and will thus return the cost of construction within a very short time.

The writer wishes to acknowledge the courtesy of J. B. Risque, general manager of the Utah Consolidated, and S. S. Webber, chief engineer of the Trenton Iron Co., who have assisted in the preparation of this article.

The farmers in Shasta County, Cal., have been so aggressive in their fight against smelter smoke that the different smelters have been compelled to meet the smoke problem in various ways.

The Mammoth Copper Co. has added a bag house to the Kennett smelter and will neutralize the gases by the process so successfully adopted at the Midvale smelter, Utah.

The Mountain Copper Co., an English corporation whose furnaces are at Keswick, have added an acid plant and entered into the manufacture of fertilizers.

Fig. 4. Tramway Terminal at Smelter

The Balaklala Co., at Coram, Cal., is installing the Cottrell process of smoke abatement, $ZnO+SO_3=ZnSO_4$, which was introduced at the Selby Smelting and Lead Co.

The Cottrell process of smoke abatement consists in precipitating solid particles in the smoke by electrical discharges.

LAKE SUPERIOR IRON ORE

A document filed for record in the mining region is a mortgage for $2,500,000, given by the Shenango Furnace Co. to the Pittsburg Trust Co., Pittsburg, Pa. The property covered by the transaction is the Webb Mine, near Hibbing, Mesabi range, and the mortgage is executed to secure the issuance of $2,500,000 in 5-per-cent. 20-year gold bonds, redeemable on any interest day after June 1, 1911. The bonds are dated June 1, 1910.

The purpose in giving the mortgage and issuing the bonds is to provide funds for the development of the Webb property. The Webb has been closed since the fall of 1908, although some ore was shipped during the season of 1908. The mine has been operated as an underground proposition, and it is now the intention to transform it into an open pit. The stripping will require a number of years and will cost, it is estimated, about $2,000,000. The Webb contains a large body of excellent ore, and once opened for steam-shovel mining the deposit can be taken out much more cheaply than under the present system.

The Syracuse Mine has been abandoned by Pickands, Mather & Co. While $6,000,000 worth of iron ore is contained in the property, it is impossible to mine it, and it rests securely under a deep, treacherous and shifting overburden, half water and half ground. The Syracuse is in the Lake Embarrass district of the Mesabi. A large sum of money was spent in a fruitless endeavor to open the deposit by the usual methods.

Pickands, Mather & Co.'s Hemlock Mine in Amasa, Menominee range, which has been closed down during the erection of a new shaft house and the installation of a new hoisting plant, has resumed operations. The shaft house has a total height of 85 feet and is built of Washington fir. The hoisting plant is of the Wellman-Seaver-Morgan Co. pattern. The duty is 5 tons from a depth of 2,000 feet.

The Jones furnace interests of Iron Mountain, who some time ago took over the old Kloman Mine, in the Republic district of the Marquette range, have decided to erect a large engine and boiler house and install a plant of machinery forthwith.

The Jones & Laughlin Steel Co., Pittsburg, has started the exploration of additional tracts of its Marquette range lands. Diamond drills are to be used. The tracts to be tested are in the Ishpeming district. The company's Lake Angeline Mine in Ishpeming, which is still a valuable property, is not the producer of high-grade ore that it once was, and it is hoped to supplement it with the development of new deposits

The Rolling Mill Mine, at Negaunee, was discovered by the company as a result of deep drilling operations some years ago, and is now becoming an important producer.

The Republic Iron and Steel Co. is restoring the Hartford Mine in Negaunee to the list of Marquette range producers. The new operators acquired the property in June, when a 10-year lease held by the United States Steel corporation expired. The ore in stock, and which is owned by the steel corporation, is being shipped at the rate of 10,000 tons a week.

Some notable improvement work is being carried on at the Zimmerman Mine, in the Spring Valley district of the Menominee range. The Spring Valley Iron Co., besides erecting two new engine and boiler houses, has under construction a commodious office building, two stories high. An entirely new plant of machinery has recently been installed. A new spur track to the property is being built by the Chicago and North-Western Railroad. The mine is controlled by Eugene Zimmermann, of Cincinnati, and his associates.

The Section Thirty Mining Co. is proceeding rapidly with the development of its property on the Vermilion range. The deposit, apparently, is extensive, and the ore is of the finest quality. The property is being given an excellent equipment. Ore is being hoisted from two shafts. The product is being mined almost entirely as a feature incidental to the development work, notwithstanding which, fully 50,000 tons will be shipped the present season. The ore is going to the docks at Two Harbors over the Steel corporation's Duluth and Iron Range road. It has cost upward of $500,000 to demonstrate the existence of high-grade ore in quantity in the Section Thirty, and this only after various operators had made a failure of the effort. The mine is being developed now by George A. St. Clair, mining engineer and geologist, associated with Alfred Merritt, of Duluth. The developments at Section Thirty have greatly stimulated exploratory operations on the Vermilion. Various properties are being drilled or test pitted, and in some cases shafts are being sunk. It is the claim that new ore bodies are being found.

Among other properties, the conditions at the North American, Vermilion Steel and Iron, and Vermilion Steel Extension, are reported satisfactory.

The Vermilion Iron Development Co. is sinking a shaft on Pine Island. The formation here is about 200 feet in width and on the surface contains a fair grade of ore mixed with jasper. The shaft will be sunk 500 feet.

SLAGS

The furnace man aims to make a mixture of ore and fluxes which will form a fluid slag, as thick or pasty slags prevent a thorough separation of the metal and much will be lost in such slags. Since acid ores are in excess in the precious metal mines, the slags are made silicates by the addition of lime or iron oxides in sufficient quantities to make the silica run easily.

Slags are classified when they are silicates according to the ratio of oxygen in the base R to the oxygen in the silica (R stands for any basic oxide).

The fusibility of a slag depends upon the character of the bases and upon the percentage of silica it contains. The silicates of lime and alumina are the least fusible. Slags of the composition $CaO \cdot SiO_2$ and $4CaO \cdot 3SiO_2$ are fusible, but the latter is not as readily fusible as the former. It requires a little over 1.6568 times as much limestone, $CaCO_3$, as silica to produce a slag of the composition $CaO \cdot SiO_2$.

Name	Slag	Oxygen Ratio
Subsilicate	$4R_2O \cdot SiO_2$	2 : 1
Singulosilicate	$2R_2O \cdot SiO_2$	1 : 1
Bisilicate	$R_2O \cdot SiO_2$	1 : 2
Trisilicate	$2R_2O \cdot 3SiO_2$	1 : 3
Sesquisilicate	$4R_2O \cdot 3SiO_2$	2 : 3

2. Another system of naming slags is according to the ratio of FeO to CaO. Thus, a slag containing 36 per cent. FeO and 24 per cent. CaO would be termed 1½ : 1 slag.

Slags, Typical Lead

	Per Cent. in Slag			
	SiO_2	FeO	CaO	ZnO
A	35	28	28	
B	34	34	24	
C	34	34	17	7
D	30	40	20	
E	30	48	12	
F	28 to 30	54	6	

A, is adapted to ores carrying Al_2O_3.
B, one of the best fusible slags.
C, favorite with high Zn in charge.
D, an excellent slag known as the one-half slag.
E, an excellent slag, known as the one-fourth slag.
F, not as good as the others.

The sum of the SiO_2, FeO, and CaO is considered as making up 90 per cent. of the slag constituents, except when the ores contain much ZnO and Al_2O_3.

THE EVOLUTION OF HOISTING

Written for Mines and Minerals, by E. B. W.

Raising mineral products is a problem, old as mining, which has been solved in many ways by succeeding generations according to the exigency of the case. Naturally in semibarbarous times the demand for coal and ore was not so great as to require the engineers of those times to devise means for hoisting from great depths; but the engineers of today must take off their hats and acknowledge that the engineers of the past solved their problems properly.

Some Ancient Methods Showing the Ideas from Which Modern Machines Have Been Derived

If it were possible to avoid hoisting by driving adits they did so, even in barren rock, without powder, using fire and water and their rude tools to make the advance. It could be understood readily from present-day practice had not Agricola, who wrote in 1550, stated, that hoisting was one of the greatest obstacles to economic mining.

Near Guanecevi Durango, Mexico, there is an ancient mine opening with stone steps leading to a subterranean excavation, which tradition states was worked in the seventeenth century by Spaniards who used Aztecs as miners. The opening is inclined, and up this inclination the "carilleros" brought baskets of ore.

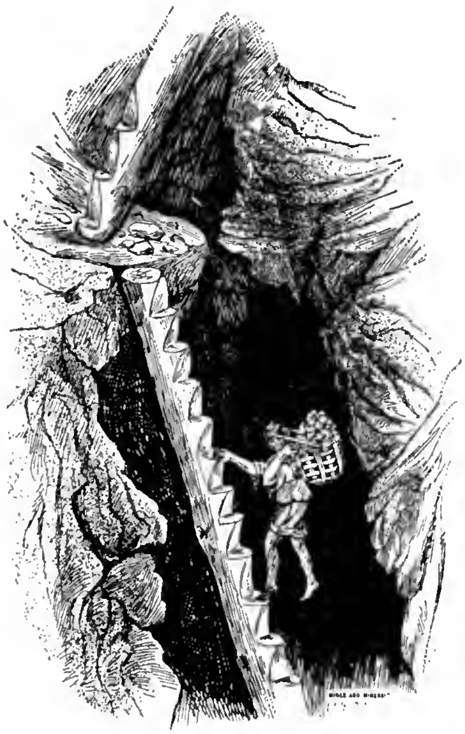


FIG. 1. NOTCHED POLE LADDER

Near the Ojuela Mine, at Mapimi, Mexico, mining is carried on in a small way, and the hoisting is done by carrying baskets of ore up steps cut in the limestone. The contrast between the old and the new way is here made impressive, because of the excellent hoisting apparatus of the Ojuela Mine.

In a number of places in Mexico where mining is carried on in a small way, the shafts are supplied with logs in which alternate notches have been cut for the feet. Ore is carried up these poles, which the English-speaking scoffers term "chicken ladders," in bags or baskets. One of these poles was to be seen protruding from the shaft in the illustration of the Mexican ore carriers mentioned in the "Evolution of Mine Haulage."* The illustration, Fig. 1, of this article is no exaggeration, besides this method of hoisting is not uncommon in other countries as well as Mexico, hence the past and present can be linked without fear of contradiction in this cameraean age.

Until the middle of the eighteenth century, in Great Britain men and women were worked under laws which practically made them slaves. Children, as soon as able, were sent into the mine to help their parents, in fact were born miners, whose bodies belonged to the operators. In 1373 Bishop Hatfield constituted John de Belgrave and Nicholas Cook commissioners to seize workmen and coal bearers for the mines of Whickham and Gateshead, wherever they could be found within his royal liberty, with full power to imprison and otherwise punish them should they prove rebellious and obstinate.† Fuller states in his "Worthies of England," that in 1296, 360 miners were impressed out of the Peak of Derby and Wales for the King's silver mines in Devon. Again in 1526 it was proposed to Henry VIII to levy men for the same mine. Queen Elizabeth gave power to impress workmen, wagons and horses to carry on mining in 1580. In the following century Lord Bacon advocated the employment of felons in recovering and working abandoned metalliferous mines; thus holding out a prospect of great treasures being obtained from "a promiscuous chaos of drowned minerals and condemned men." The Tennessee Coal and Iron Co. employs convicts in coal mines of Alabama, so that at present the states of Tennessee and Alabama are about in the position of enlightenment that England was in 1600, or the middle ages. The services of bondmen being available, the bulk of all kinds of work was no doubt performed by manual labor, and the "bearing system" continued well into the eighteenth century. From 1706 to 1842 mine workers hired out for a whole year under conditions and laws which practically made them slaves. Able-bodied men were paid 20 cents per day for mining and the women who carried the coal out of the mine received 6 cents per day. Lord Ashley made such a stir over the living and working conditions of miners that a Royal Commission was appointed to examine into the truth of his statements that boys and girls had to work 16 and 18 hours and never saw daylight except Sunday. The commissioners evidently found Lord Ashley's statements conservative for their report changed the complexion of coal mining in England, and from it Fig. 2 has been copied. One of the claims made by Lord Ashley was that men, women, and children who worked in the coal mines were subjected to unnecessary dangers and hardships, besides the remuneration they received was not enough to supply their demand for food. This illustration and several others appeared in the Royal Commissioners report and is therefore authentic.

FIG. 2. WOMEN CARRYING COAL IN 1842

Jonathan Bartley, of Trenton, N. J., in writing on "Graphite Mining in Ceylon," in 1909, uses Fig. 3 as an illustration of the method of hoisting in that country. He says: "The method of mining is the simple hand process with pick and shovel. Steam or air drills are unknown. As a rule shafts are sunk to water level, which usually is about 60 feet below the surface. The method of entering or leaving a mine is by means of one of two kinds of ladders. In soft soil where it is necessary to 'crib,' strong vines are cut and woven into a ladder such as shown in Fig. 3. The natives go up these ladders with alacrity

* June, 1910, MINES AND MINERALS, page 683, Fig. 2.

† Annals of Coal Mining, R. L. Galloway, page 48.

carrying bags or baskets of graphite on their shoulders. Where the rock is firm and hard, holes are drilled in the walls of the shaft into which are driven round sticks of wood. These are placed about 12 inches apart and extend about 14 inches from the wall into the shaft."

Mining machinery with men to set it up was sent to these mines in Ceylon, but nothing could be accomplished as the natives broke the machinery as fast as it was put in place. The objection to machinery seems to be an inherent quality of miners, whatever their nationality, yet the growth and development of the machine system in mining has been sure, and no attempt to resist it will be successful. The machine is a labor-saving device which lightens the miner's work, increases the output of the mines, and does not decrease wages. At some of the graphite mines in Ceylon a rope and windlass are used for hoisting, this, however, is no innovation, since it was used from time immemorial. Agricola speaks of the windlass being used in Ger-

Fig. 3. Shaft Mining in Ceylon

many. Owens "History of Pembrokeshire," which is supposed to have been written in 1602, states: "They used not engines for lifting up the coles out of the pitt, but made theire entrance slope, soe as the people carried the coals uppon theire backes along stayers, which they call landwayes; whereas nowe they sinke theire pittes downe right foure square, about 6 or 7 foote square, and with a wyndles turned by foure men, they drawe up the coles a barrel full at once by a rope."* Pits were sunk from 70 to 120 feet deep, and were drained wherever possible by water levels or adits which Owens remarks are "very chargeable."

While the windlass is the oldest mechanical hoisting apparatus known, it is still in use and will continue to be until prospecting ceases. Frequently it may be found in out-of-the-way places, constructed entirely of wood, yet nevertheless portable. Fig. 8 shows a strong-arm hoister constructed in the twentieth century. The barrel on which the rope coils is about 10 inches in diameter and is supplied with a strap brake, shown to the left of the barrel. With one man on each handle the capacity of a strong-arm hoister is 4 tons per day of 10 hours from a depth of 100 feet. In sinking winzes in ore mines, the windlass is almost invariably adopted and there are few ore-mining men who have not had the pleasure of being hoisted in a strong-arm hoister bucket which would insist on turning completely around a few times to permit of their being scratched before they reached the top.

The crab winch is an improvement, in that one man can safely handle as heavy a load as two with a windlass, but the fixed principle in mechanics that what is gained in power is lost in time prevents increased output over the windlass. The hand winch, Fig. 4, will find many uses about a mine where heavy bodies are to be lifted from time to time, and the purchase of one will not be regretted. It is probable that when animals were first used for hoisting they were attached to a rope passed over a pulley, and by means of their dead weight pulled the load up the shaft. The ancients probably found as many objectionable features to this way of hoisting from a shaft as the present generation of miners, for it seems to have been entirely overlooked in early reports, which again makes us respect our progenitors. It is presumed that the equivalent of the horse whim with several sweeps was introduced by the Egyptians; however, instead of horses, slaves were used on the sweeps.

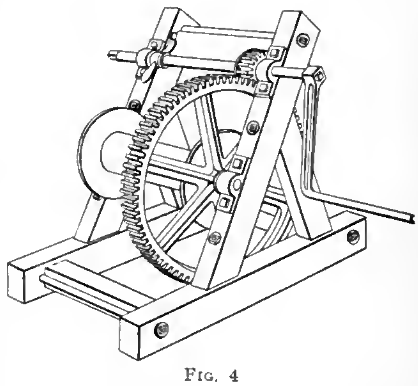


Fig. 4

From Court Records,* dated 1667, the style of horse whim, known as horse-gin,† is described, and Galloway has it illustrated, as shown in Fig. 5. It was termed a "cog-and-rung gin," and is the earliest known form of horse engine used in drawing coals.‡ The drum was placed horizontally over the shaft, as the windlass had been, but was more elaborate and of better construction, besides, as may be seen from the cut, it had a horizontal rung wheel at one end. The cog wheel, which was arranged horizontally on a vertical shaft meshed with the rung wheel and when revolved by the lever or sweep drawn by horses caused the drum to wind or unwind. It will be noted that the hoisting

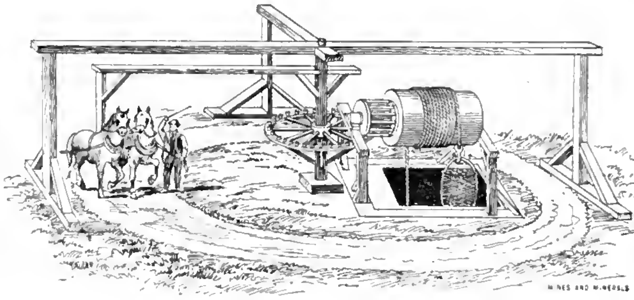


Fig. 5. Cog-and-Rung Gin, 17th Century

is done in balance and that as one bucket is drawn from the pit the other bucket descends, thus practically leaving only the weight of the coal to be raised by the horses. If one looks at the details in this engine of the seventeenth century he will find various mechanical devices which now, even if elaborated and improved, are still the same old sixpence. Galloway says that "on account of the position of the drums directly over the shafts they were frequently wrecked by colliery explosions."

*Annals of Coal Mining, R. L. Galloway, page 120.

* Transactions North of England Institute, Vol. XV, page 269.

† Contraction for engine.

‡ Annals of Coal Mining, R. L. Galloway, London, 1898, page 168.

It is possible that on this account the "cog-and-rung" gin was soon replaced by the vertical gin shown in Fig. 9, although there are other advantages in favor of the latter which the ancients learned from experience. It is interesting in following up the work of ancient Spanish miners, and the work of native miners six generations ago, to notice that their ideas of mining and the general fitness of things were equal to the miners of the present; in other words, our generation of miners are merely elaborating on the experience of their predecessors.

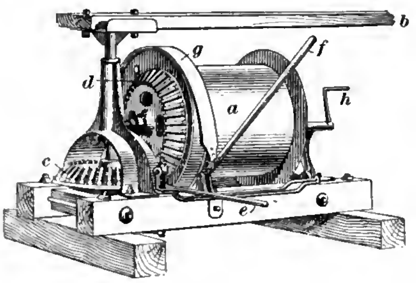


Fig. 6. Geared Horse Whim

The advantages of the "whim gin" in Fig. 7 are that the barrel is not placed directly over the shaft to interfere with the handling of the buckets, and it is not badly injured in case of an explosion. The "whim gin" shown can be fitted to hoist in balance, although this particular product of the twentieth century was not, while that shown in Fig. 7, which belonged to the seventeenth century and was used at a coal mine in England, was so arranged. It has been stated that two men working with a windlass could raise 4 tons from a depth of 100 feet. With an animal pulling over a loose wheel about 10 tons can be raised 100 feet in a day. With the horse whim in Fig. 9, 200 pounds can be raised 175 feet in 4 minutes; however, to this must be added the time for delays, so that it is not probable that more than 20 tons daily can be hoisted by this affair. With the "whim gin" in Fig. 7 a good day's work was about 90 tons of coal from a depth of 360 feet. It will be noticed that the drum is large, and the records from which it was taken state that eight horses were required to work the gin in four relays of two at a time, or, if it were necessary, four at a time. The hoisting ropes were of hemp; the drum large; while the sweeps were attached directly to the axle of the drum where the horses could pull naturally from a whiffletree, and their number was increased as required. In lowering a bucket it was necessary to reverse the direction in which the horses traveled. With proper arrangements this could be quickly accomplished, although later some one with less mechanical skill than the ancients used a swiveled whiffletree that was suspended from the ends of a

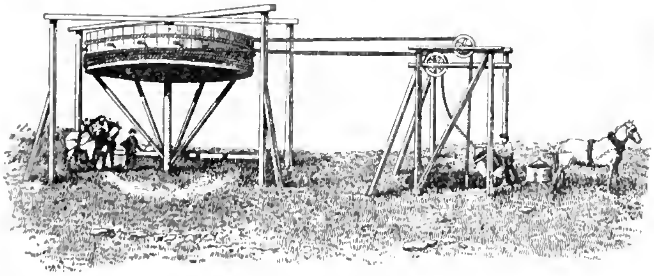


Fig. 7. Whim Gin of 17th Century

sweep above the horse. This latter arrangement greatly reduced the tractive effort of the animal, and the size of the drum, and was not adopted generally. At some mines the drums for the horse gins were housed; at other places the drum protruded above the roof of the shed which covered the teams and drivers in inclement weather. When the friction due to the rigidity of the rope and that of the axle and other sources are considered, a strong horse cannot raise more than 2,500 pounds 1 foot high in 1 minute with a whim and keep it up for any length of time. If this work be taken as the basis of foot-pounds, the power of any similar machine can be readily calculated.

In recent years the geared horse whim and horsepower have been introduced for hoisting purposes. These machines are great improvements over the old style whim gins so far mentioned; for instance, sometimes a horsepower similar to that shown in Fig. 6 has two gears, one which acts slowly for heavy lifts, and a faster one for lighter lifts.

These hoisters are well known owing to their being used so extensively by railroad contractors. They are very serviceable in prospecting and at small mines, for they are capable of raising 100 tons from a depth of 200 feet with one horse in 10 hours. The working parts of Fig. 6, with the exception of the bedplate *e* and the sweep *b*, are of iron. The machine shown has but one set of gears *c* and *d*, which are thrown in by the lever *e*

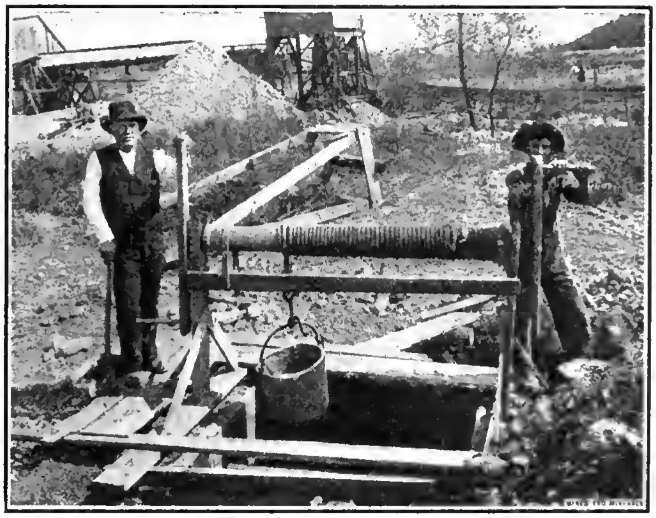


Fig. 8. The Strong-Arm Hoister

Fig. 9. Modern Vertical Whim Gin

when hoisting is to be done, or out when it is desired to lower the load. Wire rope of $\frac{1}{2}$- or $\frac{5}{8}$-inch diameter is generally used on the drum *a*, and in a pinch $\frac{3}{4}$-inch rope may be used although the rope winds stiffly. When lowering a load by gravity the brake lever *f* is used to draw the brake band *g* tight on the brake shoe of the drum. In case it is necessary the rope can be coiled on the drum by the handle *h* independent of the gears. The crank is also supplied with a dog and ratchet to prevent the rope running back when power is removed from the handle. Small loads can also be raised by the use of the crank *h*. It is possible to hoist 2,000 pounds 26 feet per minute with the slow speed, or 800 pounds 55 feet per minute with high gear. Horsepower hoisters may be used to advantage in places where prospecting, shaft sinking, and mill building, is going on, also in mining districts where fuel, water, or both, are scarce.

For some years the horse whim, shown in Fig. 10, was a favorite in the West on account of its portability and other good points. The driving gear of this machine is connected by a shaft which turns a drum, not shown, but located below ground at one side of the shaft. On the drum, a hemp or wire rope is wound, one end of which passes over the pulley on the head, frame. In this case the head-frame is more elaborate than usual, a tripod answering in most cases; however, this is not a prospect hole but a small zinc-ore mine being worked by a farmer in Missouri, between crops. It will be noticed that the bucket is landed on a small truck, which is on the platform that is the continuation of a trestle. In many cases zinc mining in Missouri is a side issue to farming that keeps the hired men busy, exercises the horses; besides "every little bit helps."

As early as 1550, at which time Georgius Agricola wrote his excellent treatise, "De Re Metallica," waterwheels were in use at German ore mines to raise buckets fastened on an endless belt which turned about a shaft and went down in the mine. The illustration, Fig. 12, is taken from Agricola, and one similar in arrangement, was erected early in 1700 for raising coal from Alloa collieries in Scotland. In 1774 a similar waterwheel hoist was constructed in Newcastle, and at the same date Smeaton erected a waterwheel for hoisting both coal and water at Griff colliery in Warwickshire.* In 1777 Smeaton designed a single bucket "water coal gin" for the Long Benton colliery. The drum motion was accomplished by means of gearing. This waterwheel gin was found to be a great improvement over the horse gin of that time, for it hoisted a basket containing 6.5 hundredweight of coal from a depth of 492 feet in 2 minutes. It is said to have accomplished the work of 16 horses and 4 men with a "whim gin."

Fig. 10. Horse Whim for Hoisting

When water was available for hoisting it was carried in flumes to the pit mouth, where of course the wheel must be situated. Such conditions necessarily limited the number of waterwheels for hoisting, nevertheless, improvements were made so that they could be quickly reversed, without the use of the gearing required in Smeaton's Griff colliery waterwheel. The simplicity of the double waterwheel, shown in Fig. 11, where it was merely necessary for the operator to move a lever attached to a gate to regulate and make the waterwheel move to the right or left, led to its general adoption wherever conditions were favorable. Those erected in 1782 in the Tyne district were abandoned in 1808, although one was in use at the Leafield colliery in the Wear district, till 1812. Even if the rise of the waterwheel gin to prominence was slow, its fall was speedy, owing to the introduction of the steam engine for hoisting purposes, in about the year 1780, when the crank was first added to furnish a rotary motion. Some of these antiquated and interesting old engines are still said to be at work in South Staffordshire.

Mr. Curr* writes in 1797 that the most ancient hoisting machine he has knowledge of was invented by Menzey, "but there are few situations that afford the requisites necessary to that invention. A stream of water with a fall of about half the depth of the pit is necessary, if any business of consequence is to be done. Its construction consists of two rope wheels fixed on one horizontal axis, which are so proportional to the depths of the water pit and coal pit as to reach separate depths of the pits by the same revolutions; and the power applied is a tub of water large enough to overbalance the weight to be drawn."

In 1891 there was a somewhat similar arrangement at Sterling Furnace, Rockland County, New York. The construction of the apparatus is shown in Fig. 13. The drum *a* is assumed to be 10 feet in diameter, the drum *b* 1½ feet in diameter, therefore, with one revolution of *b* the circumference of drum *a* will have traveled 31.416 feet. The drum *a* had cage and load to lift; from the drum *b*, which was on the same shaft *c*, a large iron tank *d* was suspended at such a height that water from a sluice *e* could quickly fill it. Assuming that the tank *d* descended to *d'*, a distance of 16 feet, or 3.44 turns of the drum *b*, then the drum *a* would raise its load 108 feet. When the tank *d* arrived at *d'* the valves *f* in the bottom were lifted and the water load discharged. The weight of the cage and empty barrow, acting on the long arm of the lever or the circumference of the drum *a*, would raise the empty tank to the position at *d* when it would be again filled with water. This machine worked satisfactorily many years at Sterling furnace. In the book termed "Fossil Fuels," a somewhat similar machine is described as follows: "The vicinity of a pit nearby sunk for working, happening to include an old shaft heavily watered from near the top, an axle with drums was placed across; to one of these was suspended by a rope a large tub, and from the other a rope was carried over the head-wheels at the adjacent pit, the tub being at the top of one pit while the basket of coal was at the bottom of the other and vice versa. The tub thus suspended near the spring is filled with water, and on the ringing of a bell, the signal for hoisting, a catch is let go by the banksman, who, pressing upon

* Farey's Steam Engine, page 297.

* The Story of American Coals, W. J. Nicolls, page 202.

a brake to regulate the velocity of the machinery, allows the tub to descend to the bottom, where a valve is opened and the water flows out. The empty corfe or basket is so much heavier than the empty tub that the former descends while the latter is brought up and secured in the situation first described."

(*To be continued*)

A SIMPLE HYDRAULIC HOIST

Written for Mines and Minerals

The auriferous gravels near Fairplay, Colo., are apparently of glacial origin, with some evidences of secondary river sorting. About 10 per cent. of the mass of material to be removed consists of boulders varying between 100 and 1,000 pounds in weight, and cannot readily be handled by ordinary means.

To rapidly and cheaply dispose of this large amount of bulky material, Mr. Jas. McChotka, Superintendent of the Platte River Placer Co., at Fairplay, makes use of a simple and efficient hydraulicly operated hoist.

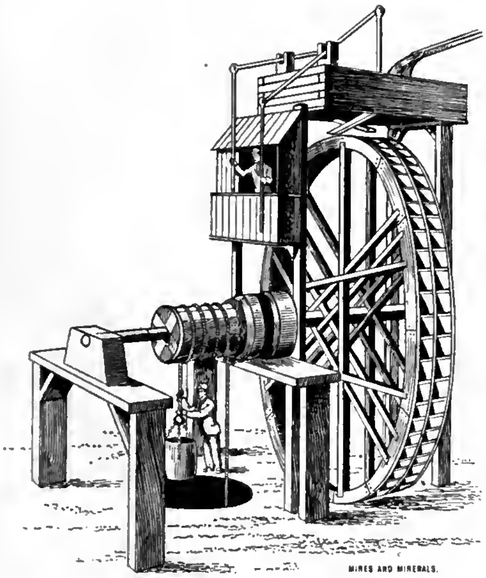


Fig. 11. Double Waterwheel Hoist

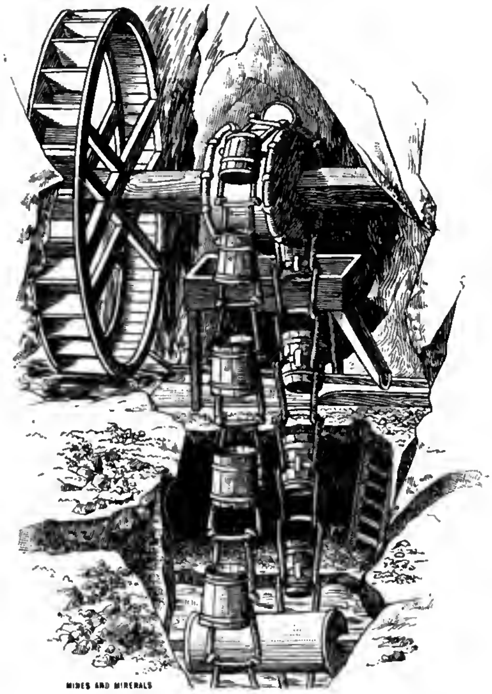


Fig. 12. Ancient Bucket Hoist

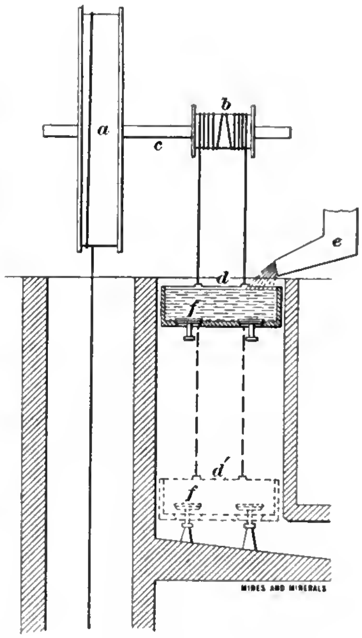


Fig. 13. Water Tank Hoist

The derrick is of the ordinary boom type provided with a detaching hook, etc. The hoisting rope leads from the foot of the mast to a spindle or drum mounted on one end of a horizontal axle, on the other end of which is fixed an impulse wheel 24 inches in diameter.

The main supply pipe line is tapped at some convenient point and the water conveyed to the hoist where it discharges through a giant, the nozzle of which has been reduced to 2 inches diameter. When not in operation the water is allowed to discharge through a wooden box beneath the wheel. After the rock to be moved is slung in chains, the operator depresses a pedal with his left foot raising the nozzle so that the water impinges upon the blades of the wheel, the drum being then brought into gear by means of a horizontal lever, actuating a pinion wheel having a ratio of 1 to 10. When the rock reaches the required height, raising the foot from the pedal allows the nozzle to drop by its own weight, the water then discharging through the box as noted above. After the water is shut off from the wheel, the drum is controlled by a brake brought into play by a pedal depressed by the right foot of the operator.

Water is supplied under a 280-foot head, or at a pressure of about 120 pounds to the square inch. Four men are required to run the hoist, one at the drum, one to release the detaching hook, and two to sling in chains the rocks to be removed. As the hoist is in use but two or three hours daily, a valve to entirely shut off the water supply is inserted in the pipe between the main supply line and the nozzle. This hoist is capable of handling eight to ten rocks of average size in 10 minutes and can be built by any blacksmith or millwright.

MINERALS OF GREECE

The mines of Greece are found chiefly on the eastern shores of the peninsula and in the islands of the Ægean, as these parts of the country are the oldest geologically, and of volcanic origin. Notwithstanding this mineral wealth, the industrial development of the country has been slow, on account of the lack of fuel, which is nowhere found in sufficient quantity. For this reason most of the minerals are exported in a crude state after the simplest possible preparation, and the exploiting only of those minerals has been found to be profitable which are valuable without being bulky and are easily mined and transported.

The following table gives the mineral exports from Greece for 1907:

Classification	Tons	Value
Iron ore	664,071	$744,529
Manganese iron	94,685	147,705
Lead ore	5,538	23,480
Argentiferous lead	9,235	623,824
Zinc ore	25,470	46,360
Manganese	9,788	58,170
Chromium	12,430	92,582
Coal	4,525	17,244
Lead	1,987	7,593

But for the lack of coal Greece would have been an important steel and iron producing country. Studies are being made as to the possibility of using lignite for smelting purposes, of which considerable quantities are found.

The iron ores are hematite, generally of high grade, and manganese in veins piercing limestone or volcanic layers. Often, as at Laurium, they appear as coverings for beds of lead, zinc, and copper.—*United States Consular Report.*

PETROLEUM INDUSTRY, VENEZUELA

Consul Ralph J. Totten, of Maracaibo, submits the following on petroleum deposits in Venezuela, the development of which will have great influence on the import trade:

There are five known petroleum deposits in this consular district. Oozings of petroleum, covering a considerable territory, are found in the district of Mara, near the Limon River asphalt lake. Oil has been located at Bella Vista, near the city of Maracaibo, and wells are to be sunk by the owners in the near future. Their object is to refine the product and to supply the local demand for illuminating and lubricating oils. Evidence of the existence of petroleum is found over a large area in the district of Sucre in conjunction with asphalt deposits. An oil field on the Sardinate River in Colombia, near the Venezuelan frontier, is being worked at the present time. The oil is refined at the wells and is sold in the near-by Colombian cities. The fifth field, and the one that seems to be the largest and most conveniently situated, is south of Lake Maracaibo in the district of Colon, state of Zulia.

A company of Maracaibo business men, who have control of these fields, is preparing to make an active campaign to interest foreign or domestic capital in the exploitation of its property. It has an advantageous contract with the Venezuelan government, which grants it free entrance for its drills, machinery, and supplies.

These oil fields can be reached by light-draft lake and river steamers. Passing up the Catumba River to its junction with the Tarra River, about 30 miles beyond Encontrados, the capital of the district, then about 50 miles through the latter river, brings one to the village of La Paloma. The oil-bearing land commences at this point and continues nearly to the Colombian frontier, including an immense area.

Abundant evidence of the presence of petroleum is to be found close at hand on some rugged hills 40 to 50 meters above the level of the river. From sources on these hills run some 20 small streams, the waters of which are constantly covered with a thick coating of petroleum. Some of the oil comes from the springs with the water and some from fissures along the banks of the streams. In one of these fissures an excavation was made 3 feet deep by 2 feet square which in almost 6 hours filled with crude oil. In this locality were found places where the surface of the earth was covered with a deposit of tar from oil that had been evaporated by the heat of the sun, sufficiently thick to destroy vegetation of all kinds.

Crude oil of two classes is found. One kind is thin enough to flow readily, having a specific gravity of .8837 at 15° C.; the other is very thick and of the color and consistency of coal tar. Both varieties show an asphaltic base, in this resembling the Texas crude oils as distinguished from oils having a paraffin base. A distillation test of the thin oil gave the following results: Between 0° C. and 120° C., .5 per cent.; between 120 degrees and 170 degrees, .5 per cent.; between 170 degrees and 235 degrees, 14 per cent. (illuminating); between 235 degrees and 271 degrees, 28 per cent. (illuminating); between 270 degrees and 370 degrees, 51 per cent. (lubricating); coke, 6 per cent. The product between 170 degrees and 235 degrees flashes at 62° C., and that between 235 degrees and 270 degrees at 83° C. The viscosity of the lubricating oil is twice that of water.

The thick oil gave the following results: Water, 28 per cent.; between 0° C. and 310° C., none; between 310 degrees and 370 degrees, 61 per cent. (lubricating); coke, 11 per cent.

The forests in this region contain great quantities of construction wood of all kinds, and, although the contract with the government permits the free introduction of all materials and machinery for the exploitation of the property, they would be of great utility in the construction of buildings, derricks, tanks, and casks. The land is sufficiently sloping for the laying of pipe lines to the river port La Paloma, where it is intended to place the refinery and tanks. Abundant wood can be had for fuel and there are coal deposits close at hand.

Venezuela alone consumes annually about 1,000,000 gallons of kerosene. The price here is seldom lower than 42 cents per gallon and often reaches 50 cents.

The location of these fields is especially favorable, having all-water connection with the city of Maracaibo and the other lake ports, as well as with the city of Encontrados, on the Catumba River. Encontrados is a city of 25,000 inhabitants and is the terminus of the Tachira Railway, which connects it with the Province of Santanner, of the Republic of Colombia. Oil could thus be shipped to all of the Venezuelan and Colombian seaports and to the interior of both republics by rail and water combined.

CALCULATING THE TONNAGE OF ORE PILES

Written for Mines and Minerals, by R. T. Hancock

The following tables are intended for calculating the weight of ore in a pile or bin whose contents in cubic yards or meters has been measured, and are based on the assumption that broken ore occupies 75 per cent. more space than it did in the solid, which is approximately true for run-of-mine, though screened large may occupy only 60 per cent., and small free from lump, up to 80 per cent. and over.

Specific Gravity	Weight Per Cubic Yard		Weight Per Cubic Meter	
	Long Ton 2,240 Pounds	Short Ton 2,000 Pounds	Long Ton 2,240 Pounds	Metric Tons 2,204 Pounds
1	.42984	.48143	.56223	.57143
2	.85968	.96286	1.12446	1.14286
3	1.28952	1.44429	1.68669	1.71429
4	1.71936	1.92572	2.24892	2.28572
5	2.14920	2.40715	2.81115	2.85715
6	2.57906	2.88858	3.37338	3.42858
7	3.00888	3.37001	3.93561	4.00001
8	3.43872	3.85144	4.49784	4.57144
9	3.86856	4.33287	5.06007	5.14287

The specific gravity used should of course be that of an actual sample of the ore in question, and not that of its principal mineral constituent as given by the textbooks.

In the absence of more elaborate apparatus the specific gravity may be approximately determined in the following way:

A weighed quantity, say 50 grams of the ore, is placed in a dry 100-cubic-centimeter measuring flask, which is then filled up to the mark with water from a burette, pausing half way to shake up and dislodge any air bubbles entangled in the powdered ore. The difference between the quantity delivered by the burette and 100 cubic centimeters, divided into the weight of ore taken, will give the specific gravity.

Example	Grams
Weight of ore taken	50.0
Cubic centimeters delivered	87.3
Cubic centimeters by difference	12.7

50 ÷ 12.7 = 3.937 = specific gravity of ore

Weight Per Cubic Meter of Ore Pile (Long Tons)

	Long Tons
3.000	= 1.68669
.900	.50600
.030	.01687
.007	.00393
3.937	= 2.21349

GASOLINE LOCOMOTIVE

The only gasoline locomotive in the coal mines of the United States is at the Midvalley Coal Co. mines, Wilburton, Columbia County, Pennsylvania. The locomotive's speed is 3 miles per hour for low gear and 6 miles for high gear. On low gear the locomotive pulls eight mine cars, about 32 tons, and has replaced five mules and one steam locomotive. Mr. H. D. Kostenbander, superintendent, writes that it consumes on an average 20 gallons of naptha per day of 10 hours, works very satisfactorily, and that it is considered very cheap haulage.

EXPLOSIVES FOR TUNNEL DRIVING

Written for Mines and Minerals

Importance of Using Explosives That Are Suited to the Conditions

The selection of explosives for tunnel blasting, probably requires a more careful study of conditions than for any other kind of excavating. Maximum speed in driving cannot be attained unless the explosive best adapted to the work is used. When starting a tunnel or drift, it is a good plan to thoroughly try out several explosives, which are distinctly different in action, before finally adopting any one of them. The results, however, from this preliminary trial will be of little or no value, unless each different explosive is used under exactly the same conditions. Care must be taken to see that no change occurs in the character of the rock, number and direction of the bore holes, strength of the detonator, kind and quantity of tamping, amount of water encountered, method of connecting up the bore holes for firing, and that the explosive is always thoroughly thawed. If a material change in any of these conditions occurs as the work progresses, further tests should be made to determine whether a quicker or slower, a stronger or weaker, explosive might not break the ground, or bottom the bore holes better, or make it possible to bring out the cut with fewer holes or deeper ones. The speed at which rock can be drilled, does not indicate how it will break, and not infrequently that which can be easily drilled is very difficult to blast.

High explosives suitable for tunnel blasting, should not give off objectional fumes on detonation, and accordingly gelatin dynamite, blasting gelatin, or ammonia dynamite should always be selected.

Gelatin dynamite is made in various grades of strength, from 25 to 80 per cent., inclusive. It is comparatively slow in action, the higher grades being little, if any, quicker than the lower ones.

Blasting gelatin is manufactured in only one strength, which for comparative purposes may be said to be 100 per cent. It is more powerful and quicker acting than any other blasting explosive. It should be used sparingly, therefore, until the maximum safe charge has been learned from experience. Good results will often be had in hard ground, if a few cartridges of blasting gelatin are used in the point of the bore hole, with gelatin dynamite on top. When this is done, it is best to put the detonator in one of the cartridges of blasting gelatin.

Ammonia dynamite is made from 25 per cent. to 75 per cent. strength. All grades are quicker than gelatin dynamites, and generally speaking the quickness increases with the strength. That is, the stronger grades are quicker, and the lower grades slower, in action.

The various grades of these three high explosives, offer a wide range in strength and quickness to select from, and it is always possible after a few trials to find an explosive exactly suited to the conditions.

Railroad tunnels, mine tunnels and drifts, highway tunnels, and irrigation tunnels, are being driven daily through various kinds of "ground." Often it is a matter of first importance to finish them quickly, and consequently details in regard to methods and equipment are matters of general interest. Within the past few months, a number of speed records in tunnels of different sizes have been made, and descriptions of them have appeared in various technical magazines.

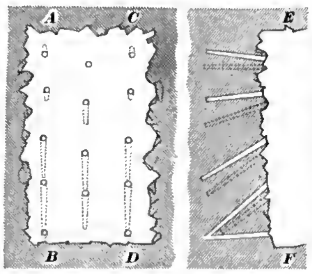
In *Engineering Contracting* of October 20, 1909, Mr. J. B. Lippincott, assistant chief engineer of the Los Angeles aqueduct, gave an interesting account of the driving of the Red Rock tunnel of the Los Angeles aqueduct system. In August, 1909, this tunnel, which is 9 ft. 10 in. $\times$ 10 ft. $8\frac{1}{2}$ in. in section was advanced 1,061.6 feet. Mr. Lippincott states that the explosives used were Du Pont 40-per-cent. ammonia dynamite and blasting powder.

In the *Engineering News* of November 18, 1909, the Red Rock tunnel is again referred to, and details are also given by Mr. C. H. Richards, division engineer, in regard to a tunnel on the Little Lake Division of the Los Angeles aqueduct. The explosives used in this tunnel were Hercules 40-per-cent. and 60-per-cent. gelatin dynamite, the average weight of explosives per cubic yard of rock, place measurement, having been only 3.3 pounds, or about 35 pounds per linear yard of tunnel almost 10 ft. $\times$ 10 ft. in section.

A short time before, accounts were given in several engineering magazines, of a record driving speed made in the Roosevelt drainage tunnel, at Cripple Creek, Col. The explosives used in this tunnel were 40-, 50-, and 60-per-cent. Repauno gelatin dynamite and Du Pont blasting gelatin.

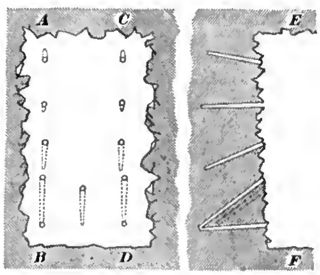
A very interesting description of the Rondout pressure tunnel of the Catskill aqueduct, written by John P. Hogan, assistant engineer of the New York City Board of Water Supply, was published in the January 1, 1910, number of the *Engineering Record*. Very rapid progress was made in this tunnel, and also in the Moodna pressure tunnel of the same system, described in an article in the *Engineering Record* of June 4, 1910. The explosive which gave best results, and which was used exclusively in both of these tunnels, was 60-per-cent. Forcite—a gelatin dynamite.

Reference to a paper by B. H. M. Hewett and W. L. Brown, on the land sections of the Pennsylvania Railroad North River tunnels, published in Vol. XXXVI of the Proceedings of the American Society of Civil Engineers, and reprinted in part in



Section E–F   Section A–B and C–D

Fig. 1



Section E–F   Section A–B and C–D

Fig. 2

Engineering Contracting of May 11, 1910, shows that 40-per-cent. Forcite was used in blasting on the Manhattan section, and 60-per-cent. Forcite on the Weehawken section.

The records of many other tunnels recently constructed, further illustrate how many kinds and strengths of explosives are used for blasting under the different conditions encountered in one class of work.

The specific cases referred to above, were all connected with large and important contracts, where equipment and methods were at the best, and several of these tunnels were driven at record speed. The fact that so many different explosives were used in the several tunnels, goes to show that care was taken to use the explosive which was best adapted to the conditions, and it is not unlikely that the speed of driving these tunnels, was largely due to the attention given to the selection of the explosives.

This point is equally important when driving narrow tunnels and drifts. After a study of the rock in a cross-cut, 3 ft. 6 in. $\times$ 7 ft. in section, recently driven by J. R. Marling in the Callie shaft at Cripple Creek, it was decided that best execution would be given by a 40-per-cent. gelatin dynamite. Repauno-40-per cent. gelatin was accordingly adopted, and it was necessary to drill 14 holes, as shown in Fig. 1, from 3 feet 6 inches to 4 feet 6 inches deep and blast them with about 35 pounds of 40-per-cent. gelatin dynamite, in order to advance the tunnel about 3 feet. In an attempt to increase the speed of driving, and to reduce the cost, the face was drilled with 11 holes, as shown in Fig. 2;

and these holes were loaded with Du Pont blasting gelatin in the points, and Repauno 40-per-cent. gelatin dynamite on top. In this method of loading about 7 pounds of the blasting gelatin, and 17 pounds of the gelatin dynamite were used, making a reduction of about 15 per cent. in the cost of explosives, and 20 per cent. in the amount of drilling, while the tunnel was still advanced full 3 feet each shift. Here the adoption of a more suitable explosive for the work, resulted in a great reduction in cost instead of increase in speed.

No matter what explosive is used in blasting, maximum results can only be attained by always having the explosive in a thoroughly thawed condition when it is used, by tamping firmly with damp clay, or similar material, from the charge of explosives to the mouth of the bore hole, and by using the strongest and highest grade detonator that can be secured.

Catalogs Received

In writing for catalogs, please mention Mines and Minerals

American-LaFrance Fire Engine Co., Elmira, N. Y., Fire Protection for Mines, 12 pages.

Carnegie Steel Co., Pittsburg, Pa., Steel Derricks and Drilling Machines, 31 pages.

Goodman Mfg. Co., Chicago, Ill., Bulletin 301, Goodman Rack Rail Haulage, 52 pages.

Ingersoll-Rand Co., 11 Broadway, New York, N. Y., "New Ingersoll" Coal Punchers, 24 pages; "Imperial" Type "E" Pneumatic Hammers, 16 pages; Davis "Calyx Diamondless" Core Drills, 48 pages.

J. Geo. Leyner Engineering Works Co., Littleton, Colo., Bulletin 1017, Descriptive Water Leyner Drills, 16 pages; Bulletin 1022, Part List No. 9 Water Leyner Drill, 4 pages.

Hill Clutch Co., Cleveland, Ohio, Catalog No. 8, Friction Clutches, 54 pages.

Mackintosh, Hemphill & Co., Fort Pitt Foundry, Pittsburg, Pa., Rolling Mills, Engines, and Machinery for Iron and Steel Works, 113 pages.

Watt Mining Car Wheel Co., Barnesville, Ohio, Catalog "F," Steel Mine Cars, 31 pages.

Pneumelectric Machine Co., Syracuse, N. Y., Coal Cutting With an Electric Puncher, 24 pages.

John A. Roebling's Sons Co., Trenton, N. J., The Log, The Road, The Rope, 8 pages.

Western Electric Co., New York, N. Y., Bulletin No. 1004, Central Battery Non-Multiple Switchboards With Lamp Signals, 20 pages; Bulletin No. 1008, Telephone Power Plant Equipments for Non-Multiple Switchboards, 24 pages.

National Electric Lamp Association, 4411 Hough Ave., Cleveland, Ohio, Bulletin No. 9A, The Cost of Light, 20 pages.

Portable Electric Safety Light Co., Newark, N. J., The Hubbell Electric Safety Lanterns for Mines, 8 pages.

Mulconroy Co., Philadelphia, Pa., Catalog No. 22 of Hose and Rubber Goods, 24 pages.

The Carb-Ox Co., Rogers Park, Chicago, Ill., Carb-Ox Specialties, 16 pages.

The "New Way" Motor Co., Lansing, Mich., Repair Price List and Book of Instructions and Suggestions to aid the user in operating "New-Way" Air-Cooled Engines, 37 pages.

General Electric Co., Schenectady, N. Y., Bulletin No. 4717, G-1 Flame Arc Lamp, 16 pages; Bulletin No. 4743, Intensified Arc Lamps, 16 pages; Bulletin No. 4750, Couplings, 8 pages; Bulletin No. 4751, Polyphase Induction Motors, 20 pages; Bulletin No. 4752, Series Luminous Arc Rectifier Systems, 26 pages; Bulletin 4754, Mill Type Motors, 14 pages; Bulletin No. 4756, Ventilation of Horizontal Steam Turbine Alternators, 2 pages; Bulletin No. 4760, Direct Current Instruments, Types D2 and D3, 12 pages; Bulletin No. 4762, Thomson Polyphase Watthour Meters, 16 pages.

Crawford & McCrimmon Co., Brazil, Ind., Bulletin G-9, Lead-Lined Acid-Proof Pumps for Mines, 4 pages.

Colorado Iron Works Co., Denver, Colo., Pamphlet No. 24, The Akins Classifier, 4 pages; Pamphlet No. 26, The Rothwell Continuous Thickener and Filter, 8 pages.

Link-Belt Co., Philadelphia, Pa., "Maximum" Silent Chain, 40 pages; Catalog No. 90, General Price List, 400 pages.

Sullivan Machinery Co., Chicago, Ill., Bulletin 57-A, The Sullivan Post Puncher for Mining Coal, 16 pages.

Sturtevant Mill Co., Boston, Mass., Sturtevant Laboratory, 8 pages.

John Davis & Son (Derby), Ltd., 110 West Fayette St., Baltimore, Md., Anemometers, Water Gauges, Hygrometers, Barometers, Engineering and Meteorological Instruments, 44 pages.

DeLaval Steam Turbine Co., Trenton, N. J., DeLaval High Efficiency Centrifugal Pumps, 96 pages.

Taylor Iron & Steel Co., High Bridge, N. J., Bulletin No. 108, Tisco Manganese Steel Mine and Skip Car Wheels, 16 pages.

Taylor Instrument Cos., Rochester, N. Y., Numbers 5 and 6, "Tycos"-Rochester. Magazines describing the product of the company.

Dodge Mfg. Co., Mishawaka, Ind., "How to Save Fuel," 8 pages.

C. W. Hunt Co., New York, N. Y., General Catalog No. 102, describing the various types of machinery, 112 pages.

Detroit Leather Specialty Co., Inc., Detroit, Mich., has issued a pamphlet called "Why" which gives reasons why their leather packing wears so well.

W. S. Tyler Co., Cleveland, Ohio, has issued a book explanatory of their screen for mailing purposes. It contains exceedingly valuable information.

Ridgway Dynamo and Engine Co., Ridgway, Pa., have combined their various bulletins into an illustrated catalog descriptive of their products. It also contains photographs of various plants which they have installed.

FOREMAN OF MINE RESCUE STATION

The United States Civil Service Commission announces an examination on October 15, 1910, to secure eligibles from which to make certification to fill vacancies as they may occur in the position of foreman in connection with mine-rescue training at the stations of the Bureau of Mines located throughout the United States, at salaries ranging from $1,080 to $1,500 per annum.

The examination will consist of the subjects mentioned below, weighted as indicated: (1) Physical ability, weight, 40; (2) training and experience, weight, 60. Total, 100.

Applicants for this position should have had experience as fire boss, mine foreman, or inspector in coal mining. They should be thoroughly experienced in underground work, capable of finding their way in the darkness of mines with which they have had no previous familiarity, and should have the courage, skill, and physical endurance required in directing rescue operations underground after mine explosions.

Applicants must execute their applications in their own handwriting.

Applicants must have reached their 25th birthday, but not their 50th birthday, on the date of the examination.

Applicants should at once apply to the United States Civil Service Commission, Washington, D. C., for application and examination Form 1800. No application will be accepted unless properly executed and filed in complete form with the Commission at Washington prior to the hour of closing business on October 15, 1910. In applying for this examination the exact title as given at the head of this announcement should be used in the application.

OXYGEN HELMETS USED AT MINE FIRE

*Written for Mines and Minerals, by T. A. Carraher**

This is written in reply to an article in the July edition of Mines and Minerals entitled "Oxygen Helmets Used at Mine Fire," by Oscar Cartlidge, a mine manager of the Hart-Williams Coal Co. I wish to correct some statements made by Mr. Cartlidge, whom I succeeded on April 1, 1909.

On the night of June 15, 1909, as the result of blasting coal with black powder, a fire started in the face of the fourth southeast entry in the mine of the Hart-Williams Coal Co. It was 9 p. m. when I received the information that there was a fire somewhere in the mine; as quickly as possible the superintendent and I drove to the mine and on arrival found great clouds of smoke coming out of the hoisting shaft, which is the upcast. I went down immediately, but on the bottom found the smoke so dense that it was impossible to reach the fire by this route. On going to the top and being assured that the men were all out I gave orders for the fan to be reversed, but on investigation found that no provision had ever been made for reversing the fan, and this idea had to be abandoned.

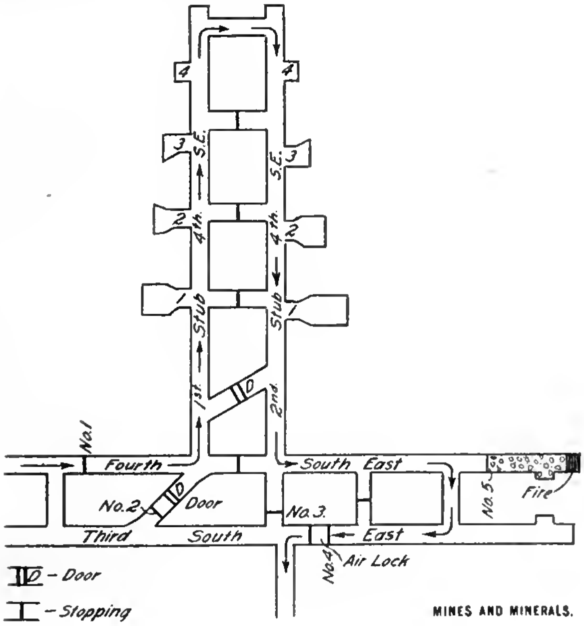


Fig. 1

I again descended, taking with me the night boss and four of his men, with whom I proceeded by way of the intake to the fourth southeast entry where I saw at once that it was useless to attempt to fight the fire, so we began to wall it off immediately, accomplishing this by means of stoppings numbered 1, 2, 3, and 4, as shown by the map, Fig. 1. These stoppings were built of boards nailed to good strong props and plastered with wood fiber instead of walls filled between with fireclay, as stated in the article referred to above, and the last wall was completed at 3:30 a. m., June 16, 1909.

This mine at this time was 3½ years old, but the average output per working day was only 590 tons, therefore, the loss of 12 working places and 1 chain machine was felt so severely we resolved to open the district as soon as possible.

On June 19 we began making the water connections and building an air lock of ship-lap lumber; and a door was cut of sufficient size to admit a man with a helmet; the location of the air lock may be seen from the accompanying plat. Two helmets and two storage-battery electric lights were borrowed from a neighboring mine and at 2 p. m. we had everything ready for entering the walled-off area.

*Mine Manager for Hart-Williams Coal Co., Benton, Ill.

31—3—5

L. T. Putman, a mining engineer, and I, then proceeded to the fire-wall where we put on the helmets and entered through the door in stopping at No. 4; we then cut a door in the second wall and entered the fire district, closing the door after us; we next proceeded to the face of the fourth southeast where we found a fall about 10 feet high and 40 feet long; the fire was, of course, smoldering beneath and the place was very hot. Having previously extended the water line to the outside wall of the air lock we then made the hose connections and dragged the hose inside and up to the top of the fall, where the water was started at 3 p. m. Saturday, and allowed to run continuously until 2 p. m. Sunday, or during the period of 23 hours, the position of the hose being changed every 3 or 4 hours. At the end of this time the face of this entry was submerged in water (it being a dip heading) and we were quite certain that the fire was extinguished, but as a matter of precaution we placed a good canvas brattice at the position shown by 5, the occasion however did not require a plastered stopping, which any one who has had any experience in Draeger helmet work will agree is very difficult to build while wearing a helmet.

We then started the ventilation on its regular course, as shown by the accompanying plat, and in less than 30 minutes the section was entirely free from gas, this being less than 5 days from the date on which the fire was sealed off.

To any one who has seen the actual conditions it would be clearly apparent that it was not a hazardous thing to ventilate this section in this manner, for we had every assurance that the fire was extinguished.

On the following Tuesday, for lack of men to load the fall, I built a plastered stopping just in front of the fall merely as a precaution in case the fire might possibly start up again.

PERSONALS

Frank Koester, of New York, in an important paper presented before the recent convention of the Society for the Promotion of Engineering Education, held at Madison, Wis., discussed in detail the "Educational System of the German Technical Universities." He also analysed the conditions and standing of the German engineer as compared with our own.

Prof. H. P. Boardman, of the Mackay School of Mines, University of Nevada, spent the summer months in teaching the junior class practical metal-mine surveying. Prof. J. C. Jones had charge of the geological work.

Charles Wilke, formerly general manager of sales of the Jeanesville Iron Works Co., has been appointed district manager at Philadelphia for the Wheeler Condenser and Engineering Co. His office is at 318 Commonwealth Building.

Marshall D. Draper and John Gross announce the establishment of the firm of Draper & Gross, mining and metallurgical engineers, with offices at 746 Equitable Building, Denver, Colo. Mr. Draper will be more directly in charge of the mining, and Mr. Gross of the metallurgical work.

Vice-Consul-General Stuart J. Fuller states that ever since Hong Kong was established the industry of vermilion making, entirely in the hands of the Chinese, has been an important one.

The manufacture of this pigment is among the foremost of the colony's industries. There are something like 100 small plants for the manufacture of vermilion in Hong Kong and Kowloon. The raw material comes from Australia, and the vermilion is prepared altogether by what is known as the wet method. The Chinese made artificial cinnabar long before Europe was a civilized country, and to this day there are trade secrets in the vermilion industry which no European has yet been able to fathom. Some of the granite stones here, between which the pulverized ore is ground, are almost prehistoric.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

Western Editorial Office: 728 Equitable Building, Denver, Colo.
Chicago Office: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to Mines and Minerals, Scranton, Pa.
CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER........Manager
EUGENE B. WILSON, Scranton, Pa........Editor
GEORGE F. DUCK, E. M., Denver, Colo........Western Editor
P. G. MOORE........Circulation Manager
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., Ltd., Dryden House, 43 Gerrard St., Soho, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 Pitt Street, SIDNEY.
SOUTH AFRICAN AGENTS: J. AFFLECK ROBERTSON, CAPE TOWN, Mansion House; JOHANNESBURG, Trust Building; DURBAN, Natal Bank Chambers; BLOEMFONTEIN, Florence Buildings.

SUBSCRIPTION RATES

Single Copies........25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada........2.50
To all other Countries........3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always give your former as well as your present address.

Table of Contents

Hydraulic Air Compression........ 129
Reclaiming Zinc-Lead Fines, by Lucius L. Wittich........ 131
Pumping at Bisbee, Arizona, by C. C. Austin........ 132
Oil Notes........ 134
Tunneling on Los Angeles Aqueduct, by R. L. Herrick........ 135
Kennett Smelter Fume Abatement........ 143
Wet Assay for Gold........ 143
Timbering in the Joplin District, by Lucius L. Wittich........ 144
Oxygen Process for Melting of Iron, by Doctor Alfred Gradenwitz........ 146
History of the Water Leyner Drill, by Chas. A. Hirschberg........ 148
Lake Superior Copper Ores........ 149
Sectional Slag Pot, by E. C. Reeder........ 149
Utah Consolidated Aerial Tramway, by Leroy A. Palmer........ 150
Lake Superior Iron Ore........ 152
Slags........ 152
The Evolution of Hoisting, by E. B. W........ 153
A Simple Hydraulic Hoist........ 157
Petroleum Industry, Venezuela........ 158
Calculating the Tonnage of Ore Piles, by R. T. Hancock........ 158
Explosives for Tunnel Driving........ 159
Catalogs Received........ 160
Foreman of Mine Rescue Station........ 160
Oxygen Helmets Used at Mine Fire, by T. A. Carraher........ 161
Personals........ 161
Editorial—
Announcement—The Director of the Bureau of Mines—Carbon Dioxide and Vegetation The Oxygen Blowpipe—Coalinga Oil Field........ 162
Book Review........ 164
Correspondence—
Recollections of William Smurthwaite—Black Powder—Method of Laying Off a Curve—Surveying—Lamp Burning in Water—Thermochemistry of Anthracite........ 165
Bunker Hill Coal-Storage Plant, by F. W. Brady........ 166
Elliptical vs. Rectangular Shafts, by Wm. Archie Weldin........ 167
Mechanical Coal Sampler, by C. E Scott........ 169
Mining, Preparing, and Coking Coal........ 171
Electric Motors for Rope Haulage, by H. W. Reybold........ 174
The Coefficient of Friction, by Wm. Clifford........ 176
Practical Fuel Values, by W. P. Young........ 178
Timbering in Indian Coal Mines........ 179
Pictou Coal-Field Lore........ 179
Power Production at Collieries by, Howard N. Eavenson........ 180
Examination Questions—
Selected Questions, Springfield, Ill., Examinations, 1909........ 182
Mine Rescue Work........ 184
Koppers By-Product Ovens, by W. E. Hartman........ 185
New Tile-Laying Machine, by Bert Lloyd........ 187
Protection Value of Humidity, by James Ashworth........ 188
Trade Notices........ 189
Societies........ 189
Sampling Coal and Coke, by E. G. Bailey........ 190
New Inventions........ 192
Squibs—
Tin Mining in China, 131—Gold Production of Alaska, 1909, 147—Proof Assay for Copper, 149—The Smelter-Smoke Problem in California, 151—Minerals of Greece, 157—Gasoline Locomotive, 158—Chinese Vermilion Industry, 161—New Western Pacific Railroad, 170—Carbic Cake, 174—Natural Gas and Petroleum in Alabama, 175—Possible Effect of the Panama Canal on Alaskan Coal Business, 175—Dan River Coal, 177—Capacity of Screen........ 181

ANNOUNCEMENT

MR. WIN. G. CONLEY has been appointed Eastern Advertising Representative for Mines and Minerals, succeeding Mr. Webster H. Taylor.

THE DIRECTOR OF THE BUREAU OF MINES

IT would be a very strong sign of the near approach of the millennium if President Taft's appointment of Dr. J. A Holmes as Director of the Bureau of Mines had been accepted without adverse criticism.

Doctor Holmes is a geologist, born at Laurens, South Carolina, in 1859; a graduate from Cornell in 1881 with the degree of B. S.; was Professor of Geology and Natural History in the University of North Carolina from 1881 to 1891; was State Geologist of North Carolina from 1891 to 1904. He was in charge of the United States Geological Survey Laboratory for Testing Fuels and Structural Materials in St. Louis, from 1904 to 1907, and since then in charge of the Pittsburg and other similar testing stations of the Survey.

There is no question that Doctor Holmes has given the subject of increased safety in the working of coal mines careful and conscientious study and has acquired a great deal of very valuable experience. As a geologist he is, from an educational point, and from the standpoint of experience, an able man. However, Doctor Holmes is not a practical mining engineer, and most of the adverse criticisms on his appointment come from those who conscientiously believe that the Director of the Bureau of Mines should be a mining engineer of education and broad practical experience.

As this is an age of specialty in mining engineering, as well as in other professions, it would probably be an impossible task to select an educated mining engineer of practical experience in all kinds of mining of sufficient ability to fill the position for the salary the government offers. Such engineers are very scarce at any price.

Mr. John Mitchell, speaking from the standpoint of the miner, accords to Doctor Holmes a due measure of credit for what he has done, but criticises the appointment on the ground that the law provides for the appointment of a mining engineer and makes impossible the appointment of a practical miner, a man whom Mr. Mitchell claims would know more about the actual conditions existing in the mines.

There is no doubt that Doctor Holmes, while not a practical mining engineer, can just as well be classed as a mining engineer as many who claim such a title. The designation, "Mining Engineer," is a very broad one, particularly in this country where there are numerous examples of men who, without a college education or an engineering degree from a college have, by sheer force of natural ability and hard work, won positions in the foremost rank of American mining engineers. Colleges confer the degree of "Mining Engineer" on young men who have simply acquired at college a fundamental education on which a mining engineering

reputation can be built by experience. Statistics show that a comparatively small number of such men remain in the profession after leaving college and, of course, all that do remain do not rise to eminent positions.

Taken all in all, Doctor Holmes secured his education at one of the best universities of the country. He has added to it nearly 30 years of experience and study which has given him knowledge of great value, and as he is a man of exceptionally clean character and much tact, he has probably just as many qualifications for making good as Director of the Bureau of Mines as any man President Taft could select. Had President Taft appointed a coal-mining engineer, no matter how eminent, the ore-mining interests of the country would have criticised him on the ground that the appointee was not familiar with the conditions existing in their mines and would be apt to cause trouble. If, on the other hand, as eminent a mining engineer in the field of metal mining as Mr. John Hays Hammond had been selected, the coal mine owners would have had just as strong a reason to object to him from their standpoint.

Viewing the appointment from all sides, we are inclined to believe that, everything considered, the appointment of Doctor Holmes was as wise a one as could be made, and we express, both for the mining fraternity and for Doctor Holmes, the wish that his record in the difficult position of organizing the Bureau of Mines and of carrying it on in its first crucial years shall be so successful as to meet the approval of all classes connected with the mining industry.

CARBON DIOXIDE AND VEGETATION

Curious surmises originate when contemplating such abstract matters as the final distribution of the millions of tons of carbon dioxide mixed yearly with the atmosphere.

Roughly estimated, there are 3 tons of coal consumed yearly for each inhabitant in the United States, and if the coal contains 90 per cent. of carbon this amount will produce 10 tons of carbon dioxide, thus from the combustion of coal 900,000,000 tons of carbon dioxide go into the atmosphere yearly.

When the exhalations of individuals and animals, the decay of vegetation and other organic substances, the combustion of oil, gas, wood, tobacco, etc., are annexed to the problem, it is probable that the total of carbon dioxide going yearly into the atmosphere will reach 1,600,000,000 tons in the United States alone.

Carbon dioxide, while not poisonous, has, when greatly in excess in the atmosphere, a toxic effect, which suggests such questions as the following: Is the carbon dioxide in our atmosphere increasing? What becomes of the carbon dioxide going into our atmosphere?

In order to answer the first question it is necessary to revert to ancient geological times and take up the formation of limestone, which, as all engineers know, is composed of calcium oxide, an inorganic product, and carbon dioxide, an organic product, in the ratio of $\frac{56}{44}$. There is no inorganic carbon, which makes it necessary to assume that organisms of some kind produced the CO_2 to form limestone. The organic life which produced the limestone came from the sea, in fact the large beds of limestone in the United States, which greatly exceed the coal beds in area and tonnage, were mostly made by shell fish. After their formation the surplus of carbon dioxide was so enormous that it formed the coal beds of the Carboniferous period, and it is probable that owing to the quantity of CO_2 in the atmosphere of that time no land animals other than amphibians, could exist, at least there are no geological records of them.

It therefore is not probable that the quantity of carbon dioxide is increasing in the atmosphere, in fact analysis shows it to contain the approximate normal one-fifth oxgyen and four-fifths nitrogen.

The second question involves a further consideration of vegetation whose structure, largely carbon, derives that element mostly from the carbon dioxide in the atmosphere, at the same time it returns the oxygen to the atmosphere. From the quantity of carbon dioxide going into the atmosphere and the great number of trees that have been cut down it would seem as if the carbon dioxide must increase in the atmosphere, or vegetation grow more luxuriant. As the forests have diminished the grain fields have expanded and kept the atmosphere normal by their absorption of carbon dioxide. Thirty years ago Kansas and Nebraska were included in the Great American Desert, today they are in the agricultural belt. The land was arid and the atmosphere contained little carbon dioxide, while at this time it is receiving considerable from various sources. Given proper soil, the results of cultivation will depend on carbon dioxide and moisture, and if one of these factors is lacking the other is likely to be also.

THE OXYGEN BLOWPIPE

In 1908 the Germans introduced the autogenous method of cutting and welding metals with an oxy-hydrogen blowpipe. This was an application of what had been known previously as the hottest flame obtainable and was produced by the combustion of hydrogen in a properly proportioned mixture of oxygen. The heat of combustion of hydrogen gas is 58,060 calories. The Davis-Bournonville Co. substituted acetylene for hydrogen and by this means obtained a calorific power of 395,420 calories, which probably is the most satisfactory welding combustion, owing to the reducing action of the carbon in the acetylene. They also ascertained that if, after heating the metal with a welding flame, an auxiliary jet of oxygen under higher pressure was added, they could cut 2-inch-thick plates of metal with little difficulty.

The Germans, still clinging to their oxy-hydrogen flame, found that if they increased the oxygen after

heating the metal, they could cut metal just as readily, and that the higher the pressure the oxygen was delivered, the quicker would be the cut. This seems at first paradoxical because the highest calorific power of the highest oxide of iron is 270,800 calories, a heat much below that produced by acetylene. When it is considered that the calorific power or total heat produced depends upon the given weight of a substance consumed in a given time and not on its reduction, the paradox is explained, as neither hydrogen nor acetylene are factors in the combustion after heating the metal to the proper temperature, and either one may be discarded.

The construction and operation of the oxygen blowpipe in opening chilled blast furnaces and plugged-steel furnaces is illustrated on another page of this issue.

IN the article on the Coalinga Oil Field in August MINES AND MINERALS to which "we point with pride" because of its neatness and meatiness, we jumbled up material from various sources. P. W. Prutzman's report furnished material for the first column, while Ralph Arnold and Robert Anderson furnished the remainder of the article with one exception, the illustration of the California Midway oil well. This cut came from the Los Angeles *Mining Review*, to whom felicitations are extended for the courtesy.

BOOK REVIEW

In recent years great advances have been made in the art of printing, not only in color effects, but in artistic typography and improved methods of illustration. The railroads are the quickest to take advantage of the newest and most up-to-date processes in artistic printing. A particularly fine example of the highest class of illustrated pamphlet work is "A Glimpse of Utah," just issued by the Passenger Department of the Denver & Rio Grande Railroad. The text is by the brilliant writer, Judge E. F. Colborn, of Salt Lake City, and the excellent illustrations picture the many unique features of that interesting state, Utah.

ALPHA PORTLAND CEMENT CO., Easton, Pa. A very attractive pamphlet, finely illustrated. The Alpha Portland Cement Co. is the pioneer manufacturer of Portland cement in America and its Alpha brand has stood the test for some 19 years. Some of the subjects included are, the proportioning of cement, sand, and stone; the comparative costs of cement; method of mixing; various forms of concrete construction; the protection of concrete from frost; sidewalks and pavement; curbing and gutters; reinforced concrete; concrete blocks; waterproofing; sea-water construction There is an explanation of cement terms, and Extracts from the Report of the Committee of American Society for Testing Materials on standard specifications. This pamphlet is not only a work of art but will appeal to all users of cement.

"THE NATURAL RESOURCES OF COLORADO, UTAH, AND NEW MEXICO," the 1910 edition of the Denver & Rio Grande folder, is especially valuable because of the large map, which takes in territory from the Rocky Mountains west to the Pacific Coast, is not only brought up to date but shows also the route and stations of the new Western Pacific Railway, the Pacific Coast extension of the Denver & Rio Grande Railroad from Salt Lake City to San Francisco

UNITED STATES GEOLOGICAL SURVEY PUBLICATIONS, Washington, D. C.: Bulletin 381-B, Investigations of Coal Fields in Wyoming in 1908, by R. W. Stone, C. T. Lupton, H. S. Gale, C. H. Wegeman, E. G. Woodruff, M. W. Ball, Eugene Stebinger, and A. R. Schultz; Bulletin 430-D, Rare Metals, by Howland Bancroft, J. M. Hill, F. C. Schrader, E. C. Harder, T. L. Watson, and Stephen Taber; Bulletin 430-J, Miscellaneous Non-metallic Products, by D. B. Sterrett and H. S. Gale. Water-Supply Paper 237, The Quality of the Surface Waters of California, by Walton Van Winkle and Frederick M. Eaton; Water-Supply Paper 246, Part VI, Missouri River Basin, by Robert Follansbee and J. E. Stewart. Bulletin 381-C, Investigations of Coal Fields in Colorado and New Mexico in 1908, by G. C. Martin, C. W. Washburne, M. I. Goldman, C. B. Richardson, and J. H. Gardner; Bulletin 430-F, Advance Chapter from Contributions to Economic Geology in 1909, Part 1, Metals and Non-metals, except Fuels, by E. F. Burchard, A. H. Purdue, J. A. Udden, T. N. Dale, J. H. Gardner, N. H. Darton, E. W. Shaw, W. C. Alden, E. C. Harder, and F. L. Hess; Bulletin 442-A, The Mining Industry in 1909 and Alaska Coal and Its Utilization, by Alfred H. Brooks. Water-Supply Paper 251, Surface Water Supply of the United States in 1907–8, Part XI, by W. B. Clapp and W. F. Martin. Professional Paper No. 68, The Ore Deposits of New Mexico, by Waldemar Lindgren, Louis C. Graton, and Charles H. Gordon.

UNITED STATES DEPARTMENT OF AGRICULTURE, Washington, D. C., Bulletin No. 81, The Forests of Alaska, by R. S. Kellogg.

PAINT MANUFACTURERS' ASSOCIATION OF THE UNITED STATES, Philadelphia, Pa., Bulletin No. 26, Second Annual Report on Wearing of Paints Applied to Atlantic City Test Fence; Bulletin No. 27, Second Annual Report on Atlantic City Steel Test Fence.

ANNUAL REPORT OF THE WATER SUPPLY COMMISSION OF PENNSYLVANIA, Thomas J. Lynch, South Bethlehem, Pa.

NORTH CAROLINA GEOLOGICAL AND ECONOMIC SURVEY, Joseph Hyde Pratt, State Geologist, Chapel Hill, N. C., Bulletin No. 18, Bibliography of North Carolina Geology, Mineralogy and Geography, With a List of Maps, by Francis Baker Laney, Ph. D., and Katharine Hill Wood.

UNIVERSITY OF MISSOURI, Rolla, Mo., Vol. 2, No. 3, School of Mines and Metallurgy, on "Some of the Essentials of Success."

MAP SHOWING MINERAL CLAIMS LOCATED IN THE ORIENT MINING DISTRICT, published by L. K. Armstrong, Mining Engineer, Spokane, Wash. Price 10 cents each.

THE DICTIONARY OF METALLURGICAL AND CHEMICAL MATERIAL, published by *Metallurgical and Chemical Engineering*, 239 West 39th Street, New York, N. Y. Price 50 cents.

A WATER-JACKET FURNACE, by G. C. De Venancourt, Engineer of Arts, Consulting Engineer of Materials for Mines, published by H. Dunod and E. Pinat, 47 Grand Augustins, Paris, France. Price 50 francs.

CANADA DEPARTMENT OF MINES, MINES BRANCH, OTTAWA, CANADA, Bulletin No. 2, Iron Ore Deposits of the Bristol Mine, Pontiac County, Quebec, Magnetometric Survey, etc., by E. Lindeman, M. E.

ANNUAL REPORT OF THE MINISTER OF MINES FOR THE YEAR ENDING DECEMBER 31, 1909, by William Fleet Robertson, Provincial Mineralogist, Victoria, British Columbia, Canada.

GEOLOGICAL SURVEY OF GEORGIA, S. W. McCallie, State Geologist, Atlanta, Ga., Bulletin No. 24, A Second Report on the Public Roads of Georgia.

TENNESSEE STATE GEOLOGICAL SURVEY, George H. Ashley, State Geologist, Nashville, Tenn., Bulletin No. 1-A, The Establishment, Purpose, Scope, and Methods of the State Geological Survey, by George H. Ashley; Bulletin No. 3, Drainage Reclamation in Tennessee, by George H. Ashley.

UNIVERSITY OF PITTSBURG BULLETIN, Cooperative System, Pittsburg, Pa.

CORRESPONDENCE

Recollections of William Smurthwaite

Editor Mines and Minerals:

Sir:—I became acquainted with William Smurthwaite in 1874, shortly after the mine inspection law of Ohio was enacted. He was the most intelligent practical miner which I have ever met and I doubt if he had a superior during his lifetime. The large mine at Stubenville, Ohio, which he managed, made copious volumes of firedamp, but he had the gas under perfect control. No miner was permitted to go down the shaft until he had personally examined every working place in the mine, returned to the surface and pronounced the mine safe. He went down the pit every morning at 4 o'clock. He enforced strict discipline and had the unbounded confidence of the miners and his word was law, for the underground employes recognized the fact that he was ever vigilant to keep the mine free of inflammable air.

He surveyed and mapped the mine himself and was the only mine manager I ever knew who carried a column of air through all the ramifications of a mine without losing a perceptible part of the column. He was ever watchful of the stoppings and never permitted the air to leak. He made the rounds of the aircourses every day, or sent a trained assistant to do so. All the entries were double, butt and main, and when a new breakthrough was made at the head of a double entry, the stream of air was so strong that he had to keep the next breakthrough below open until the workmen advanced the entry several yards, for the strong current of air chilled the miners and it was sometimes difficult if not impossible to keep the strong current from blowing out the workmen's lights.

Were all mines ventilated in this manner, explosions by firedamp would be few. When an explosion occurs in a mine, somebody is blamed for carrying a naked light into a column of gas. No standing gas was ever permitted to accumulate in the mines in charge of William Smurthwaite. He regarded an explosion as a crime, asserting that there was no occasion for permitting standing gas in a mine, and took the ground that explosions were never caused by the fault of the workmen, a statement he proved by a period of more than half a century of underground life.

The miners of Stubenville, Ohio, owe it to themselves to erect a fitting monument to his memory. If all mine bosses and mine inspectors were half as earnestly devoted to their duty explosions would be like angels visits, "few and far between."

Girard, Ill. Andrew Roy

Black Powder

Editor Mines and Minerals:

Sir:—I wish to take exception to Mr. Beard's answer to question:

"What powders are best adapted to various conditions of mining?

"Ans.—Black powder."

In the first place black powder is not fit for use in many of our coal mines owing to the smallness and hardness of the bed. Black-powder action is such as to break the coal and bone so fine as to make it almost impossible to separate them, while some of the Du Pont Powder Co.'s permissible explosives in these same beds will produce lump coal and leave it in a condition to be cleaned. The permissible explosive reduces the danger to life and limb; sparks from the miner's lamp do not ignite it, etc. I think there are so many things in favor of the permissible explosive, that black powder should be eliminated from mine use.

West Pittston J. S.

Method of Laying Off a Curve

Editor Mines and Minerals:

Sir:—I enclose you a method of laying off curves which may be of occasional use. It is elaborated more than is necessary for practical purposes, but this was done to make the procedure clearer.

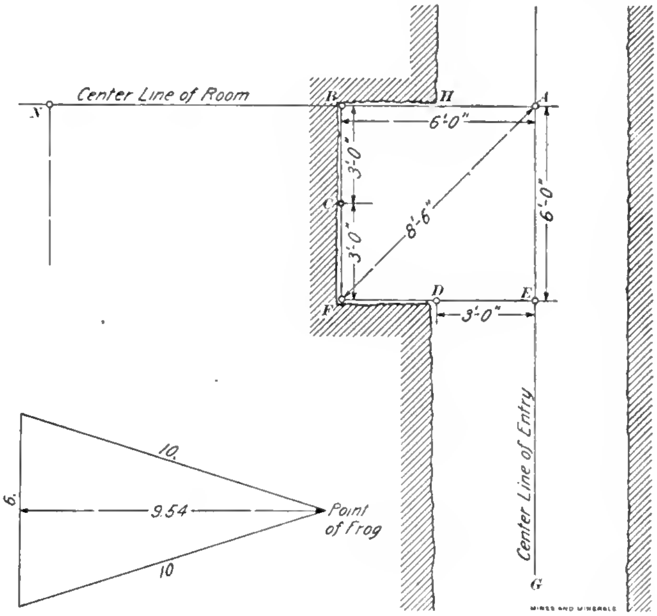


Fig. 1

Note that the two points, *C* and *D*, on the curve are obtained at the start of the operation and that these practically control the work. Also, that the radius of the curve is always $2\frac{1}{2}$ times the side of the square.

To make use of this method:

First. Excavate *H B F D*, Fig. 1. At the center line of the entry and the room make a right angle. *A B F E* is a 6-foot square. *C* and *D* are the middle points of two sides, they are also centers of curves into room. The 6-foot square should be laid off from a long string stretched from *A* to *G*. By the use of an 8-foot 6-inch rod having a 3-foot mark on it and hinged so as to be 6 feet long when folded, *A B F E* may be

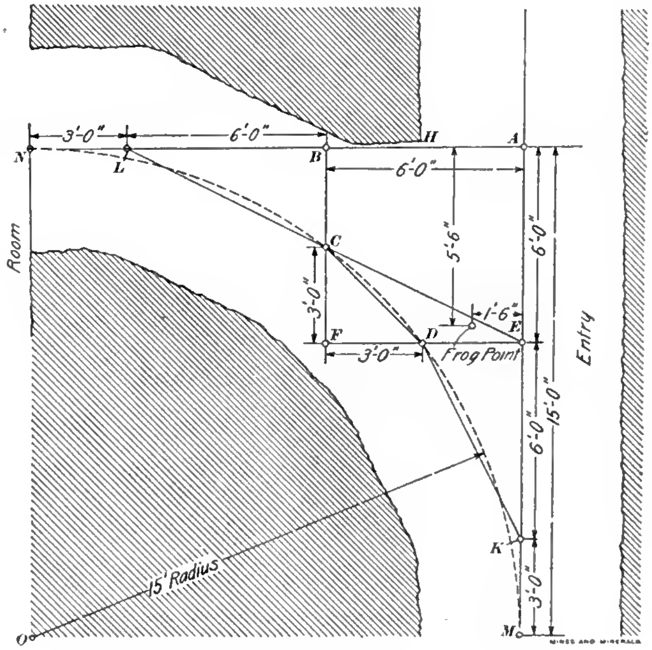


Fig. 2

made exactly square. Good pins are to be driven at the points A, B, C, D, E, and F.

Second. Mark the point K Fig. 2 on center line of entry 6 feet from E, and then use $K\ D$ as a center line for the side cut from K to D.

Third. Use $D\ C$ as a center line for side cut from D to C.

Fourth. Excavate along line $E\ C$ continued as a center line to the point L, making $CL = EC = 6$ feet $8\frac{1}{2}$ inches. L is also on the center line of room 6 feet from B. From L continue along center line of room for 3 feet to N.

Kansas City J. R. Stephens

Surveying

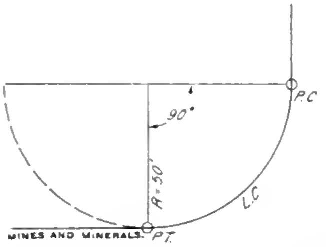
Editor Mines and Minerals:

Sir:—Replying to the question by Gordon S. Cox, Esq., in your issue of July: "What would be the length and degree of a 50-foot radius curve, central angle = 90 degrees?"

$$\text{L. C.} = \frac{(2 \times 50) \times 3.14159265}{4} = 78.539816 \text{ ft.}$$

$$\text{L. C.} = 50 \times 1.5707963 = 78.539815 \text{ ft.}$$

Definition: The degree of curve is an expression identical with the angle at the center of the curve subtended by a chord of 100 feet. When $R = 50$ ft. $D = 180°\ 0'$. It is often found convenient to express D in terms of a chord shorter than 100 feet when by the formula,



$$\text{Sine } \tfrac{1}{2}\, D = \frac{C}{2R}$$

Chord (C) = degree of curve (D); 10 feet = 11° 28′; 20 feet = 23° 04′; 30 feet = 34° 54′; 40 feet = 47° 10′; 50 feet = 60° 00′; 60 feet = 73° 44′; 70 feet = 88° 52′; 80 feet = 106° 16′; 90 feet = 128° 20′; 100 feet = 180° 00′.

Boston Mint Lee Fraser

Lamp Burning in Water

Editor Mines and Minerals:

Sir:—In answer to Student, in the August number of Mines and Minerals, I would give it as my opinion that the burning of his lamp flame in the discharge water of the pump is probably due to the gas, sulphureted hydrogen (H_2S), which has been absorbed by the water in the old lower workings, thence carried to the surface or discharge of the pump I presume the water in the workings is under considerable pressure and if so, gas would be dissolved in the water at a greater density than it would at the ordinary pressure of the atmosphere, so when the water reaches the surface the tendency would be to disengage the gas, together perhaps with some air, which causes the lamp to burn for a short time in the water. This is only an opinion and not intended to be taken as an authoritative statement.

Montcalm, W. Va. E. W Bailey

Thermochemistry of Anthracite

Editor Mines and Minerals:

Sir:—In reply to Mr. R. T. Hancock's communication, page 37, August issue of Mines and Minerals, regarding the apparent heating value of anthracite, would state that in the laboratory we have noted the same discrepancy and have traced it to the incomplete combustion of high-ash coals in the calorimeter. We have after considerable experimentation evolved a slightly modified method by which the British thermal units per pound of combustible, as determined with the Atwater calorimeter vary but slightly, irrespective of ash content, provided this ash content be less than 40 per cent.

Pottsville, Pa. Edwin M. Chance

BUNKER HILL COAL-STORAGE PLANT

Written for Mines and Minerals, by F. Webster Brady

The large anthracite companies in order to work regularly when there is a car shortage or stagnation in the coal market, have places where they stack coal in great heaps. These heaps are drawn on in time, or when for any cause the collieries are unable to supply the market's demand.

Coal-storage plants are arranged usually on level ground with machinery for stacking and reloading from the piles, and provided the company does not possess such ground in the vicinity the coal is transported to a convenient place and there stacked. The Hillside Coal and Iron Co. has taken an unusual course in the matter of coal storage and has made use of the southern exposure of Bunker Hill, in Dunmore, Pa., as shown in Fig. 1. The storage floor is 800 feet long by 230 feet wide and has a capacity for the storage of 80,000 tons of coal. The floor which has a slope of 32 degrees is made of ash concrete 10 inches thick, with gutters lined with stones and suitably arranged for drainage. The coal comes on the trestle, shown in upper part of the floor, in hopper-bottomed gondolas When the hopper doors are opened the coal slides through the trestle and to a small extent over the floor, although the bulk of the storage piles are made by means of a portable belt conveyer, operated by an electric motor. These piles are made as shown in Fig. 2, so that the coal will slide by gravity to the reloading chutes at the lower side of the floor. It will be noticed that the framework of the chutes rests on a retaining wall 10 feet 4 inches high, 5 feet 6 inches thick at the bottom and 3 feet thick at the top, and that the floor of the framework slopes backwards forming a dam. This arrangement prevents the chutes from being pushed out by the coal stacked back of the continuous framework, and permits the chute to act as a blow-off for the dam. Each chute is boarded up on the track side, except where the coal flows through, and this is closed by a swinging apron gate, operated by a lever. The coal shown on the floor is steam fuel, known as anthracite barley. The entire floor is surrounded by a board fence forming a stockade to keep out the public.

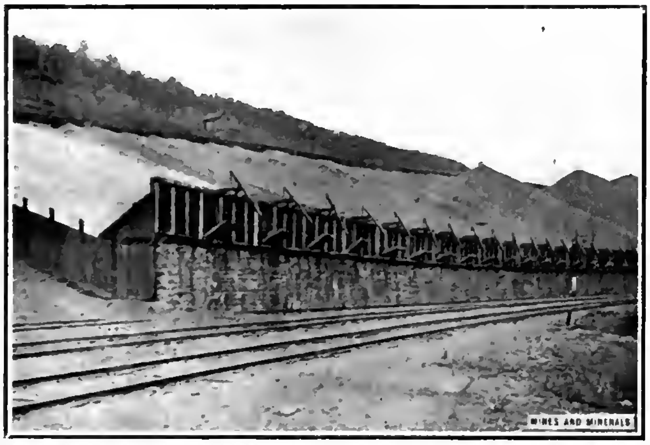


Fig. 1. Bunker Hill Storage Plant, Empty

Fig. 2. Bunker Hill Storage Plant, Full

ELLIPTICAL VS. RECTANGULAR SHAFTS

*Written for Mines and Minerals, by Wm. Archie Weldin, C. E.**

The article in Mines and Minerals by Mr. Francis Donaldson, on "Modern Shaft Sinking," which was concluded in the May issue, is of great value to mine operators and engineers, as it contributes to the advancement of the art.

Some of the Advantages of Rectangular Over Elliptical Form for Concrete Lined Shafts

This is particularly true of the last chapter, dealing with concrete-lined shafts, as such lining is of comparatively recent development, and is an especially fit subject for discussion.

Most shafts of this type recently constructed have been of circular, elliptical, or other curved form; but Mr. Donaldson's suggestion of a rectangular concrete shaft is deserving perhaps of even more consideration than he seems inclined to give it; for reasons which will be explained.

The matter of comparative cost is important, and Mr. Donaldson's comparative estimates are of especial interest and value. Naturally, the results will vary for different cases, and it would be well to make comparisons for each contemplated shaft, before deciding on the type.

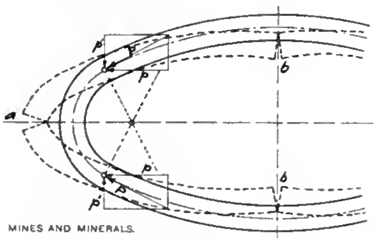


Fig. 1

In designing a shaft lining it is unreasonable to assume (as is done in the article) that the lining must resist the full hydrostatic pressure due to a head of water the entire depth of the shaft.

Water rings, as illustrated in the article referred to, in the February chapter, page 405, and in Mines and Minerals, April, 1910, page 561, should be formed at frequent intervals in the depth; then even if no other precaution were taken, this would effectually prevent the formation of a head of water of greater height than the distance between water rings.

In order to realize the conditions assumed in the article, it would be necessary to have the water completely sealed out of the shaft at the bottom, and backed up clear to the top in an unbroken sheet.

In certain shafts recently sunk, porous drain tile were placed behind the lining; one vertical line at each end, and one at the middle of each side. Of course more lines of tile can be used if desired. They discharge into each water ring, and certainly prevent the formation of a great head. It has also been suggested that bore holes be sunk and lined with tile, just clear of the lining. This would insure thorough draining.

It is recognized that it may be desirable to seal off the water encountered in a porous stratum, particularly if there be an impervious layer below, so that this water may be kept out of the mine entirely. In this case, it will be necessary to proportion the lining for a head of water extending from the impervious stratum up to whatever height the water is likely to accumulate. This height may be limited by the installation of a water ring above the water bearing layer. The thickest portion of the lining will then be at the bottom of the porous material. Below this point it may be reduced again.

The above remarks do not apply to the case of the aqueduct shaft where it is desired to exclude all ground water which might contaminate the water supply.

As explained by Mr. Donaldson, the exact pressure actually exerted on the lining cannot be determined.

It might be a good plan, and is suggested, that water rings be formed at vertical intervals of approximately 75 feet, and that four or more lines of drain tile be installed between and discharge into them; that the lining for the full depth be then proportioned for a static head of 50 feet. This would allow for quite a general stopping up of the drains. That this method is amply safe has been demonstrated by the successful completion of shafts with 12-inch lining of depth and radius such as would necessitate a thickness of 3 or 4 feet to resist the full head.

Before leaving this phase of the subject, it would be interesting to learn more fully Mr. Donaldson's objections to permanent weep holes, which he says are most undesirable. It is hard to see how weep holes would weaken the lining, and if water running down the inside of a timber lining is not objected to, why may it not be tolerated in a concrete-lined shaft?

The unit stresses used by Mr. Donaldson are surprisingly high. He adopts 2,500 pounds per square inch as the ultimate compressive strength of concrete in bending, and 3,000 pounds in direct compression, and uses a factor of safety of 3. This would give a working stress of 833 pounds and 1,000 pounds per square inch, respectively, when most authorities recommend from 350 to 750 pounds. It is unfortunate that the mixture used is not specified, but surely it is not economical to use a richer mixture in the shaft than is advisable in building construction. It is also to be noted that tests show that concrete is stronger in compression due to cross bending, than direct compression. This is indicated in the railroad specifications quoted below.

The uncertainties as to the actual applied loads, and the actual distribution of stresses are at least as great in the case of shaft lining as in other forms of concrete work; and the allowance to be made on account of practical limitations of workmanship and difficulty of inspection must be even greater in shafts than in buildings. Another unfavorable item is the fact that the full load is liable to come on the concrete sooner than in other work.

For unit stresses in general use, the following may be cited:

Specifications for reinforced concrete adopted by the American Railroad Engineering and Maintenance of Way Association, and reported in *Engineering News*, April 14, 1910, page 444. To be used in design; working stress 450 pounds per square inch for direct compression; 750 pounds per square inch for compression due to cross bending.

C. C. Schneider, Paper No. 997, "The Structural Design of Buildings," presented to the American Society of Civil Engineers, October 19, 1904. Working pressure in 1 : 2 : 4 Portland cement concrete walls, 350 pounds per square inch.

New York Bureau of Buildings, according to C. W. Hall in *Engineering Record* of January 22, 1910, 350 pounds per square inch for 1 : 2 : 4 concrete.

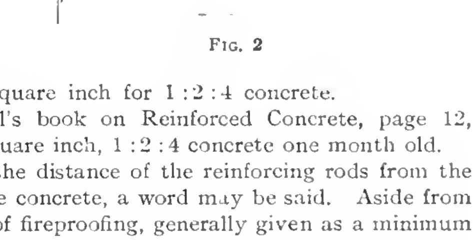


Fig. 2

Buel and Hill's book on Reinforced Concrete, page 12, 500 pounds per square inch, 1 : 2 : 4 concrete one month old.

In regard to the distance of the reinforcing rods from the finished face of the concrete, a word may be said. Aside from the requirements of fireproofing, generally given as a minimum of 2 inches of concrete, and which do not apply here, this dimension need be only sufficient to surround the rods with enough concrete to ensure their being properly gripped. This is frequently taken by the best designers as a fraction of the total depth of the beam, generally $\frac{1}{8}$ to $\frac{1}{10}$. So that for the case cited, assuming the correctness of the calculations, for a lining designed to resist a head of 50 feet, 2 inches would be ample.

Mr. Donaldson's demonstration of the forces acting on the

* 5121 Center Avenue, Pittsburg, Pa.

elliptical lining is very clear and accurate, but it should not be overlooked that this reasoning applies only to sections in solid rock, as clearly explained in the text. The unbalanced thrusts shown require unyielding abutments; otherwise bending stresses are introduced.

The extreme case is sometimes realized when quicksand is to be penetrated. In this case the lining must resist the outside pressure without any assistance from the surrounding material.

In Fig. 1, the unbalanced thrust P produces bending moments about the points a and b. The compression due to

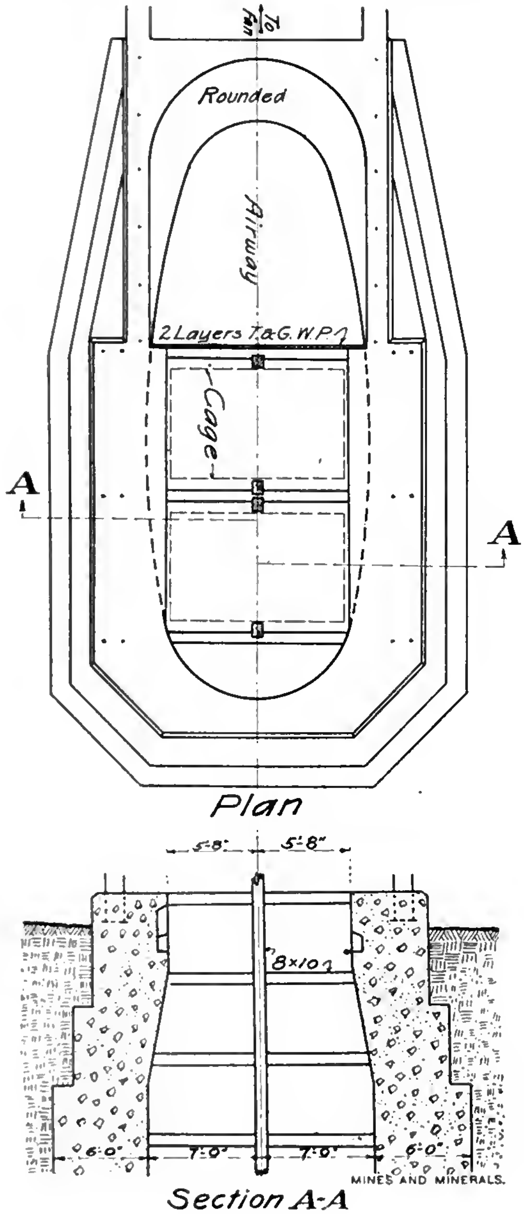


Fig. 3

this bending, combined with the direct compression, gives the maximum compression at these points. The ring of unit length tends to fail in the manner indicated by the dotted lines. The force P may be resolved into its components p and p', respectively, parallel to the longitudinal and transverse axes of the shaft.

It is evident that the effect of the force p' is to cause bending in the end of the shaft. This bending being a maximum at the point a, and producing tension in the outer portion of the lining, and increased compression in the inner.

The components p, may be considered as causing bending in the sides. This will be a maximum at the point b, and will cause tension in the inner portion, and increased compression in the outer.

The lining ring may then be considered in two parts as indicated in Fig. 2. These parts may be treated as explained in "Merriman's Mechanics of Materials" under eccentric loads. By the method there set forth the tension and compressive stresses due to these loads may be found, and added algebraically to the direct compression in the ring acting as an arch. Of course steel will be inserted to care for the resulting tension. It is evident that the relative thickness required at a and b will depend on the ratio of the radii. For a shaft such as is shown in Fig. 3, which is the top of a shaft recently completed, the thickness required at a may be greater than that at b; thus reversing the proportions given by Mr. Donaldson for sections in rock.

The extreme case may be considered to be that in which the long side is of so flat an arc as to be computed as a beam, and the short ends as eccentrically loaded columns whose loads are the reactions of these beams.

Deflections and consequent distortions of the curves will alter the problem and increase the stresses. This may be allowed for by reducing the safe unit stresses used.

The above solution of the problem is only an approximation, and it would be interesting to learn the views of others who have been called upon to meet this case.

A more precise and theoretical treatment may be possible, but in view of the fact that the actual amount of fluid pressure to be resisted is unknown, and on account of other uncertainties, an approximation is all that is required, it being only necessary that we assure ourselves that the proportions are such as to yield stresses on the safe side.

That is to say, on account of the uncertainties inherent in the design, we must add a certain amount of concrete to that obtained by calculations in order to feel safe. So that, in comparing the so-called elliptical with the rectangular design, the former has really less advantage, in point of cost, than is apparent in the comparisons given in the article; particularly if much material other than solid rock is to be penetrated.

Another point in this comparison which is not quite clear: How is Mr. Donaldson justified in reducing the area of the ventilating compartment in the case of the curved form? It is to be doubted if area in the hoistways can be counted as very effective for ventilation, on account of the baffling action of the cages in their rapid travel up and down. They act to some extent as pistons, violently forcing the air, first up, then down; or at least tossing it from one compartment to another.

For an interesting exhibit of the possible effect of hoisting on the air pressure of a mine, see Mines and Minerals of June, 1909, page 489.

Aside from the matter of cost, the rectangular design possesses other advantages over the curved form. Perhaps the most important of these is the fact that it can more easily be made fireproof than can the curved form.

The latter as illustrated in the article contains a considerable amount of wood in the form of guides, buntons, and partitions. Many shafts also contain stairs. A small fire originating at the bottom of the chimney-like shaft, feeding on this fuel, may quickly attain such proportions as to do great damage and cause the mine and the coke plant, which may be tributary to it, to be shut down for a considerable period. The air-way partition is frequently made of reinforced concrete even in shafts of curved outline in order to reduce this danger.

In comparing the relative cost of the two designs in order to place them on an equal basis, this partition should really be included in the curved form. It can be formed after the lining has been completed, by attaching stiffened lath, such as "Hy-rib" or "Trussitt" to suitable grounds, and plastering with cement mortar on both sides.

In the best designs buntons and stairs are made of steel in order to eliminate wood altogether. This is no doubt good

practice, but has its defects. Steel must frequently be cleaned and painted and probably renewed.

The concrete division walls between compartments suggested by Mr. Donaldson form a very desirable step in advance. They should entail no expense and therefore the rectangular design should be credited with the capitalized maintenance of the steel buntons as well as their extra cost over wood.

Another point of advantage of the rectangular form is that it can be made to conform as closely to the outlines of the cages as may be desired. In the curved form, the width of the shaft is not constant for the whole width of the cages, and therefore if the cage platforms be made alike and rectangular, there will be a large open space between the end of one of the cages and the shaft lining This is a source of danger in hoisting men and materials. It has been suggested that the cage platforms be made to conform in outline to the curved lining. There are, however, several serious objections to this. The cages will not be alike and interchangeable, and therefore more spares will be necessary. The landing devices will be complicated and unwieldy. With self-dumping cages it will be found very difficult to arrange chutes at the dumping point so that coal will not be spilled.

The best practice at present seems to be to bring the sides into a straight line at the top, as shown in Fig. 3. This is open to the objections that it complicates the form work, increases the material, and fails to eliminate the danger mentioned. The nature of this danger is well illustrated in the account of the accident at the Ellsworth colliery, in which two mine surveyors lost their lives. A timber being hoisted with the men struck a projecting bunton and, tilting, hurled the men to the bottom of the shaft. This was a timber lined-shaft, but it seems to the writer that the danger is even greater in the case illustrated in Fig. 3. Here there is a temptation to load, at the bottom, material longer than the cage platform, as the shaft apparently allows plenty of room for it. Such material may have ample clearance all the way up the shaft until it suddenly strikes the remaining portion with disastrous results.

In conclusion, if Mr. Donaldson's valuable paper should have the cost comparison revised as indicated, the result will be more favorable to the excellent rectangular design shown.

When considerations other than cost are given due weight, it is believed that the rectangular design will be found the one to adopt in many cases.

MECHANICAL COAL SAMPLER

*Written for Mines and Minerals, by C. E. Scott**

The coal-sampling machines in use at the Pratt Street power plant of the United Railways & Electric Co., Baltimore, Md., have been in continuous operation for more than a year, and have given results which have been satisfactory to both the railway company and the company furnishing the coal.

Description of Apparatus Used at Plant of United Railways and Electric Co., Baltimore

The sampling machine proper, shown in elevation, Fig. 1 and in plan Fig. 2, is almost identical with the one in use at the plant of the Interborough Rapid Transit Co., New York City, described in September issue. Instead of being automatic it is operated by the weighman, who releases the coal in the sampler by pulling a rope.

The coal passes from the sampler through a pipe to a crusher, fitted with plates for pulverizing the coal, and from here drops into the quartering box which consists of series of seven riffle samplers Fig. 3; the final sample dropping into a can which may be unscrewed and sent to the laboratory.

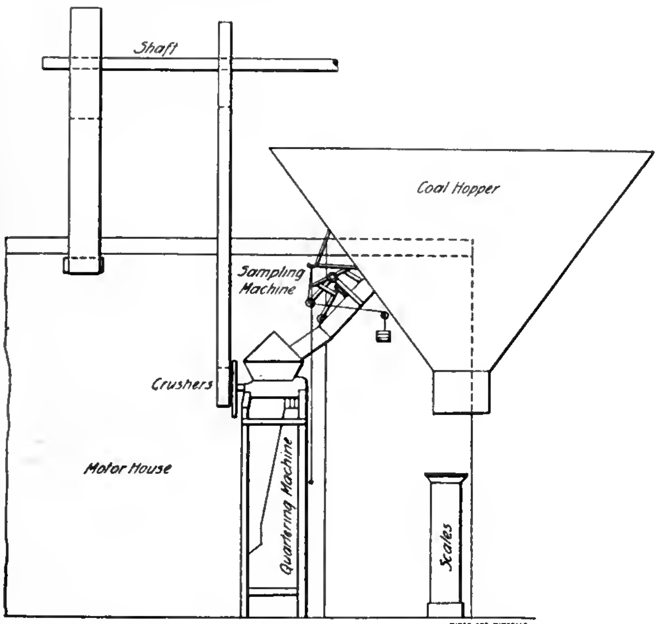


Fig. 1. Elevation of Mechanical Coal Sampler

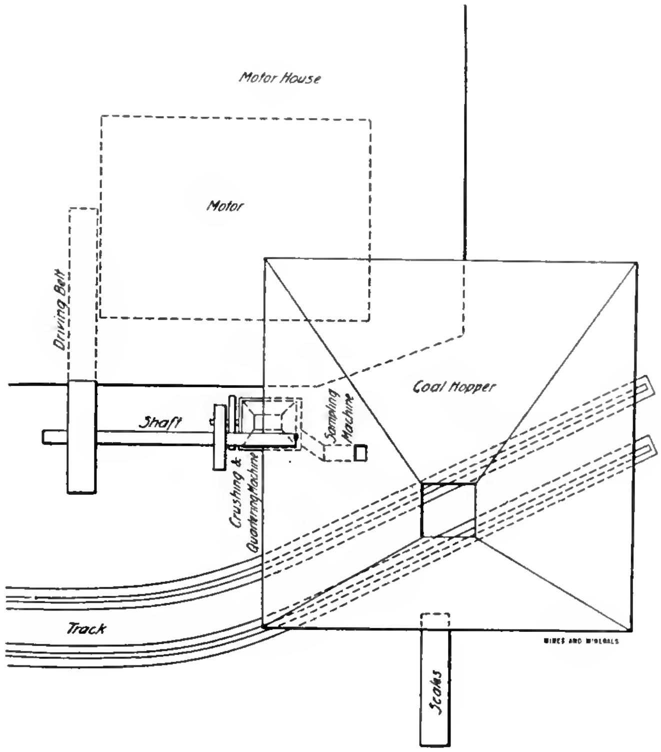


Fig. 2. Plan of Coal Sampler

Dust-tight galvanized-iron connections are made from the bottom of the sampler to the hopper of the crusher, and from the crusher to the quartering machine. A receptacle with a sliding door is also built below the quartering machine to catch the discard. The sample is not exposed to the atmosphere after it passes into the machine and loss of moisture in this direction is thus reduced to a minimum.

The quantity of coal taken for a sample depends upon the number of times the rope operating the machine is pulled, the size of the gross sample desired may be regulated to suit the demands.

At the United Railways plant the gross sample is approximately $\frac{1}{1000}$ of the total weight of the coal sampled. For example, from a barge containing 200 tons, the total weight of sample taken is 400 pounds. This sample after passing through the apparatus weighs from 3.1 to 3.2 pounds, and consists of coal which will pass a number 10-mesh seive.

*Chief of Testing Department, Consolidated Coal Co.

The uniformity of the samples taken with the machines, both as to total weight of sample and sizes of coal which compose the samples, is shown in Tables 1 and 2. From these it may be seen that the machine-taken sample shows almost constant percentage weights of each different size on successive tests,

Table 1. Machine Samples

Test	A	B	C	D	E
Number cars dumped	47	30	69	30	30
Weight of sample, pounds	253.30	151.30	350.00	150.90	173.20
Weight of sample per car dumped, pounds	5.01	5.04	5.07	5.03	5.77

Size of Coal Before Going Through Crusher

	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Over 1 inch	33.7	37.0	36.5	34.7	41.8
1 inch to ½ inch	23.9	24.6	24.9	23.4	22.4
½ inch to ¼ inch	16.0	15.5	15.9	14.2	15.4
Under ¼ inch	26.4	22.9	22.7	27.7	20.4
	100.0	100.0	100.0	100.0	100.0

while the samples taken with a shovel, show no uniformity in either total weight or percentage of sizes.

The uniformity with which the quartering machine does its work is shown in Table 3. These figures are the results attained on one sample of coal which went through the quarterer 10 times. The operation was repeated with other samples.

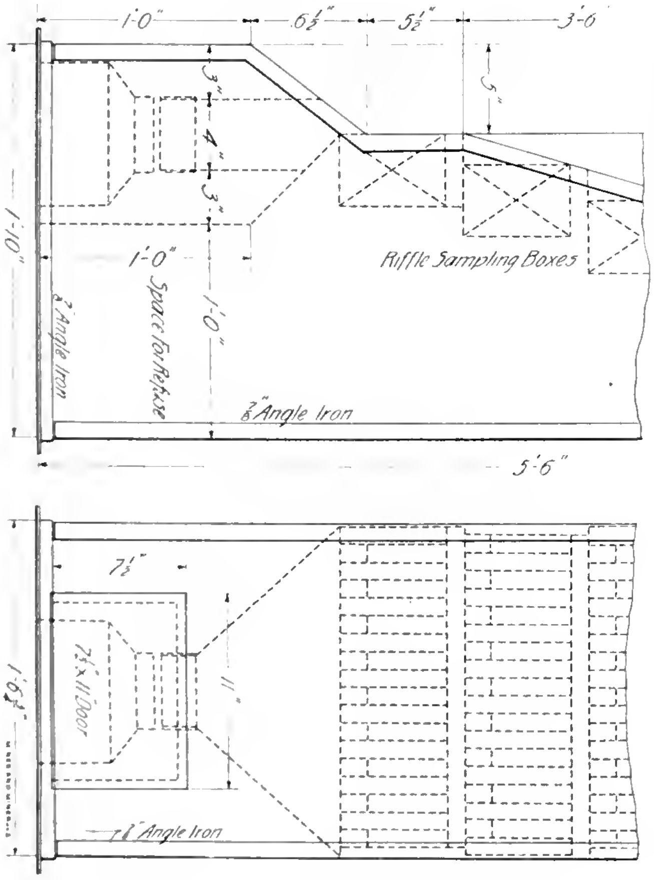


Fig. 3. Plan and Elevation of Riffle Samplers

The ratio of the weights of the samples to the weights of the original quantity of coal is almost constant, as shown in Table 3.

While it is true that the size of the gross sample depends upon the man who loads the tram cars and weighs the coal, the process is mechanical, and there is nothing to cause a variation in the quality of the sample. The machines are efficient and accurate and at their worst are better than the method of hand sampling, which has produced results so radical as to cause endless discussion between the consumer's chemist and the coal company.

Table 2. Shovel Samples

Test	A	B	C	D	E
Number cars dumped	47	30	69	30	30
Weight of sample, pounds	183.30	91.30	258.20	80.40	82.70
Weight of sample per car dumped, pounds	3.90	3.05	3.74	2.68	2.76

Size of Coal Before Going Through the Crusher

	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.
Over 1 inch	38.7	21.9	42.4	38.2	35.3
1 inch to ½ inch	18.6	16.9	17.9	18.6	20.2
½ inch to ¼ inch	14.0	18.0	13.4	14.1	15.1
Under ¼ inch	28.7	43.2	26.3	29.1	29.4
	100.0	100.0	100.0	100.0	100.0

Table 3. North Tower

Weight of Coal Ounces	Parts	Weight of Sample Obtained Ounces	Theoretical Weight of Sample Ounces
1,763	1 in 125	14.0	13.8
1,712	1 in 126	13.5	13.4
1,672	1 in 126	13.0	13.1
1,656	1 in 127	13.0	12.9
1,624	1 in 126	13.0	12.7
1,592	1 in 123	13.0	12.4
1,584	1 in 126	12.5	12.4
1,568	1 in 124	12.5	12.3
1,528	1 in 127	12.0	11.9
1,508	1 in 125	12.0	11.8
South Tower			
1,851	1 in 126	14.5	14.5
1,814	1 in 125	14.5	14.2
1,786	1 in 127	14.0	13.9
1,779	1 in 127	14.0	13.9
1,763	1 in 130	13.3	13.8
1,742	1 in 134	13.0	13.6
1,723	1 in 128	13.5	13.5
1,702	1 in 131	13.0	13.3
1,636	1 in 124	13.5	13.2
1,670	1 in 133	12.5	13.0

One month's record of analyses totaled and then averaged gave the following percentages: Moisture, 1.26; volatile material, 18.96; fixed carbon, 72.33; ash, 7.37; sulphur, .85. The uniformity of the results is indicated by the ash, which varied from 6.28 low to 7.97 high, and averaged 7.37 during the 31 days.

NEW WESTERN PACIFIC RAILROAD

The Denver & Rio Grande Railroad has issued a new coast to coast map, which should be in every mining engineer's office, East and West. It is descriptive of the new Western Pacific Railway, an extension from Salt Lake City to San Francisco, a line 921 miles long that has been in course of construction since 1905, and was opened for passenger traffic August 22, 1910. For 80 per cent. of its entire length the heaviest grade is $\frac{4}{10}$ of 1 per cent., about 20 feet to the mile, while the maximum grade is 1 per cent. The maximum curve is 10 degrees, and few curves exceed 6 degrees. The new line crosses the Sierras at an elevation of 5,712 feet, which is 2,535 feet lower than the Sierra crossing of any other railroad. No snow sheds are needed on this account.

There are 40 steel bridges on the line, aggregating in length 9,261 feet. There are 43 tunnels, with a total length of 45,494 feet. On the western slope of the Sierras, in order to maintain a light and uniform grade, the line rising 41 feet, loops itself within 1 mile, which forms one of the wonderful engineering features of the new road.

MINING, PREPARING, AND COKING COAL

Written for Mines and Minerals, by E. B. W.

A Description of the Methods Used and the Plant at Marting, W. Va.

The origin of coal has been traced from peat bogs to graphite and natural coke, in fact, carbon the chief constituent of coal has always been derived from organic matter. In San Domingo there is a bed of dun-colored material which is about as heavy as punk and has no resemblance to coal, yet on being ignited with a match burns and gives off the same odor as black cannel coal. This has a conchoidal fracture and might be termed white cannel coal. In the Richmond, Virginia, coal basin natural coke is found, and in Rhode Island, anthracite resembling graphite is obtained, all of which goes to show that in different places coal varying in color, texture, and chemical analysis will be found. The geology of coal is so well established that one accustomed to the rocks can determine them by casual observation from a railroad car when he is traveling through coal formations. Not all coal measures are productive, and in a new field prospecting is absolutely necessary for determining the thickness and area of a coal bed.

A chemical analysis is also a necessity in order to determine the impurities in a coal and draw conclusions on its apparent value from an economic standpoint. At one time coal was believed to exist only in that geological period which is termed the Carboniferous; later it was ascertained that excellent coal was to be found in the Cretaceous and Tertiary periods; however, no coal worth mentioning has been found below the Carboniferous formation, because prior to this time very little vegetation existed. The coal beds of the Appalachian and Blue Ridge mountains belong to the Carboniferous period, and while they are in layers, one above the other, with sandstone, slate, and fireclay intercalated, they are of different qualities. The same coal bed, even though it varies in thickness and has a great area, differs but slightly in chemical and physical qualities, so that when one versed in coal hears of a particular coal bed like Pocahontas No. 3, he understands the value of the coal whether it comes from the Elkhorn or New River districts in West Virginia. As another illustration, when one versed in coal hears of the No. 2 gas seam in West Virginia he knows its quality, whether it is mined in the Tug, the Guyandotte, or the Kanawha districts.

If the anthracite fields of Pennsylvania be neglected, most of the coal beds of the Appalachian region are almost horizontal, and in West Virginia many of them are above water level so that the coal may be mined by drifts. There are several advantages in drift mining which tend to decrease the cost of coal production, although as a rule, such mines are so far from market this advantage is overcome, and were it not for the careful preparation and excellent quality of the coals there would be little profit in mining them. Fortunately the Pocahontas No. 3 and the No. 2 gas coals are so different in quality and situation there is little competition between them, although both have strong competition from other fields. In recent years many labor-saving devices have been introduced in coal mining to decrease the cost of production and prepare coal for the market; few changes, however, have been made to improve the method of mining and reduce fixed charges. It is well known that the proper way to develop a coal property is to drive the main entries to the end boundary line and at a suitable distance, say 300 feet from this line, turn cross-entries to the side boundary lines. As the cross-entries are driven, room necks should be turned but no room mining done; after this has been accomplished a second pair of entries 300 feet from the first should be started inside the first pair, or both entries may be driven at the same time. All rooms on the first pair of entries should be started at the same time, and when they reach the boundary all room pillars should be pulled back together. If it has been customary to mine a 20-foot room and leave a 40-foot pillar in any particular field, the practice under the proposed method may be changed, say to 15-foot rooms and 15-foot pillars. In the case of the larger pillars, made necessary if they are to stand long, as almost invariably is the case where the attack is made soon after entering the mine, but one-third of the coal is won and two-thirds is left standing; therefore, in a half-mile cross-entry but 44 rooms can be opened. If coal is needed, probably not more than 10 or 12 rooms will be working on one entry at a time, because they will be attacked soon right after the entry drivers have passed them, and to increase the number of rooms another pair of entries will be started. By this means the mining becomes spread over a wide area, thereby increasing costs in several ways.

Fig. 1. Surface Plant at Marting, Fayette County, W. Va.

In the proposed case, with the smaller pillars 88 rooms can be worked at once, thus centralizing the work, and decreasing costs in every way to such an extent that the first extra outlay of capital is returned many times over. Labor, supplies, ventilation and haulage costs will be decreased; better safety to the mine insured; besides, instead of costs increasing as the mine expands, as they do on the working-in system, they will be diminishing, and better coal will be obtained by practising the retreating system. This is not a theory but a well-known fact, which has failed to be adopted in the United States because most mine operators in the past, through lack of capital, have devoted their energies to increasing their cash accounts by quick returns, and the remainder have been driving their employes to show temporary results.

If one considers the cost of narrow work and the extra expense that entails when it stands idle for years; the extra rails on the entries that are not used so long as the pillars are not

mined; the extra-room tracks that must be pulled and relaid; the loss of ties and props; the labor in fixed charges; doors, overcasts, and other arrangements for ventilation that can be discarded, it will be found that large sums of money can be saved by the retreating system of coal mining.

Fig. 2. Loading Hand Picked Splint Coal

Bituminous coal companies in Pennsylvania and West Virginia have made great strides in the last 15 years toward improvement in the preparation of coal for market. These improvements have been noted from time to time in Mines and Minerals and in this particular article the plant of Columbus Iron and Steel Co., at Marting, Fayette County, W. Va., is illustrated. Fig. 1 shows the general arrangement of the buildings, with slack bin, shop, stable and power house, in order from left to right. It also shows the trestle from the slack bin to the coke ovens and the tipple to the rear of the slack bin.

The coal mined by this company is the Olive splint and the celebrated Kanawha No. 2 gas coal, which has the following analysis at their mines: Fixed carbon, 60.15 per cent.; volatile matter, 31.90 per cent.; moisture, 1.05 per cent.; ash, 6.90 per cent.; sulphur in ash, .75 per cent.

At Marting the average thickness of the coal bed is 70 inches, the upper 20 inches of which, however, is a hard semi-smokeless splint coal. As splint coal is much harder and less frangible than the coking coal of the lower bench, it is separated in the mine as shown in Fig. 2. Splint coal has a laminated structure which prevents its breaking readily across the bedding planes. This physical condition keeps the coal from slacking during handling and is one feature that makes it a desirable domestic fuel. When splint coal is kindled, little splinters fly from it. This decripitation is due to the sudden heating which causes unequal expansion, or to the pressure of minute cavities of gas, which expand and explode. The Smithers Creek Olive splint coal is used at metallurgical plants for remelting, reheating, and annealing purposes, where long rolling-flame coals, low in sulphur, are in demand. At Marting the splint coal is prepared by carefully selecting the lumps in the mine, loading it by hand into separate cars run on wooden rails over the mass of broken coal from the lower bench, as shown in Fig. 2. When the coal reaches the tipple it is passed over shaking screens having 5-inch diameter holes.

In the preparation of the lower bench or 50 inches of No. 2 gas coal the splint coal is entirely eliminated. The No. 2 gas coal is passed over ¾-inch shaking screens previous to its going to the railroad car and is as free from slack as it is possible to make it. Usually run-of-mine contains all sizes of coal but this "¾-inch gas mine-run" receives the same careful preparation as egg and lump coal, in order that it may fully come up to the steam users' and gas makers' expectations, and always be uniform and dependable in quality. Three sizes, lump, egg, and pea coal are made at the steel tipple, shown at Fig. 3. The greater part of the slack is riddled from the coal before it reaches the weigh basket, and whatever slack is made at the shaking screens passes through them to a slack pocket in the tipple from which it is lifted by a conveyer and carried to the slack bin. Any bituminous coal heated in a retort will furnish gas, but the coal with the highest percentage of hydrocarbons will produce the most gas. The results of experiments made by the United Coke and Gas Co. on the by-products to be obtained from the leading gas coals in the United States and Canada is given in Table 1.

From the results obtained it is seen that the No. 2 gas coal of the Kanawha River district leads in the quantity of by-products, but not in coke, as might be anticipated, when coals lower in volatile hydrocarbons are compared with it. It is frequently stated that the value of a coal is indicated by the number of heat units it contains, as determined by a

Fig. 3. Tipple at Marting, W. Va.

calorimeter. While as a theoretical hypothesis this statement may approximate the truth, as a practical proposition applied to the combustion of coal beneath a boiler it is erroneous, and should be made to read, "the value of a coal as a steam fuel depends on the way it is fed to the grate and the kind of boiler it is burned under." A British thermal unit, B. T. U., is the heat required to raise 1 pound of water 1° F., and is the basis on which the heat in coal is calculated. Theoretically 1 pound of carbon should produce 14,600 British thermal units; however, since coal is not pure carbon, the heat units may be increased or decreased by the impurities in the coal. Thus the heat units per pound of combustible in No. 3 Pocahontas coal is given as 15,700; that of Thacker coal, which is somewhat higher in volatile hydrocarbons than No. 2 gas coal, is given as 15,200; while that of Olive coal is given as 14,632.

The Columbus Iron and Steel Co. have blast furnaces at Columbus, Ohio, and consume the entire product of their coke ovens, which are shown in Figs. 4 and 5. The illustrations are so good they are inserted for the benefit of those not familiar with coking operations in beehive ovens. The oven side and end walls are all that can be seen but inside these walls are hemispherical ovens built of firebrick, having a door opening through the side walls. Two ovens are built back to back making a long double row with oven doors on each side wall. The oven crowns have a charging hole in the top and all but this hole is covered with loam to prevent the heat of the oven radiating and to make it possible for men to travel over them.

On piers built lengthwise of the ovens there is a railroad on which the larry runs. In this case the plant machinery is run by electricity and the larry shown is run by a motor, where a few years ago small steam locomotives or mules drew the larry. After the ovens are charged with slack from the larry they are leveled, the door closed with brick, and the joints in the brick through which air might pass, are luted with clay. Above the brick door there is a small space left to furnish air for the combustion of the gas which rises from the coal. In Fig. 4 will be seen half-buried barrels for the clay lute. The coal is subjected to the oven heat from 48 to 72 hours, according to the size of the slack charge and the kind of coke it is desired to make. To pull the coke from the ovens the temporary brick doors are broken down, a stream of water allowed to play on the hot coke inside the oven until it is extinguished; after which the coke-drawer uses a long-handled hoe and pulls the coke from the oven. In Fig. 5 the first door is seen bricked and luted; the next has the bricks pulled down and is ready to pull, having been watered; the next oven is being pulled; the next oven has been pulled and the coke stacked on the yard; the next oven is being watered, as is evident from the steam rising through the tunnel head on top. The man in the rear is just commencing to water his oven, as may be seen from the steam coming out of the door and the distance he stands away from the oven.

FIG. 4. COKE OVENS

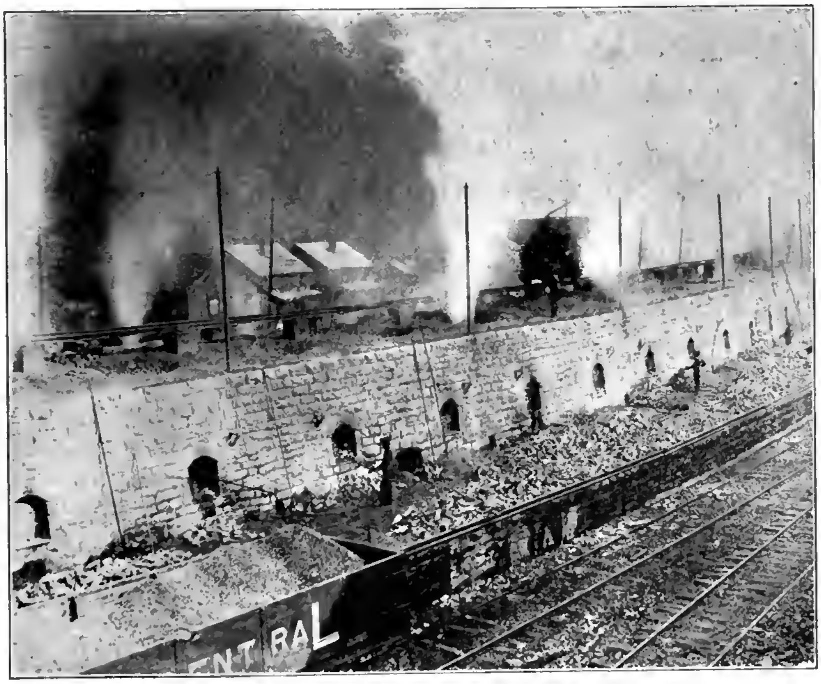


FIG. 5. COKE DRAWING

The nearer a beehive oven is made like a navel orange when bisected between its ends, the stronger and more serviceable it will be. The navel orange is not an exact sphere, nor is it flattened at the ends, yet not too oval to greatly exaggerate a hemisphere. When the crown of the hemispherical oven commences to sag it soon comes down, while a navel-orange-shaped oven will sag slightly and still behave. Ovens constructed on the later design remained unfired for 10 years, which is surely an endurance test, and since firing they have been running 5 years.

In Fig. 6 is shown the interior of the company's power house, in which electric power is generated to light the town and works, haul cars and larries, run the machinery at the tipple, etc. In 1908 the Columbus Iron and Steel Company produced 81,837 tons of coal, made 24,150 tons of coke and shipped 45,105 tons of coal. In 1909 the same company

produced 129,048 tons of coal, made 34,868 tons of coke and shipped 71,851 tons of coal. This large increase is due to the natural growth and development of a good coal property.

Table 1. Products from Gas Coals

Coal	Coke Per Cent.	Tar Per Cent.	Amm. Sul.	Total Cubic Feet Gas Per Ton Coal
Kanawha	73.60	6.40	1.010	10,289
Pittsburg	68.25	4.38	.903	8,884
Eastern Pennsylvania	78.00	2.00	.800	8,400
Virginia	66.01	4.70	1.070	10,090
Youghiogheny	75.60	4.99	1.000	9,000
Fairmont	72.58	5.22	1.270	10,200
Cape Breton	72.83	4.99	1.270	9,000

The writer is indebted to Mr. Fenton, of the Columbus Iron and Steel Co. for the use of the illustrations which enter into this article.

Fig 6. Interior of Power House

CARBIC CAKES

The "Carbic" cakes for acetylene lighting of all kinds are made of a mixture of granulated calcium carbide, sulphur, and sugar, compressed into cylindrical cakes in a powerful press. As a result of this treatment the fragments of carbide are surrounded with a protective covering which renders them proof to a large extent against atmospheric moisture, decomposition taking place only when they come in contact with moisture in the form of liquid. In this way the handling of the material is simplified and waste diminished. Further, these cakes present certain advantages in use. With carbide in the ordinary form there is apt to be a rapid evolution of gas as soon as it and the water are brought together in the generator, and in consequence the pressure may be unduly high. With the cakes, however, since the surface reached by the water consists only in part of carbide, the gas is produced more gradually and the pressure is more equable. The other ingredients in the cakes also cause the lime, which is formed by the reaction, to fall away freely from the cake to the bottom of the generating vessel, whence it can be easily washed out. In the generators designed for use with the cakes the latter are suspended in a cage and the water is forced up to them by the pressure of the gas itself, so that the generation is automatically adjusted according to the consumption. The acetylene, after it is formed, is first washed by passage through water and then chemically purified. In consequence of this purification and of the regularity of the pressure, it burns with a clear white flame, free from "haze," and does not give rise to carbonization and choking of the burners. For motor cars a device may be fitted by which the jets can be electrically lighted from the magneto.—*London Times.*

ELECTRIC MOTORS FOR ROPE HAULAGE

*Written for Mines and Minerals, by H. W. Reybold**

So much interest has been shown in the article on "Electric Rope Haulage" which appeared in the July issue, and so many requests have been received for detailed information that it has been considered worth while to give a more complete description of the steel frame hoists used, these, of course, being the most important machinery used in the work. In the article mentioned, the system of electric rope haulage in use at the Midway Mine of the Chicosa Fuel Co., Huerfano County, Colo., is very clearly described. The installation consists of single-drum electric hoists on cross-entries, and double-drum electric-driven tail-rope engines, or hoists, on levels. Each hoist handles easily one pair of cross-entries or about 48 rooms, while the tail-rope engines care for all the cars coming from 2 pairs of cross-entries or about 96 rooms working 192 men. The hoists set out the empty cars and pick up the loaded cars at the room mouths. One engineer, 1 rope driver, and 1 hoist can move as much coal as 8 or 10 drivers and mules, without the risk of killing both mules and drivers.

The hoists, as shown, are built with the entire frame of standard steel shapes. The advantages of the steel construction are readily seen, the frame can be built stiffer and more rigid, with about one-third less weight than equivalent cast-iron frame (a 30-horsepower hoist with cast iron-frame weighs 5,900 pounds while the same size with steel frame will weigh only 4,150 pounds); the frame is literally unbreakable; the hoist can be knocked down and rebuilt in a short time with little expense—a decided advantage for mountain transportation where the machine must be taken underground.

The base of the machine consists of four **Z** bars bolted together forming hollow girders with projecting bottom flanges on each side which bear on the foundation. The holes for the foundation bolts are drilled in these flanges.

The bearings for the drum are carried on two standard I beams, Fig. 1, which are in turn bolted to the four **Z** bars in the base. The motor, Fig. 3, is bolted to two small **Z** bars, which are also bolted to the base. These latter can be bolted at any desired point on the base to accommodate any style of motor that could possibly be used on the hoists. This construction lends additional stiffness to the frame, and is superior to the separate motor pad. In cases where it is advisable to place the motor either in front or in the rear of the drum, this construction is easily modified, without sacrificing any features of the standard design.

The hoists have a new patented band brake, the construction being such that the band bears on about 87 per cent. of the band ring. The power is applied from the hand lever to the brake by means of a simple, but powerful system of multi-

* Engineer for Hendrie & Bolthoff.

plying leverage, all the members of which are steel forgings. The band is supported by a bolt and spring, leaving the drum free from any drag when hoisting. The band is lined with Gandy belting which is fully as durable as wood blocks and can be renewed by any one in 30 minutes. The brake ring, or surface, on which the brake band bears is of large diameter and ample width, insuring plenty of wearing surface and reducing the liability of heating. The brake is on the opposite end of the hoist from the clutch, and the latter cannot possibly be affected in any way by the heat from the brake

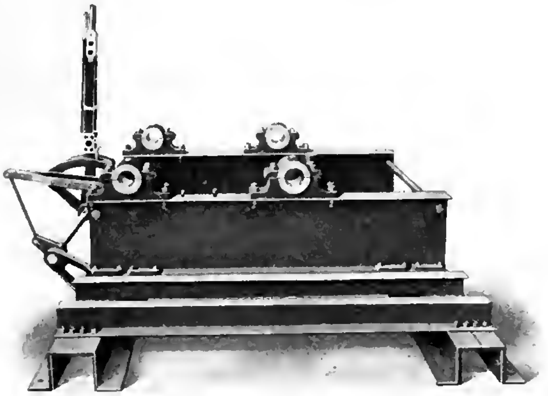


Fig. 1. Base and Drum Bearings

The clutch used on the hoists is an improvement on the well-known Lane clutch. The power is applied radially to cranks on the two halves of the clutch at the same time, releasing or tightening the band as desired. The clutch band bears directly on a flange on the drum itself, transmitting the power from the motor directly to the main gear and through the rim of the gear to the drum. The clutch is powerful and easy to adjust and maintain.

All the gears are extra heavy and milled from the solid metal. The Chicosa Fuel Co. now has at its various mines five of these hoists with single drum, and three with double drum as shown in Fig. 3.

The power used, as stated in the article in the July issue, is three-phase, 440 volt, and is supplied by two 125-kilowatt generators driven by 16″ × 16″ self-oiling engines.

NATURAL GAS AND PETROLEUM IN ALABAMA

For more than a year the existence of natural gas in considerable quantities has been known, in a small area 3 miles east of Fayette, Ala. This region was recently visited by David T. Day, of the United States Geological Survey, who reports that four successful wells have been drilled, showing gas with a closed pressure estimated at 600 pounds per square inch, and nine additional wells are in process of drilling.

The gas is found at a depth of 1,400 feet in a close-grained sandstone, estimated by the Providence Oil and Gas Co. to be 50 feet thick. Small quantities of crude petroleum similar to that usually found in the Appalachian region were encountered at various depths above the gas. The strata pierced in the drilling consist of alternating layers of shale, sandstone, and tight sticky clay, under ideal conditions for the accumulation of gas and oil.

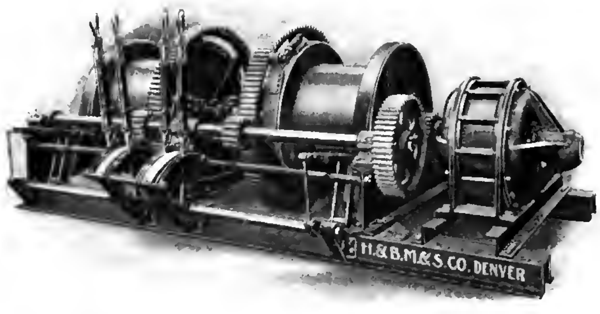


Fig. 3. Double Drum Hoist

It is estimated that five million cubic feet of gas per day can be furnished from the four wells now completed. The gas is pure and is free from sulphur.

It is difficult to determine the extent of the field on account of the overlying cap of Tuscaloosa drift by which the oil-bearing formations are concealed.

The conditions for successful gas wells are ideal in the immediate neighborhood of the present wells but it is impossible to determine how far these conditions may extend without interruption by faults or other disturbance of the necessary impervious cover. The thickness of the gas-bearing stratum and the pressure developed indicate a satisfactory yield of gas as compared with other known gas fields. The amount of development work thus far done is not sufficient to show whether a supply adequate to the needs of Birmingham for domestic and manufacturing purposes can be obtained. This will be determined, however, by the completion of the wells now drilling. Mr. Day urges the importance of detailed geologic mapping of Fayette County by the Alabama Geological Survey.

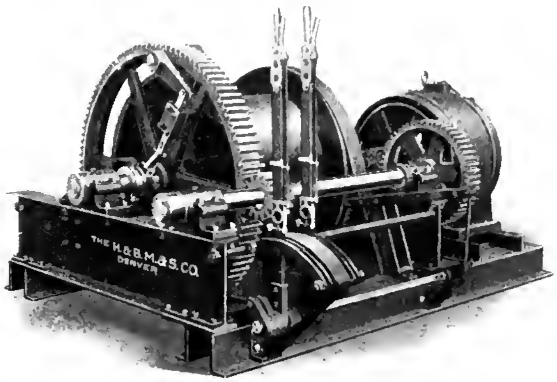


Fig. 2. Single Drum Hoist

POSSIBLE EFFECT OF THE PANAMA CANAL ON ALASKAN COAL BUSINESS

The completion of the Panama Canal may enable eastern operators to deliver high-grade coals on the Pacific coast at prices about the same as those that can be offered by operators in Alaska. Other competing fields will be those of Vancouver Island and New South Wales.

The present markets for Alaska coal may be grouped under three heads—the local market, without competition, about 120,000 tons a year; a market competitive, yet favorable, about 350,000 tons; and a competitive market, about 1,000,000 tons. These rough estimates indicate that Alaska coal of the better grade could perhaps find a market to the extent of 1,000,000 tons a year.

It is impossible to forecast how rapidly the market for Alaska coal may expand, for its expansion depends on the rate and amount of industrial advancement made along the Pacific seaboard. That the demand for high-grade steaming and coking coal will increase rapidly there can be no doubt, but that Alaska fuel will be a strong competitor with some of the imported coals and also with the eastern coals after the completion of the Panama Canal seems equally certain.

THE COEFFICIENT OF FRICTION

*By William Clifford, M. E.**

In a recent issue of Mines and Minerals one of the correspondents endeavored to show that the total pressure required to pass a predetermined volume of air through a coal mine could be obtained by measuring the rubbing surfaces and using Atkinson's formula and a certain German coefficient.

Difficulty of Calculating the Necessary Pressure by Estimating the Rubbing Surface

If it is claimed (which is not stated) that the projection of a mine, in case of a new one, should be taken, I would like to see the mine that is worked to its original projection.

A good many years ago Mr. Austin King, Sr., endeavored to test the practical value of Atkinson's coefficient by measuring roads. To this end Mr. King selected a fairly new colliery with straight roads and good roof, and made his experiments when all was still, and I understand he obtained a gauge as high as a house for the same volume of air the mine actually passed with under 3 inches water gauge. This was about the time when in Pennsylvania mine inspectors and mine foremen began to be subjected periodically to that form of mental calisthenics called examinations.

The application of formula and coefficients to a whole mine is of no value in guiding us to the obtaining of the amount of

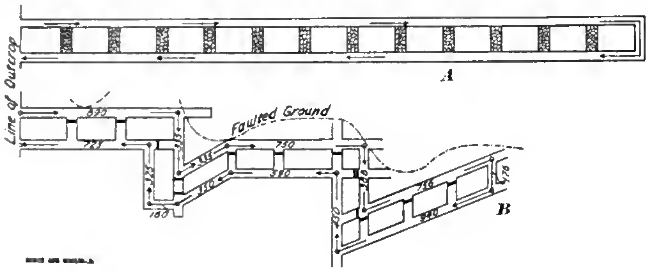


Fig. 1

pressure or motive column we must use; and if we inquire into the laws governing the friction of air in mines, as now laid down by authorities, and first put before the mining world in a concise form by the late John Job Atkinson, in a paper read before the Manchester Geological Society, nearly 50 years ago, it will be found that it was in this paper that Mr. Atkinson gave to the world his famous equation:

$$p = \frac{k\,s\,v^2}{a}$$

In this paper (which is published in small book form) we shall find, that at the very outset it reads:

"Numerous experiments have been made to find out the laws that govern the friction of air and gases, both in pipes having a uniform section, and to a less extent in the irregular airways of mines. By these experiments the following laws have been found to hold good in practice:

1. "The pressure required to overcome friction of air increases and decreases in exactly the same proportion that the area or extent of rubbing surface exposed to the air increases or diminishes." (I would add that this is true only so long as the rubbing surfaces and the initial direction of the air are parallel.)

2. "That the pressure varies inversely as the sectional area of the airway.

3. "That the pressure required to overcome friction in the same airway varies in the same proportion as the square of the velocity increases or diminishes." (I would add the same qualifying condition as in the case of Law No. 1.)

That Mr. Atkinson had in his mind individual, straight airways of uniform section, and not a district or whole mine, embracing hauling roads, turnouts, room necks, and roads, and the return airways, is plain from the way he states his examples.

For instance, in the very first one he says: "In an airway 10 feet square—100 feet area, and 25,000 feet or nearly 5 miles long," etc.

I think it is proper here to say that nothing is more unfair to the memory of so bright a man as the author of the little book "Gases Met With in Coal Mines," who sacrificed his material interests during his lifetime to the elucidation of problems affecting the safety of coal mines, and in the spreading of knowledge to that end, than to father upon him the crudities of misrepresentors of his theories. I must leave to my readers the decision of whether or not it is possible that a writer, whose lucidity of thought and expression has captivated the mining students in all countries into whose language the little manual we have referred to has been translated, would fall into the error of misapplying the rules specifically governing the flow of air in straight roads of even section, to the mixed conditions and irregular areas and surfaces of a whole mine.

Mr. Atkinson's knowledge and views were not merely academic, but were the outcome of many years of daily practice as chief manager of mines of the highest standing, and as a government inspector in the county of advanced mining, Durham, England, at a time when inspection of mines in Great Britain was very much of a parental character, and the men selected for the duty were usually eminently qualified for the role of authoritative advisers.

I have drawn two plans showing two different forms of exploitation, in order to make plain my contention that every colliery that is developed has its own particular coefficient, distinctly different from every other, and that it is safer to take the estimate of an experienced mining engineer than to rely on figures obtained by straining authorities and stating as facts things which are strictly speculations. After inspection of the two plans, the decision as to whether or not my first contention is correct, the reader can judge.

In the two mines, shown in Fig. 1, marked *A* and *B*, the roadways in each case aggregate 7,149 feet in length, and are assumed to have equally smooth walls and to be of identical cross-section throughout their length, say 60 square feet. It is desired to circulate 6,000 cubic feet of air per minute through each mine, and applying the method of whole-mine coefficient, the pressure required to put this 6,000 feet through mine *A* must equal that required to put it through mine *B*.

In mine *A*, stoppings are built at both ends of the breakthroughs, nearly flush with the face of the respective entries, for the reason that in case of an explosion the expanded gases caught in the cul-de-sac of an open-ended breakthrough, aided by the ignition of the local deposit of dust usually present, will invariably blow out the stopping at the closed end. The small openings left at the top of each pair of stoppings are 9 square inches area, which is enough to keep the spaces in the stowing clear, except in a few highly gaseous mines. The openings may ordinarily be closed after 2 or 3 years.

The mine *B* supposes a less fortunate location than *A*, and is feeling its way along the edges of an imaginary fault, and though driven on paper with the object of being irregular, it is less so than many coal mines, but it sufficiently answers the purpose of showing the futility of endeavoring to compute the total resistance of a mine by measuring the roads and applying Atkinson's formula with any of the modified coefficients.

We will now trace the air-current round these two mines, beginning with *A*. Entering at the upper road or intake, the air sweeps along in a straight line. The frictional resistance being that normally due to the velocity with which it rubs against the sides, roof, and floor, varied by striking against vertical and horizontal timbers, and perhaps under cribbing supporting high falls, or projections due to slabbing of the sides; these latter two forms of resistance afford compensations in the shape of locally increased areas.

* Jeannette, Pennsylvania. Paper presented before the Coal Mining Institute of America.

It may also be added that the workmanship of the collier enters into the attainment of results; smooth cutting and freedom from offsets, large or small, have a good deal to do with capacity of a road to pass wind. Our current in mine *A* reaches the far end of the intake, around two right-angle turns into the return; from this point to the surface it encounters substantially the same resistance that it did coming along the parallel intake road.

We will now follow the course of ventilation round mine *B*. For about 950 feet from the surface, the road is straight, following the direction of the general projection of the mine. At this point in the entry, a fault, which had shown itself on one side at 400 feet, stretches across the entry. Coming back to 890 feet a road is turned away from the fault at right angles.

(It may be here observed that none of the authorities on coefficients have given one for a right-angled turn. Daniel Murgue gives one about one-third greater for a "slightly sinuous" road than for a straight one.)

The road turned off at 890 feet continues straight to a blind end beyond the back heading; but at 335 feet a road is turned off toward the fault, which it touches at 335 feet more. Here a deflection is made to a line of direction approximately that of the base line of the mines alignment, and along the road so deflected the fault is struck again at 775 feet, and the driving continued 50 feet into the faulted ground. Back at 770 feet a right-angled heading is turned away for 368 feet, and an acute-angled road is turned off and continued for 850 feet, in the hope of getting round the end of the fault, but it is reached. At 758 feet, a cut through to the back entry is driven 175 feet. The length of the intake up to this point, plus the cross-cut 175 feet, together with the measurements along the return or back entry to surface, make 7,149 feet, the length of mine *A*.

In mine *A* are two right-angled turns. In mine *B* are five right-angled turns, four acute-angled turns, and four oblique- or obtuse-angled turns. Now if "slight sinuosity" adds 33 per cent. to the resistance of a road (Daniel Murgue's coefficient), what must a turn of 90 degrees add, and what more must a turn of about 115 degrees add, and four oblique-angled turns, which cannot be called "only slightly sinuous?"

Now, if slight sinuosity adds 33 per cent. to the resistance encountered by air, in *B* mine we have five right-angled turns, against two in *A*; four acute angles against none in *A*; four oblique angles against none in *A*; therefore, the resistance to the passage of 6,000 cubic feet of air in *B* is enormously greater than it is in *A*; whereas, by the methods of figuring adopted in the paper mentioned, the rubbing surfaces and the areas being equal, the resistance should also be equal in each mine for same velocity.

The form of *B* is not much exaggerated for the purposes of argument; in fact, falls short in its crookedness, of many mines I have known.

The block coal of Mahoning and Trumbull counties, Ohio, afforded little scope for skill in laying out a mine; its main roads must per force follow the swamps. No such excuse, however, could be made for the tortuous character of many mines in Western Pennsylvania 35 years ago.

Recently I have come across some notes by a clever recent writer which, in the main, support the position I have taken. One paragraph reads as follows:

"The hydraulic engineer can hit on one or two values of *k* that practically suffice for general use; but it is the miner who finds *k* a different value in every part of the mine. Not only the amount, but the nature of friction, varies continually. At one part it is sliding friction, at another concussional. Sometimes dry and sometimes wet friction; sometimes the friction is against trams and trains traveling in the same direction as the current. Sometimes against trains moving the opposite way. Such medley of resistances does not lend itself readily to practical, much less accurate, measurement."

The following table shows the values of the coefficient of friction, represented by the letter *k*, being the height in feet of air column of the same density as the flowing air, required to overcome the frictional resistance encountered by 1,000 cubic feet of air per minute in passing through a passage having 1 foot of sectional area, and presenting 1 square foot of rubbing surface to the air in motion.

Atkinson's Table

Nature of Material Composing the Pipe or Airway	State of the Internal or Rubbing Surface Exposed to the Wind	Observers' Names	General Temperature of the Air or Gas	Head of Column of the Same Density as the Moving Air or Gas Required to Overcome the Friction Being the Coefficient of Friction, *k*
Burnt earth	Clean	Peclet	Hot	.26881
Galleries of a coal mine.....	Ordinary state	C. C. Greenwell	Cool	.25436
Sheet iron..	New and clean	Peclet	Hot	from .10583 to .06773
Cast iron...	Ordinary	Mons. Rudler	Hot	.08466
Cast iron...	Sooty	Peclet	Hot	.05292
Cast iron...	Old, tarred	Girard	Cool	.04844
Gas in pipes, cast iron..	Ordinary	Mr. Hawkesley	Cool	.03014
Water in pipes, cast iron	Ordinary	Eytelwein	Cool	.03028
Sheet iron..	Old and rusty	Girard	Cool	.02752
Tinned iron		D'Auhuisson	Cool	.02540

Remarks.—In applying these values of *k* to the formulas since they are calculated for velocities of which the unit is 1,000 feet per minute, the real velocities in feet per minute must be divided by 1,000, to give the value of *v* in the formulas; and *v* in the formulas must be multiplied by 1,000 to give the velocity in feet per minute.

Circumstances of Passage or Passages	Value of *k* in Inches of Water	Observer
1. Levels of uniform section without obstruction..	.00038	Devillez
2. Arched passage, straight, normal area.........	.00033	Murgue
3. Arched passage, slightly sinuous..............	.00051	Murgue
4. Arched passage, continuous curvature..........	.00062	Murgue
5. Arched passage, straight, smaller area.........	.00055	Murgue
6. Brick-lined shafts...........................	.00040	Raux
7. Unlined levels, normal area...................	.00094	Murgue
8. Timbered levels, normal area..................	.00156	Murgue
9. Intake airway of English mine................	.00079	Elwen
10. Return airway of English mine...............	.00106	Elwen
11. Road at working face........................	.00263	Elwen
12. Working places at face......................	.00269	Devillez
13. Working places at face......................	.00273	Raux
14. Timbered passage, straight, small area.......	.00238	Murgue
15. Unlined passage, straight, small area........	.00122	Murgue
16. Average for the whole of a mine..............	.00178	Devillez
17. Average coefficient obtained from experiments made at three collieries....................	.00180	Raux

The United States Geological Survey has recently made an examination of the Dan River district, in Stokes and Rockingham counties, North Carolina, to determine whether or not it contains any important coal beds. All the known prospect pits were examined and a number of them reopened.

The coal-bearing rocks consist of a narrow belt of black slaty shale which extends from a point just north of the Virginia line southwestward through Leaksville, Madison, and Walnut Cove to Germanton, N. C. Some people in the district believe that these black shales, which they regard as a "sign of coal," are thicker where they lie under cover and that if they are followed far enough down they will be found to be coal. This idea has no foundation in fact but has been the incentive to much useless labor and unwarrantable expense. At a few places on Dan River in the black shale belt thin beds of a hard semi-anthracite have been found. This coal disintegrates very slowly and consequently should be as thick at the surface as it is underground. It varies in thickness, however, from place to place and is as likely to be thinner at considerable depth as it is to be thicker. Scarcely more than a foot of good coal has been found in any one bed in the district of 30 miles along the outcrop, a fact which further diminishes the probability of finding thicker beds below the surface.

PRACTICAL FUEL VALUES

*By W. P. Young**

Qualities that Add to or Subtract From the Value of Coal for Special Uses

Every one connected with a coal company is familiar with the fact that we receive complaints from the consumer. Generally speaking, are these complaints justifiable? Is the producer altogether at fault? How many troubles arise through selecting the wrong coals, or handling them wrongly when received?

As a sidelight on these and similar questions, it may be of interest to consider some of the effects accompanying relatively small and seemingly unimportant differences and changes in physical, mechanical, and chemical properties of various fuels, with reference to their practical fuel value, more especially in metallurgical operations.

For many years there has been a persistent belief that coal should be very lumpy in order to be good. There are some grounds for this belief, but the better effects found with lumpy coals are nearly always due to furnace conditions and not to the fuel. The modern idea of selecting a fuel suited to local conditions is gaining ground and it should be observed that this does not always mean the selection of a fuel highest in heat units, as coals of lower theoretical value may give better results.

In a recent engine test on one of the Eastern roads, a fuel running under 13,000 British thermal units replaced one of over 14,000, it being found necessary to use 25 per cent. more of the latter to make the same engine mileage. This was due to the lighter gravity and finer condition of the higher valued coal, a considerable quantity of which was lost by reason of the powerful draft.

In one of the large steel plants certain furnaces were heated by coal, the bed of fuel being about 2 feet in depth. It was found to be more economical to use a coal running 5 per cent. higher in ash than another, on account of the clinkering of the latter and consequent loss in the frequent cleaning of fires.

If the coal used be of a coking nature, the presence of fine coal does not usually cause any diminution of available fuel value.

When the blast-furnace man demands a coke that is solid and in comparatively large pieces, he is not actuated by any whim, but really needs such a fuel. To him the principal use of coke is to reduce iron from its ores, but as a matter of fact, the actual quantity so used is small compared with that used in heating the slags resulting from the earthy matter of the ores, limestone, and coke ashes, together with the losses of fuel in burning to carbon dioxide in the upper zone of the furnace, without doing practical work. By far the larger amount of ores now in use are very fine and if the coke be in large pieces, it tends to prevent the packing together of the ores and the obstruction of the blast.

Large coke presents much less surface per ton, to the action of the blast, than does that in small pieces, with a corresponding minimum loss, by burning to carbon dioxide, before doing reducing work. The film of graphite, found on some well-burned and well-drawn cokes, prevents the penetration of the blast and thus saves fuel. On the other hand, if the coke be spongy, the loss from the above cause will be high, if indeed a good deal of fuel is not crushed by the weight of the stock and blown through the gas passages.

In the carriage and storage of soft coals, the presence or absence of slack is an important factor. When moderately lumpy coals are handled roughly, as in dumping cars by turning them over, a small percentage of slack mixed through the lump, prevents breakage to a degree that is surprising, though if the coal is to be stored, for any length of time later, this good effect may be more than offset by the increased liability to spontaneous combustion, by reason of the slack preventing the access of air to the interior of the pile, as well as the fact that the slack gives up its hydrocarbons more readily and in greater volume at ordinary temperatures than does the lumpier coal. In fact it would appear that there is a decided tendency in some slack coals, for the hydrocarbons to assume the gaseous form whenever the coal is finely enough divided.

This seems to be more particularly the case with those coals whose breakage produces a very fine greasy kind of slack, from which water runs without leaving a trail of wet coal. This greasy nature is probably due to both chemical and physical conditions and is a deciding factor in spontaneous combustion on storing. Slack of this nature results from the breakage of a coal that is tough rather than hard, the harder coals usually breaking down into a coarser slack and one having more definite cleavage planes.

It would appear that the composition of the coals, in the seam is gradually changing, this change being more marked where the coals are exposed in the outcrop, or penetrated by streams of water. The hydrocarbons composing the volatile matter seem to undergo more or less decomposition with resultant changes on the rest of the coal and the production of various gases, which are either occluded, or mechanically held in the crevices of the seam. While the real nature of the volatile matter, as it exists in the solid coal, is not well understood, if indeed at all, yet it is known that various compounds were formed according to the heat and pressure, when the coals came to their present condition and generally the volatile matters of superimposed seams are similar in total percentage and character if the seams have the same physical properties; for instance, the same hardness, cleavage, and appearance of slack.

There is a relationship between the physical properties of the coal and the state of combination of its volatile contents and may I suggest that possibly the very high explosive power of some coal dusts may be due to their being loosely held together in some higher organic compounds.

The coking process depends more on the nature of the volatile matter than upon its quantity, provided there be enough present to keep the ovens in condition to fire the succeeding charge of coal.

When the charge in the beehive oven catches fire quickly, the coke will be solid in structure, while if the coal remains in the oven for an hour or more after the charge is leveled before catching up, and if it then comes to a semiplastic state slowly, the coke will be spongy and carry more of the sulphur from the coal.

A simple wetting of the slack before charging, will materially reduce the sulphur contents of the coke.

Coke men are apt to consider complaints of slightly higher sulphur and phosphorus as being whims, claiming that the furnace man uses ores running many times as high in both these elements without much trouble, but the cases are entirely different, as the sulphur and phosphorus in ores, mill cinder, etc., are liberated from them before they pass below the tuyeres of the furnace and consequently have a chance to get away, either in the gaseous state, or in the form of slags combined with lime, while the same elements in the coke pass with a certain amount of the solid coke down into the crucible of the furnace and are there greedily absorbed by the iron and it is then too late to remove them by means of lime.

The clinkering of coals in furnaces is a source of great loss in efficiency. Clinkering is caused by the presence of iron, or lime in the ash, or by the melting of the coal before combustion.

The results of clinkering are direct losses of solid fuel dragged out in cleaning fires and the heating to the melting points of slag of various compositions, requiring the use of fuel otherwise doing efficient work. Red coal, which commonly carries a notable percentage of iron oxides, combined with water, is an illustration of these sources of loss. The iron oxide is

* Read before the West Virginia Coal Mining Institute, Huntington, W. Va., December 7, 1909

reduced to metallic iron, with attendant formation of silicate of iron, and forms a particularly mean clinker with its attendant losses of heat and solid fuel. As most of the red coal is also soft and will melt under heat without complete combustion, we have a combination causing much complaint. It is surprising, however, to note the relatively great loss of heat in a coal that is so slightly different from the original solid, black fuel, as to be almost unnoticeable. This loss of heat is more frequently found where boilers are being overloaded and as a rule the consumer has no real idea of his trouble, simply complaining that the coal "wont make steam" or that it has no "life." Red coal, however, has one good point, in that it usually carries very little sulphur, and consequently can be used where an excess of this element is prohibited.

On account of the fact that the red coal has undergone changes in its volatile contents, it does not make good coke in the beehive oven.

The amount of foreign matter, as slate, bone, and sulphur streaks is usually overestimated, the total percentage in any half decently prepared coal being very small. It is rarely that total visible foreign matter in soft coals reaches two-tenths of 1 per cent. of the whole, yet one-half of this amount will bring forth strenuous complaint and the assertion that the coal "was half bone and slate."

If the foreign matter is a light bone, it will do no harm in ordinary practice and will not reduce the available heat units at all.

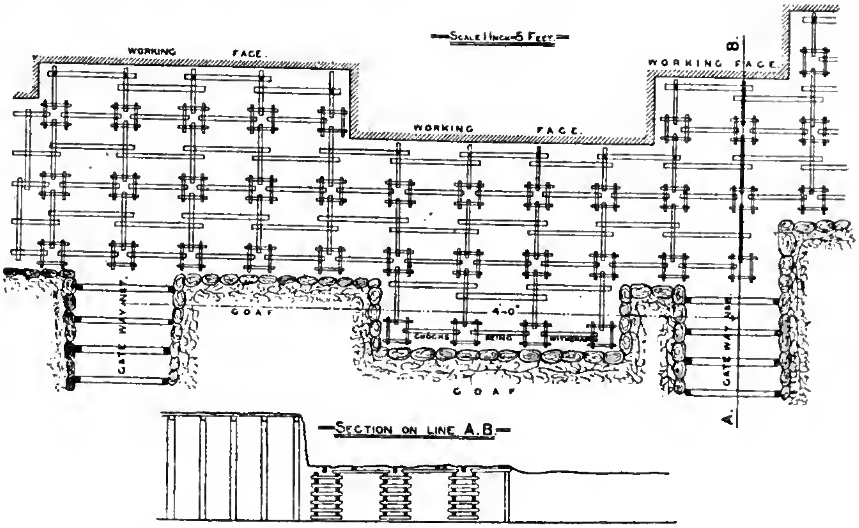
Slate is inert and only causes real losses by interfering with the draft when lying on the grates. The total quantity is too small to materially lower the available heat units.

While slates and sulphurs should be removed in all cases, it is doubtful if we should yield altogether to the whims of the consumer in the matter of the removal of the lighter bone coals, as they have a definite fuel value and should therefore be used.

Some day those following us will reproach us for the wanton waste of many thousands of tons of good fuel in days gone by in order to satisfy the unreasoning demands of consumers.

TIMBERING IN INDIAN COAL MINES

In the coal mines of England and of the continent where a treacherous roof is the rule, systematic timbering has been strongly advocated to reduce the death rate from falls of roof and sides, and in this country special rules regarding systematic setting of timber in all coal mines came into force recently. Several systems have been advanced as models, and Mr. W. H. Pickering, the Chief Inspector of Mines in India, reports that in the Punjab collieries where the conditions are somewhat similar to those of England, a system of timbering which will compare favorably with any in the world has been adopted. We reproduce a plan showing the timbering of an actual working place in Dandot coal mine. At this mine the timber is set and drawn entirely by Indians, and the system is so well carried out that during an inspection (extending over several days) no fault could be found with it. The seam is from 2 feet to 3 feet thick, and is worked by longwall method. Wooden "chocks," 2 feet square, are built at a distance of 4 feet 6 inches apart, measuring from center to center. Props are set in advance the same distances apart, and these form centers for the chocks as the work progresses. Planks $1\frac{1}{2}$ inch thick and 6 inches wide are set above the chocks and props and kept close to the face. Thus the miners are always protected by timber when working. The timber is set by specially appointed men.—*Iron and Coal Trades Review.*



PICTOU COAL FIELD LORE

C. Ochiltree MacDonald in his book "The Coal and Iron Industries of Nova Scotia," says:

"It has been suggested that the name 'Pictou' originated from the combustion of coal discovered under 4 feet of burnt clay and ashes on the East River. The Indians had a traditionary account of a fire at that point; and as 'Bucto' signifies fire in the Micmac tongue it has been conjectured that 'Pictou' is a corruption of that word, in that connection. An authority on the Micmac language has further stated that 'Pict' means an explosion of gas; and he suggests that the name 'Pictou' was given to this important district owing to the escape of natural gas from the coal underlying the East River—a phenomenon noticed and utilized by the mining classes as late as 1827. Patterson, in his history of Pictou has, however, pointed out that the phenomena of the charred coal and gas appeared only at the East River, whereas it seems certain that the name 'Pictou' was specially applied to the north side of the harbor on which no such phenomena existed. It is probable that the district has derived its name from 'Bucto,' a word applied by the Indians to a portion of the West River where one of their large encampments was burned during their absence on a cruise down the harbor. It is interesting to note that these theories derive the name of this coal field from fire; and the application of such a name to a district which has been the scene of the worst colliery disasters in Nova Scotia may be regarded as a curious coincidence.

"The productive measures of the Pictou coal field, an almost continuous section of which is exposed on McLellan's Brook, are affected by faults. The known worked seams are gaseous, and some of the coal has discharged more explosive gas than any other Nova Scotia mines. From the freshly cut coal of a Pictou colliery ventilated by 10,000 cubic feet of air a minute, gas has issued pure enough to extinguish the flame of a safety lamp held in it; and a pair of levels in solid coal has yielded sufficient gas to foul a current of 19,000 cubic feet of air per minute. Workings of the old collieries have been found infested with black damp; cars, harness, canvas cloth, tools, and even hay, lost for 20 years, have been recovered from the Deep Seam winnings well preserved by this noxious element, and when the 'Burnt Mines' were reentered in 1849 the explorers waded through black damp, with inflammable gas overhead."

POWER PRODUCTION AT COLLIERIES

*By Howard N. Eavenson**

If the mine owner or superintendent finds conditions favorable to natural drainage he is to be congratulated, and if he makes conditions favorable under adverse circumstances and without excessive cost he is to be complimented. Under other conditions pumping must be resorted to and the problem in some mines is serious enough to demand a careful investigation as to the most economical equipment. If highly corrosive water in large or small quantities is to be handled, it does not pay to install ordinary cast-iron pumps. In the larger sizes, cast-iron, wood-lined reciprocating pumps having acid-resisting metal or wood plungers have given general satisfaction and, as a rule, the inside-packed or piston pumps are not to be recommended. If the mine has air power only, the pumping is usually done by standard steam pumps, air driven, and as these pumps do not utilize the expansive force of the air when working under full load we must expect very low economy from this combination. Where electric power is available the motor-driven, single-acting triplex, or double-acting outside-packed pumps are generally selected, with geared connections to the motor, Fig. 1.

Choosing the Most Economical of Different Kind of Machines Consuming Power

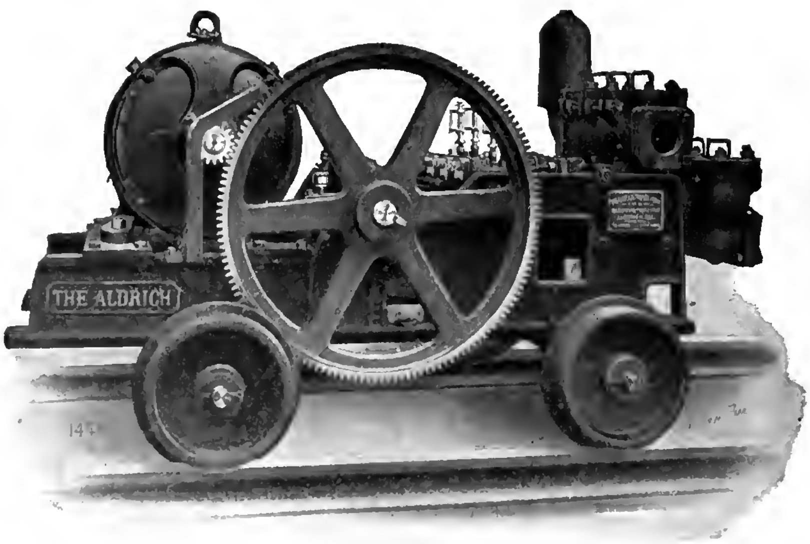


Fig. 1. Motor-Driven, Single-Acting Triplex Pump

During the last 5 or 6 years the development of the centrifugal and turbine pumps, Fig. 2, has made remarkable strides and it is now possible to get pumps of this type at from one-half to one-fourth the cost of reciprocating pumps of the same capacity and of equal efficiency and reliability. In the matter of durability the reciprocating pumps have the advantage, but considering first cost, including installation and maintenance charges, the advantages are in most cases in favor of the centrifugal pump. A case may be cited where a triplex, electric, wood-lined pump, having a normal capacity of 1,000 gallons per minute against 120-foot head, occupying a space of 1,000 cubic feet and weighing 90,000 pounds, was replaced by an acid-resisting bronze centrifugal pump, having a capacity of 1,200 gallons per minute against the same head, and requiring about the same space as an ordinary sewing machine The cost of the triplex pump was about three and one-half times that of the centrifugal, and the installation costs were in the ratio of 1 to 10 in favor of the latter. The depreciation rate and maintenance costs are considerably in favor of the reciprocating type of pump, but not by any means in the ratio of 3½ to 1.

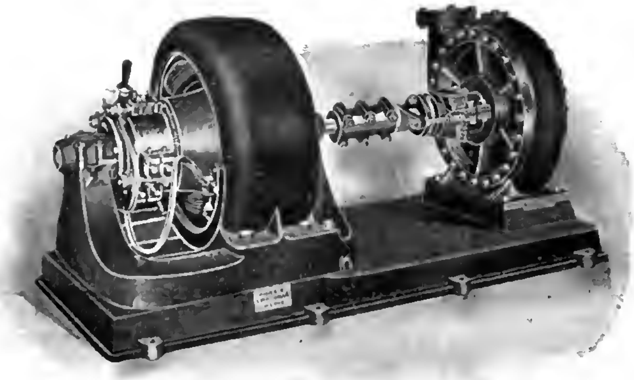


Fig. 2. Motor-Driven, High-Lift Pump for 100 Feet Elevation

*Abstract from paper read at the June meeting, West Virginia Coal Mining Institute.

It is commonly supposed that centrifugal pumps designed for a certain head and capacity will not operate satisfactorily under any other conditions. This supposition is a little at fault under the light of recent developments, as it has been found possible to so design these pumps that they will deliver under other heads than that of their normal rating, at reduced efficiencies and smaller power requirements. A case was recently observed where such a pump designed for 117 feet head and 1,200 gallons per minute was successfully used in a temporary service where there was but 19 feet head, delivering upwards of 2,000 gallons per minute with about 70 per cent. of the power demanded under normal conditions.

In ordering centrifugal pumps it is well to specify that the cases shall be split horizontally in order that the rotating parts may be inspected or removed without disturbing the pipe connections and that detachable wearing rings shall be fitted to both the pump case and the impeller.

In the matter of efficiencies, there is, on the general average, probably not a great deal of difference between the two types, the friction in the gearing and plungers, and the slippage on the part of the reciprocating pump approximately balancing the losses in the case and impeller of the direct-connected, single-stage, centrifugal pump. A common impression is that with heads of 150 feet and over, requiring two or more stages, the centrifugal pump is not adapted to mine work. This may be true under some conditions, but there are instances where these pumps are in every-day use in mines working successfully against heads approximating 1,000 feet.

Under the head of equipment for mine ventilation our interest lies chiefly in the consideration of the various types of fans; and we will all perhaps agree that there is no class of machinery used in and around mines, the development of which is in such a chaotic condition as that of the up-to-date mine fan. We find at some mines the huge, slow moving, costly, and reliable 30-foot wheels, Fig. 3, and at others the tiny, many bladed, high-speed Sirocco type, Fig. 4, having alleged double-hurricane velocities of 2 miles and more per minute through their narrow throats, and in still other cases the whirligig impeller type, Fig. 5, usually installed because it is cheap and handy.

A fan is a comparatively simple contrivance and it is not easy to understand why it has required 50 years to evolute from a Guibal to the Sirocco. In our search for light upon the subject we do not get much assistance from the fan builders as a class, for theirs seems to be an art with-

out recognized standards, though dependent upon some simple laws that have always existed.

Large fans are almost always driven by direct-connected steam engines. As a rule these engines are single expansion, with variable hand-adjusted cut-off and gravity oiling systems. As they are often run 24 hours each day under constant load, there would seem to be an excellent opportunity to cut down the fuel consumption 50 per cent. by installing compound condensing engines. In some locations this would not be feasible, on account of the lack of a sufficient quantity of water for condensing, but in other locations near small streams cooling ponds could be resorted to. Economical engines are about as reliable and durable as those of the simple type and it is reasonable to maintain that the money saved in fuel at the fan house is just as good as that saved by the high-class machinery in the power house.

Fans requiring a maximum horsepower output of 150 and under, when located within reach of a central power system and at some distance from a boiler house, are best driven by alternating-current motors, preferably of the three-phase induction type, the connections being direct in the case of high-speed fans and with belts or high-speed chains for lower speeds.

In regard to tipple equipment not much can be considered in detail in this paper as each tipple usually presents a distinct engineering problem in itself, depending upon location and method of handling and preparing the coal.

The prevailing styles in mine cars are so varied and opinion regarding their construction so divided that any attempt to pass upon their respective merits and faults would produce mostly dissenting opinions.

Taking up again mine equipment in general, there is one point in particular which should be given full consideration, that of standardization in a single mine or in a group of mines from which requisitions are sent to the same supply house.

Referring to requisitions, many of you have noted the slipshod manner in which they are often prepared, and some of us have had experience in trying to determine from them what is wanted, the good guessers generally getting there first. If there was summed up the time unnecessarily lost on account of requisitions of this class, including that of the purchasing agent, engineers, and clerks, together with the delays due to correspondence with the manufacturers on account of insufficient data, the amount in dollars and cents would be astonishing. Cases are numerous where a high-priced man from the central office is required to make trips requiring half a day or more in order to get a few measurements of the most simple nature. There should be at least one man at each mine who has had sufficient instruction to enable him to prepare legible rough sketches of parts of machines for which new parts are wanted, and it is of course essential that such sketches should be complete and the dimensions marked thereon as accurate as it is possible to make them. In ordering new parts for machines there should in all cases be obtained the name of the maker, the shop number, if any, and the pattern number if the part wanted is a casting.

Where a number of mines are operated under one management it is essential to have at the central office a mechanical department in which there are men who have had special training and considerable experience in mechanical and electrical engineering in their application to coal-mining machinery. In the files of this department there should, among other things, be kept individual records of each machine, either in the card index or loose-leaf form, so that records of new machines or transfers of old ones may be easily inserted at their proper places. It is an excellent plan to attach a brass plate to each machine upon which there has been stamped with steel dies the company's serial number and the date of purchase, these data being recorded upon the corresponding sheet or card in the machinery record.

In making up an inventory of mine equipment, the itemized present value of the different machines is often fixed by taking the cost price and deducting therefrom a sum figured from the number of years which it has been in use at an arbitrary annual rate of depreciation. This plan leads us into very serious errors, even when there are different rates of depreciation fixed for different classes of equipment, for the value of any piece of equipment depends largely upon the care which it has received, the kind of service to which it has been subjected, and the state of repair in which it has been kept. For example, a mining machine depreciates in value very rapidly during the first 2 or 3 years of its life, after which it remains in about the same average condition due to the repairs made to the various parts in order to keep it in service. The best method of determining present values consists in an appraisement made at some fixed date in each year by some qualified person, a method involving considerable work, but justified by results.

In many instances where there are groups of mines under a common management, systems of cost statements in the form of daily and monthly reports have been devised and successfully carried out, these statements showing, in connection with each machine, the nature and cost of repair parts, the amount and cost of repair labor, the amount of oil and waste used, the approximate number of days idle during the month and other information from which the manager can get a fairly correct idea as to the individual and comparative costs of maintenance and condition of each piece of equipment.

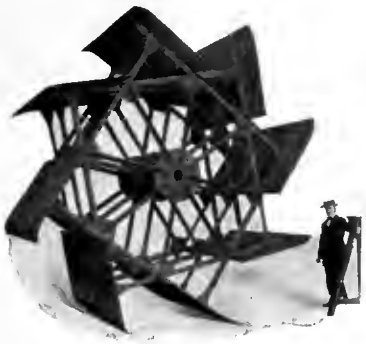


Fig. 3. Modified Guibal-Type Fan Wheel

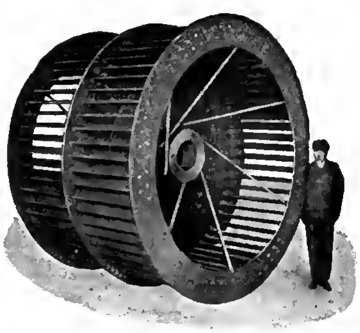


Fig. 4. Sirocco Fan

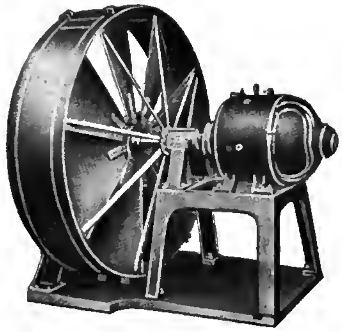


Fig. 5. Stine Disk Fan

The capacity of a revolving screen will depend very largely upon how the ore breaks. Ores which break cubical, or nearly so, in shape, go through a screen very fast, but ores which break in specules or needle-shaped particles do not screen nearly as fast as cubical breaking ore. The following average may meet the requirements of the engineer: Screen making 20 revolutions per minute, 30-mesh screen, capacity 1,000 pounds per 24 hours per square foot of screen surface; 20-mesh, capacity 1,250 pounds per square foot; 16-mesh, 1,700 pounds; 12-mesh, 2,200 pounds; 8-mesh, 3,000 pounds.

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

Note.—The following questions, selected from different examinations, are here numbered consecutively. The examination from which the question is taken and the number of the question are found at the end of each.

Selected Questions of the Illinois Examinations for 1909, Held at Springfield

Ques. 1.—What load will break a a white-oak timber 10 inches (wide) by 12 inches (deep) and 12 feet long, if the load is equally distributed along the length of the beam?—Mine Manager's Examination, Q. 3; 4-12-'09.

Ans.—The moment of inertia (I) for a rectangular beam; breadth, $b = 10$ inches; depth, $d = 12$ inches, is

$$I = \frac{b\,d^3}{12} = \frac{10 \times 12^3}{12} = 1,440$$

Assuming this beam is simply supported (not fixed) at each end, and taking the breaking fiber stress of white oak as $f = 10,000$ pounds, the load that will break the beam, in this case, is

$$L = \frac{4}{3}\left(\frac{f}{b\,d}\right) I = \frac{4}{3}\left(\frac{10,000}{12 \times 12}\right) 1,440 = 133,333 \text{ lb., or } 66\tfrac{2}{3} \text{ tons}$$

Ques. 2.—If a cross-bar 12 inches in diameter and 8 feet between notches carries a certain load, what should be the diameter of a cross-bar 14 feet between notches to bear the same load (with the same degree of safety)?—Mine Manager's Examination, Q. 4; 4-12-'09.

Ans.—For the same total load supported, the diameter of the cross-bar varies as the cube root of the length of clear span; or, the diameter ratio of two cross-bars is equal to the cube root of the length ratio. Calling the required diameter of the 14-foot bar x

$$\frac{x}{12} = \sqrt[3]{\frac{14}{8}}; \text{ and } x = 12\sqrt[3]{1.75} = 14.46 \text{ in.}$$

Ques. 3.—What power will be required to hoist an output of 800 tons of coal (per day) on an incline $\frac{5}{8}$ of a mile long and rising 2 inches to the yard, extending from the inside parting to the shaft bottom, and return the empty cars, which weigh one-half as much as the load they carry; assuming the resistance of the cars to be $\frac{1}{60}$, and the power required to move the rope equal to that required to move the cars. The speed of the rope is 3 miles per hour, and a working day 8 hours; the cars are moving $\frac{4}{5}$ of the time.—Mine Manager's Examination, Q. 5; 4-12-'09.

Ans.—This will be assumed a double-track passage in which the weight of the descending empties partly balances that of the ascending loaded cars and rope. In this case, the load on the engine consists of the gravity pull of the unbalanced load (coal and rope) when loaded trip is at the bottom of incline, plus the friction pull of the loaded and empty cars and the rope. The length of the incline is $\frac{5}{8}(5,280) = 3,300$ feet; the speed of winding is $3 \times 5,280 \div 60 = 264$ feet per minute. The time of hoisting one trip is then $3,300 \div 264 = 12.5$ minutes, which is $\frac{4}{5}$ of the total time per trip, including changing trips and delays; full time per trip is $12.5 \div \frac{4}{5} = 15.625$ minutes. The output is 800 tons in 8 hours, or 100 tons in 60 minutes, or $1\frac{2}{3}$ tons each minute. This makes the weight of coal to be hoisted each trip $1\frac{2}{3} \times 15.625 \times 2,000 = 52,083$, say 52,000 pounds, or 26 tons. Since the cars weigh one-half as much as the coal carried, the total moving load, including the empty trip, is $26 \times 2 = 52$ tons, not counting the rope, for which allowance is to be made later. The total car resistance, including both trips, is therefore $\frac{1}{60}(52 \times 2,000) = 1,733+$ pounds. The inclination of the plane being 2 inches per yard (1 in 18), the gravity pull on the rope due to the coal hoisted per trip is $\frac{1}{18}(26 \times 2,000) = 2,889$ pounds. This makes the total load on the engine, exclusive of the rope, $1,733 + 2,889 = 4,622$ pounds. At a speed of 264 feet per minute, the exerted pull would be $\frac{4,622 \times 264}{33,000} = 36.976$, say 37 horsepower. Since the question states that as much power is required to move the rope as to move the cars, this power should be doubled, giving $2 \times 37 = 74$ net horsepower. Assuming this engine to have an efficiency of say 80 per cent., the indicated horsepower of the engine should be $74 \div .80 = 92.5$ horsepower.

Ques. 4.—Give the pressure per square inch at the bottom of a column pipe 452 feet long; the pipe is pitching 42 degrees, and is full of water.—Mine Manager's Examination, Q. 6; 4-12-'09.

Ans.—The vertical rise of this pipe is $452 \times \sin 42° = 452 \times .66913 = 302.44+$ feet. The static pressure due to this head of water, or the pressure at the bottom of the pipe when the same is standing full of water, is $302.44 \times .434 = 131.25+$ lb. per square inch.

Ques. 5.—A sump in a mine is 62 feet long, 8 feet wide, and 9 feet deep, and full of water. How long will it take a 5-inch pump to empty this sump, at a velocity of 100 feet per minute, if the leakage of the valves and resistance of the pump cause a loss of 20 per cent.? There are five 2-inch pipes running full and discharging water into the sump at a common velocity of 100 feet per minute.—Mine Manager's Examination, Q. 7; 4-12-'09.

Ans.—The cubical capacity of the sump is $62 \times 8 \times 9 = 4,464$ cubic feet. The 20-per-cent. loss evidently refers to the leakage of valves and "slip" of water past the piston, or leakage of plunger packing, and relates wholly to the water end of the pump; although the mention of the "resistance" of the pump would seem to indicate that the 20-per-cent. loss was a loss in power and as such related both to the water and the steam end of the pump. But, in that case, the question would be incapable of answer because what is required is to ascertain the actual discharge of water without reference to the power necessary to operate the pump. Therefore, the mention of *resistance* should have been omitted in the question.

The piston displacement of a 5-inch pump at a speed of 100 feet per minute is $12 \times 100\,(.7854 \times 5^2) = 23,562$ cubic inches per minute, which makes the discharge, assuming a loss of 20 per cent. in the water end, $(1 - .20)23,562 =$ say, 18,850 cubic inches of water each minute. The inflow into the sump from five 2-inch pipes flowing full, at a velocity of 100 feet per minute, is $12 \times 100 \times 5(.7854 \times 2^2) =$ say 18,850 cubic inches of water each minute. Since this inflow is exactly equal to the discharge of the pump under the assumed conditions, the latter could only handle the inflow from the five pipes, and the sump would remain full of water till other means were adopted.

Ques. 6.—Give the size of pipe required to transmit 800 cubic feet of (free) air per minute a distance of 2,500 feet, the initial pressure being 100 pounds per square inch, and the plant being located 4,000 feet above sea level where the reading of the barometer is, say 26 inches under normal conditions. —Mine Manager's Examination, Q. 8; 4-12-'09.

Ans.—The question should state the loss of pressure allowed in the transmission, or the terminal pressure required, as it is possible to use any reasonable diameter of pipe, depending on the pressure required to be delivered. For example, assuming a pressure of 90 pounds is required to be delivered at the end of the pipe, makes the allowable drop of pressure in transmission $100 - 90 = 10$ pounds. The weight of 1 cubic foot of dry air at a barometric pressure of 26 inches (temperature, say 60° F.) is

$$w = \frac{1.3273 \times 26}{460 + 60} = .06636 \text{ lb.}$$

The atmospheric pressure corresponding to the given barometer is $26 \times .4911 =$ say 12.75 pounds per square inch. To produce a gauge pressure of 100 pounds at the given elevation would require $(100 + 12.75) \div 12.75 = 8.84$ compressions of the free air. Using a constant for a comparatively small pipe, say $c = 50$, and

substituting the given values, the required diameter of the pipe, for this transmission, is

$$d=\sqrt[5]{\frac{.06636\times 2{,}500\times 800^2}{8.84\times 50^2\times 10}}=3.43,\text{ say }3\tfrac{1}{2}\text{ in.}$$

QUES. 7.—(*a*) What is a safe voltage to use in a mine for electric haulage? (*b*) If the generators are producing a 500-volt current what is necessary to be done to retain the same power but reduce the voltage to 250 volts?—Mine Manager's Examination, Q. 9; 4-12-'09.

ANS.—(*a*) A current of 250 volts is generally considered safer for mine work than one of higher voltage. (*b*) In order to reduce the voltage of an electric current while maintaining the power constant, it is necessary to use a transformer to step-down the current to the desired voltage. In doing this the amperage or strength of the current is increased in the same ratio in which the voltage is decreased, and the power remains constant.

QUES. 8.—If one ventilating fan is producing a current of 25,000 cubic feet of air per minute, in a mine, what total quantity of air can be produced by the addition of another fan of the same dimensions and operated with equal power.—Mine Manager's Examination, Q. 11; 4-12-'09.

ANS.—If one fan is exhausting and the other blowing, the quantity of air will not be increased. It is impossible to run two ventilating fans tandem, since one or the other will do the work and the other will simply churn the air with no practical effect. If, however, the two fans are arranged both blowing or both exhausting, they will do equal work. In this case, the power on the air is doubled and, for the same conditions in the mine the quantity of air circulated will be increased in the ratio of $\sqrt[3]{2}=1.26$; or, the increase will be 26 per cent. of the current that one of these fans will circulate when working alone.

QUES. 9.—What horsepower will be required to deliver 1,350 gallons of water per minute against a pressure of 80 pounds per square inch, the suction lift being 20 feet, and allowing 20 per cent. for friction?—Mine Manager's Examination, Q. 14; 4-12-'09.

ANS.—A pressure of 80 pounds per square inch is equivalent to $80\div.434=184+$ feet delivery head to which must be added the suction (20 feet), making the total head against which the pump must operate $184+20=204$ feet. Allowing 20 per cent. for friction, the horsepower required for this work would be

$$H=\left(\frac{1{,}350\times 231}{1{,}728}\right)\frac{62.5\times 204}{.80\times 33{,}000}=87+\text{ H. P.}$$

QUES. 10.—Give the breaking strain of a $\frac{11}{16}$-inch, cast-steel, wire rope, of seven strands and 19 wires to the strand; also, the safe working load.—Mine Manager's Examination, Q. 15; 4-12-'09.

ANS.—American wire-rope manufacturers use five, six, and eight strands in ropes; we know of no American wire ropes of seven strands. This is probably an oversight in writing the question, or a typographical error. The breaking strain of a six-strand, 19-wire, ordinary cast-steel rope, 1 inch in diameter is 34 tons; and the breaking strain of a similar rope $\frac{11}{16}$ inch in diameter is $34(\frac{11}{16})^2=16.07$, say 16 tons. It is common mining practice, except in deep shafts, to use a factor of safety of 5, which makes the safe working load for this rope $16\div 5=3.2$ tons.

QUES. 11.—The total quantity of air passing over a certain mine furnace is 200,000 cubic feet per minute, when the furnace is burning 5 tons of coal in 12 hours; what increase of air will be gotten by doubling the coal consumption, or burning 5 tons more of coal each 12 hours?—Mine Manager's Examination, Q. 16; 4-12-'09.

ANS.—The power developed by the furnace is proportional (approximately) to the weight of coal burned in a given time; and, for the same conditions in the mine the quantity of air in circulation varies as the cube root of the power or the cube root of the weight of coal burned per hour. In this case, the ratio of coal consumption (tons per hour burned) is 2, and the power ratio is also 2, which makes the quantity (air) ratio $\sqrt[3]{2}=1.26$. The increase in circulation due to burning double the weight of coal per hour is then $200{,}000\times .26=52{,}000$ cubic feet of air per minute, making the total circulation 252,000 cubic feet per minute.

This can be shown in another way, as follows: For the same shaft conditions, the height of motive column or unit of ventilating pressure (p) varies as the temperature rise. But the temperature rise in a furnace shaft varies clearly as the coal (C) burned per hour, and inversely as the quantity (Q) of air passing per minute, or as the expression $\frac{C}{Q}$. Therefore, the pressure varies as this expression; or since for the same conditions in the mine p varies as Q^2, we have

$$Q^2 \text{ varies as } \frac{C}{Q}$$

or

$$Q^3 \text{ varies as } C$$

and

$$Q \text{ varies as } \sqrt[3]{C}$$

QUES. 12.—(*a*) How much greater resistance will a current velocity of 600 feet per minute develop than one of 500 feet per minute, in the same airway? (*b*) If the former gives a water gauge of .76 inch, what water gauge will the latter produce?—Mine Manager's Examination, Q. 18; 4-12-'09.

ANS.—(*a*) The resistance of an airway or mine is always proportional to the square of the velocity of the air-current; or, more simply, the resistance ratio is equal to the square of the velocity ratio, in the same airway or mine. Hence in this

$$\frac{R_2}{R_1}=\left(\frac{V_2}{V_1}\right)^2=\left(\frac{500}{600}\right)^2=\left(\frac{5}{6}\right)^2=\frac{25}{36}=.694+,\text{ say }.7,$$

or $R_2=.7\,R_1$; that is to say the resistance due to the 500-foot velocity is .7 of that due to the 600-foot velocity. (*b*) For the same airway, the pressure and the water gauge, like the resistance, vary as the square of the velocity of the current; or the pressure or water-gauge ratio is equal to the square of the velocity ratio; and, here also, w. g.$_2$ = .7 w. g.$_1$ $=.7\times.76=.532$ inch, which is the water gauge corresponding to the 500-foot velocity.

QUES. 13.—A compound, triple-expansion, three-cylinder engine takes steam from boilers at 192 pounds absolute pressure, and exhausts at 3 pounds absolute; what is the total number of expansions in the steam, and what number of expansions take place in each cylinder? Give also the cut-off and the relative area necessary in order to equalize the work of each cylinder.—Mine Manager's Examination, Q. 19; 4-12-'09.

ANS.—The total number of expansions in the steam is found by dividing the initial absolute pressure by the terminal absolute pressure; since the volume varies inversely as the pressure during expansion, and the volume ratio is therefore equal to inverse absolute-pressure ratio. The total expansion in this case is $\frac{192}{3}=64$ expansions. The number of expansions that take place in each of the three cylinders when the work is equally distributed is $\sqrt[3]{64}=4$ expansions. To produce this result each cylinder should cut-off at $\frac{1}{4}$ stroke. Since all the pistons have the same length of stroke, in order to equalize the work in the several cylinders, the areas of the pistons from the high-pressure to the low-pressure cylinder should bear the relation indicated by the numbers 1, 4, 16; or the diameters of these cylinders should be to each other as the square roots of these numbers, or 1, 2, 4. For example, if first or high-pressure cylinder is 6 inches in diameter, the second or intermediate cylinder would be 12 inches, and the third or low-pressure cylinder 24 inches in diameter.

QUES. 14.—The winding drum of a hoisting engine is 4.5 feet in diameter; the shaft is 300 feet deep, or 330 feet from top landing to the shaft bottom where coal is caged. (*a*) How many revolutions will this drum make in hoisting a load of men from bottom to top? (*b*) How many feet per second may the cage make in compliance with the mining laws of Illinois?

(*c*) Give the total time of making one hoist in this shaft. —Hoisting Engineer's Examination, Q. 6; 4-12-'09.

Ans.—(*a*) The circumference of this drum is $3.1416 \times 4.5 = 14.137$; but allowing for the diameter (say 1 inch) of the rope when coiled on the drum, what is called the working diameter is $4.5 + .08 = 4.58$ feet and the working circumference is then $3.1416 \times 4.58 =$ say 14.39 feet. Ignoring any stretch in the rope, the number of revolutions of the drum required to hoist 330 feet is $330 \div 14.39 =$ say 23 revolutions. (*b*) Section 28 *c* of the Illinois mining law limits the speed of hoisting or lowering men to a rate of 600 feet per minute, or 10 feet per second, except with the written consent of the inspector. (*c*) At the speed mentioned above, the time required to hoist 330 feet would be $330 \div 10 = 33$ seconds; but to this must be added a sufficient time to allow for starting and stopping the cage, and for loading and unloading between each hoist, say 27 seconds, which would make the total time 1 minute per hoist.

Ques. 15.—Taking the weight of water as 62.5 pounds per cubic foot, and assuming steam at the atmospheric pressure has 1,640 times the volume of an equal weight of water, calculate (approximately) the weight of steam that would be used per hour by a pair of engines, the cylinders being 30 inches in diameter and the stroke 5 feet, the engine making 30 revolutions per minute and discharging steam at atmospheric pressure.—Hoisting Engineer's Examination, Q. 9; 4-12-'09.

Ans.—The piston speed of this engine is $2 \times 5 \times 30 = 300$ feet per minute. The piston displacement for the two cylinders, at this speed is $2 \times 300(.7854 \times 30^2) \div 144 = 294.25$ cubic feet per minute. Assuming there is no compression, but the exhaust ports remain open to the end of the stroke, and ignoring the back pressure due to the slight resistance of the exhaust ports and valves, the piston displacement, as calculated above, may be taken as the volume of steam exhausted per minute, measured at atmospheric pressure. This steam, according to the question, is assumed to weigh 62.5 pounds for each 1,640 cubic feet. The weight of steam used per hour is therefore

$$\frac{60 \times 2{,}945.25 \times 62.5}{1{,}640} = 6{,}734.6, \text{ say } 6{,}735 \text{ lb. per hr.}$$

Ques. 16.—How would you determine the proportionate areas of the cylinders and the proper point of cut-off in each, in a compound engine of two cylinders, the initial pressure being 65 pounds above the atmosphere, and the final (terminal) pressure 10 pounds below the atmosphere?—Hoisting Engineer's Examination, Q. 10; 4-12-'09.

Ans.—Divide the absolute initial pressure by the absolute terminal pressure to find the total number of expansions in the steam. The square root of the total number of expansions, in two-cylinder compounds, gives the expansions in each of the two cylinders, which is also the ratio of the area of the low-pressure cylinder to that of the high-pressure cylinder, or the *cylinder ratio* as it is called. The square root of the cylinder ratio is the diameter ratio. The reciprocal of the cylinder ratio is the fraction that indicates the point of cut-off in the cylinder. For example, in the case stated in the question, assuming sea-level atmospheric pressure (15 pounds per square inch). The cylinder ratio, or the ratio of the areas of the low- and high-pressure cylinders is $\sqrt{\frac{15+65}{15-10}} = \sqrt{\frac{80}{5}} = \sqrt{16} = 4$. The diameter ratio of these two cylinders is $\sqrt{4} = 2$. That is to say the area of the low-pressure cylinder is four times, and the diameter twice, the same respective factors in the high-pressure cylinder. The point of cut-off in each cylinder is the reciprocal of the expansions in each; or, in this case, ¼.

Ques. 17.—There is required 12,000 pounds of coal every 8 hours to do the hoisting and pumping and to run the fans, at a certain mine when the temperature of the feedwater is 40° F. What weight of coal will be necessary to perform the same work if the feedwater is heated to 95° F.?—Hoisting Engineer's Examination, Q. 13; 4-12-'09.

Ans.—A common rule among stationary engineers is as follows: The saving of fuel by heating the feedwater amounts to practically 1 per cent. of the fuel for each 11° F. the temperature of the feedwater is raised; or in this case, the saving of fuel is $(95 - 40) \div 11 = 5$ per cent., or $(12{,}000 \times .05) \div 8 = 75$ pounds each hour.

Ques. 18.—If a certain boiler is allowed a pressure of 150 pounds per square inch, the thickness of the steel being .045 inch and its tensile strength 60,000 pounds per square inch, what should be the thickness of a 22-inch pipe of the same material to withstand the same pressure.—Hoisting Engineer's Examination, Q. 14; 4-12-'09.

Ans.—It is necessary to first calculate the diameter of the boiler from the data given; thus,

$$d = \frac{2\,S\,t}{p} = \frac{2 \times 60{,}000 \times .045}{150} = 36 \text{ in.}$$

For the same material and the same pressure per square inch; the thickness of the shell of a boiler or pipe is proportional to its diameter; or these ratios are equal; thus, calling the required thickness of the pipe x,

$$\frac{x}{.045} = \frac{22}{36}; \text{ or } x = \frac{11}{18}(.045) = .0275 \text{ in.}$$

(*To be concluded in November*)

MINE RESCUE WORK

The Department of the Interior of the United States Geological Survey, Technologic Branch, has established a mine-rescue training station at Seattle. Many of the coal-mine operators throughout the western country are taking advantage of this by sending to the station one or more men from each mining camp to be educated and trained in the art of rescue work in case of mine accidents. Certificates of competency are issued to those showing the proper proficiency.

There are a number of different kinds of rescue apparatus in use in Europe and America and it has become practically a matter of necessity, and legislation will undoubtedly soon make it a matter of compulsion, for every person or corporation engaged in operating coal mines to provide one or more suits of some approved style of rescue apparatus, the number depending upon the number of employes engaged in the mine and the extent of the operation, and to see that some of its employes are made familiar with the use of the apparatus, so that in case of accident, there may be no delay in the work of rescue.

The Northwestern Improvement Co., the largest operator and producer of coal in the state of Washington, has recently sent several of its employes to take the work at the training station at Seattle, and has also recently secured a number of suits of rescue apparatus, consisting of helmets, oxygen tanks, and all the accompanying paraphernalia, and has organized a rescue corps from among its employes, one corps for each mine. These men meet at stated intervals for practice with the apparatus so as to become familiar with it until they become proficient in putting them on quickly and understand the manner of wearing and working in the suits. This company also has plans made and is preparing for the erection of a small building, especially devoted to storing and preserving the suits. This building will contain a "smoke room" into which the employes who are members of the rescue corps will have to go, after putting on the suits, and they will there be taught how to put up a temporary stopping or perform other emergency work, while wearing the suits. The room will be tight, and filled with burning sulphur or other gas fumes in the meantime, so that a practical test can be made of the reliability of the suits. The attendance of the members of the corps at these practice meetings is made compulsory, however, a small compensation is paid for each attendance. The men are taking a good interest in the work. The apparatus is under the control of one man, who also acts as instructor in explaining its use.

KOPPERS BY-PRODUCT COKE OVENS

*Written for Mines and Minerals, by W. E. Hartman**

The Koppers by-product coke oven was invented by Henrich Koppers, of Essen, Germany. By-product coke ovens were introduced in this country some years ago and the Otto-Hoffman and the Semet-Solvay are fairly well known. In both of these types of ovens it is difficult to distribute the heat uniformly the entire length of the oven. To economize in fuel Messrs. Otto and Hilgenstock devised a method of underfiring by admitting several jets of gas to a combustion chamber from which the flame passed up the flues and out through an escape in the central part of the oven. This somewhat improved heat distribution in the Otto-Hilgenstock ovens induced Doctor Schniewind to combine the underfiring system with the regenerative system.

A Process By Which Ammonium Sulphate Is Made Directly From the Gas

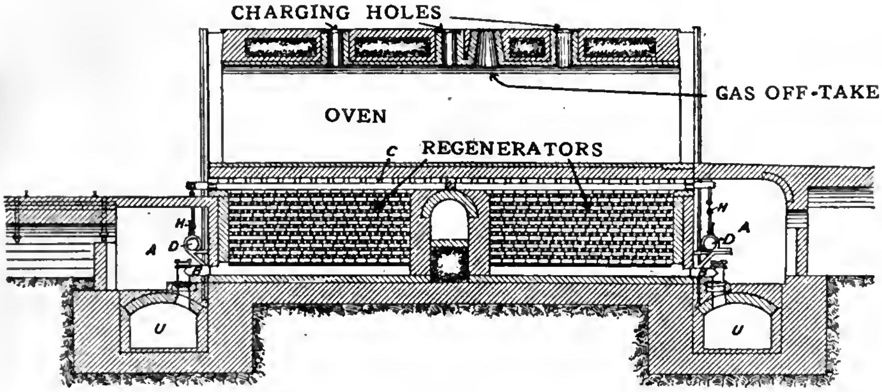


Fig. 1

The original retort ovens were non-by-product and were introduced to accelerate the coking process and increase the quantity of coke from a given quantity of coal. In the beehive oven the gas evolved from the coal unites with air admitted above the charge and by burning heats the oven, and so makes coke. At the same time, however, there is a considerable loss of fixed carbon or coke through combustion. This is avoided in the retort oven. For example, from 100 pounds of coal the beehive oven produces 66 pounds of coke, while the Koppers retort oven furnishes over 80 pounds of coke. In addition to this the comparative daily output from each style of oven is in the proportion of 2 to 1 in favor of the retort. The next step in the improvement of retort ovens was to introduce the Seimens regenerators to preheat the air for combustion of the gas. It can be readily understood that if air is heated before coming in contact with the gas for combustion it adds its own heat to that of the flame. The higher the temperature to which the gas is raised, the less gas will be required to heat the ovens. By using the regenerators, only from 40 to 50 per cent. of the gas derived from the coal is needed to keep up sufficient heat to drive off the volatile matter from the coal. It is possible by the use of Koppers improved cross-regenerators to raise the temperature of the air to 2,000° F. The regenerators are constructed of a series of bricks built up with air spaces between the brick, as shown in Fig. 1, and so, resembling a checker board, it is called "checker work." After the gases of combustion have traveled in a given direction a certain time the air entering the regenerator cools the checker work on one side, while that checker work through which the hot gases pass to the escape flue is heated. After a certain time, usually 30 minutes, the air-current is reversed and the hot checker work receives the cool air and the cooled checker work is reheated.

The Koppers oven is arranged with vertical flues *F*, Fig. 2, and is a development of the Otto-Hoffman oven. The main feature in which the Koppers oven differs from the Otto-Hoffman, is in the distribution of air and gas, which causes combustion to take place in each separate flue along the whole length of the oven. This improvement compels the hot gases to pass up each flue, instead of making their way from a combustion chamber up any flue according to the line of least resistance. The result of this perfect distribution in the Koppers oven is the production of coke uniform in quality throughout the length of the charge.

That the iron and coke industry appreciates the merits of the Koppers ovens is evident from the fact that in 7 years over 5,900 of these ovens have been constructed, and in 3 years 1,376 ovens have been contracted for on this side of the Atlantic. In analyzing the causes of its popularity it will be found to be attributed to the high quality of the coke produced; its economy in the use of gas for heating the ovens; the individual type of regenerator; its simplified construction; and the increased saving in all by-products.

By-product ovens are constructed the same as retort ovens, but have arrangements for saving the tar, oil, and ammonium, which is distilled from the coal. The dimensions of the retorts in the different types of ovens vary from 16 to 20 inches in width; from 26 to 32 feet in length, and from 5.5 to 7 feet in height. In Fig. 1 the space marked "Oven" is the retort. On each side of the retort or coking chamber, and separated from it by firebrick walls, the combustion flues *F*, shown in Fig. 2, are constructed. The flue chambers being between two ovens, heat one side of each oven. To make a good homogeneous coke, an oven must be uniformly heated, and this the Koppers oven accomplishes through the combustion of gas in separate flues. There being 30 flues on each side of this oven, each supplied with a gas nozzle, there is a combustion of gas at 60 different points, which subjects the coal charge to the same intensity of heat at all points and the desired uniformity in the quality of the coke is accomplished.

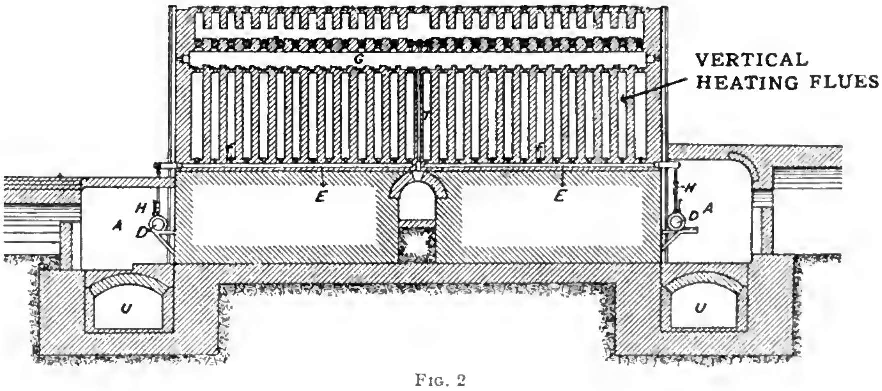


Fig. 2

The Koppers oven is exceedingly simple in design and the combustion of gas in the heating flues is easily regulated as the burners are all accessible for inspection, or adjustment, by means of openings into the vertical heating flues, on top of the ovens. An additional advantage gained by this uniform dis-

*Gas engineer for Koppers, Contractor for By-product Coke Ovens, Joliet, Ill.

tribution of heat, is the shortening of the coking time, thus increasing the efficiency of the ovens.

In the by-product regeneration coke oven, the gases evolved from the coal are conducted to the condenser and saturator house, where the tar and ammonia are recovered. Some of the cleaned gas is then returned to heat the ovens, while the surplus gas is available for power purposes or for distribution to domestic consumers. It is in the percentage of surplus gas that the different types of ovens essentially differ from each other. The Koppers oven, because of its improved system of heating flues and its efficient regenerator chambers, is remarkably low in its fuel consumption, and the yield of surplus gas is 50 to 60 per cent. of the total quantity evolved from the coal. Usually the regenerators of by-product ovens consist of two long chambers running the length of the battery and in case repairs to the regenerators are required this arrangement necessitates the whole block of ovens being shut down. To obviate this disadvantage and to enable the hot air to be more satisfactorily distributed to the heating flues, the Koppers system of regenerators was introduced. In this system there is a separate pair of regenerators, as shown in Fig. 1, for each oven in the battery. During a period of 30 minutes air is admitted into the regenerator at one end of the oven, and after being heated by contact with the hot checker brick in one-half of the regenerator, it enters the vertical heating flues where it unites in combustion with the gas entering the flues throughout one-half of the oven length. The products of combustion pass up these heating flues and down the vertical flues throughout the other half of the oven length, then through the other half of the regenerator, finally through flue *B* to the stack flue *U*. In their passage through the regenerator the waste gases heat the checker brick, and after 30 minutes, the gas valves and air dampers are automatically reversed. During the next 30-minute interval, cold air and gas enter at the opposite end of the ovens; the air is preheated by the previously-heated checker brick in that half of the regenerator, and combustion takes place in the vertical flues, which during the previous period served for the passage of the waste gases to the regenerator. The principal advantages of the Koppers cross-regenerator over the long regenerator type is that it permits of a better regulation of the air supply for each oven, and this air is heated to a higher temperature, thus promoting fuel economy.

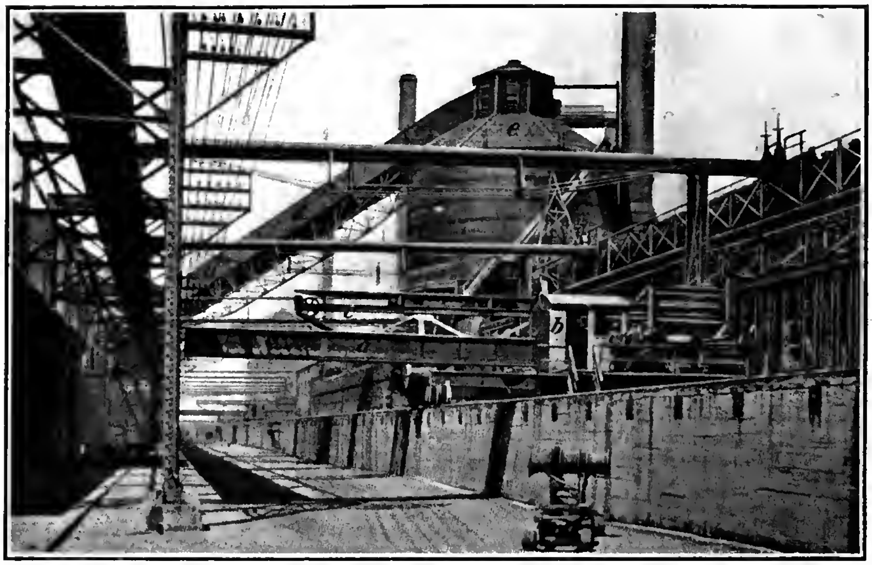


Fig. 3. Pusher Side of Koppers Coke Ovens at Joliet, Ill.

The working of the Koppers oven is exceedingly simple, and its construction is such that it is stronger than any other type of retort oven. The oven walls consist of a series of hollow columns reaching from the bottom to the top of the structure. Added to this, the uniformity and regularity of heating is conducive to a long life and a low repair account. Fig. 3 shows the pusher side of the ovens at Joliet plant of the United States Steel Co. The ram *a* projects from the lower rear of the pusher house *b* while the leveling machine *c* is above the ram and slightly to the rear of the pusher house. There is a belt conveyer housed in *d* that raises the coal to the top of the coal storage bin *e*.

Koppers patented direct process for the treatment of the gas and the recovery of ammonia has been installed in 28 plants, including 2,693 Koppers by-product coke ovens. In the old system of treatment, the gas was first cooled in condensers to remove the tar and was then washed with water in a series of

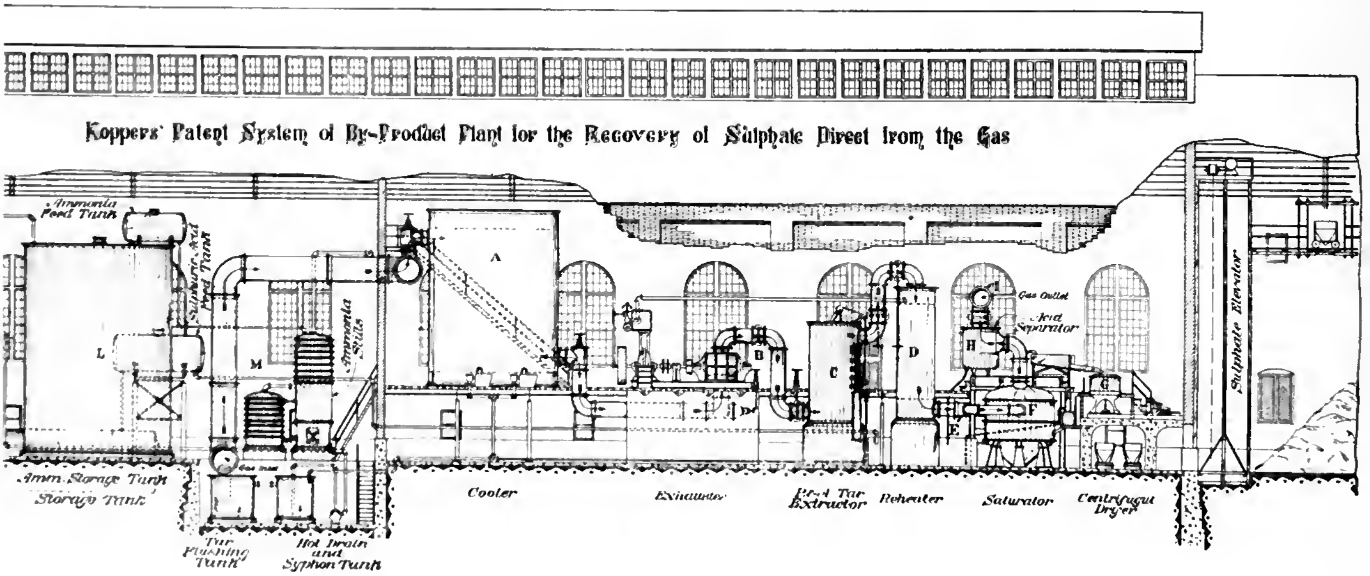


Fig. 4

scrubbers to remove the ammonia. The ammonia was collected as a solution of weak liquor which was then concentrated by treatment with steam in ammonia stills. The product from the stills was either in the form of concentrated liquor, containing 15 to 20 per cent. of NH_3 ammonia, or the hot concentrator vapors were passed into sulphuric acid saturators to form ammonium sulphate salt.

In the Koppers direct process, the former complicated process is much simplified, a larger yield of ammonia is recovered from the gas, and a superior quality of ammonium sulphate is produced.

In the Koppers direct process, the tar is removed by condensation in a tubular condenser *A*, Fig. 4, and by mechanical friction in a P. & A. tar extractor *C*. The gas, free from tar and at a temperature of 90° to 100° F., is then reheated by passing through a reheater *D*, the heat being supplied by the exhaust steam from the exhauster *B*, which apparatus pumps the gas with a constant vacuum from the ovens and forces it through the by-product apparatus. In the reheater, the temperature is raised from 155° to 165° F., at which temperature the gas passes directly into the sulphuric acid saturator *F*. Here all of the ammonia in the gas unites with the sulphuric acid in the bath and is recovered directly as ammonium sulphate. The gas, free from tar and ammonia, next passes through *I* into the balancing holder, not shown, which forces some of it back to the oven burners to heat the ovens. The balance of the gas, amounting to from 50 to 60 per cent. of the total, is available for gas engine fuel or other purposes. If it is desired to market this surplus gas, it is necessary to pass it first through iron oxide purifiers to remove the sulphur existing as hydrogen sulphide, and then into a storage holder supplying the distribution system.

Koppers direct process has many advantages over the old system of gas treatment. Besides being a much simpler process, it is decidedly more economical in operation. In this process, the only ammonia liquor to be handled is that which condenses out of the gas in the foul gas main, coolers and tar extractors; while in the old system, the gas is scrubbed with water to dissolve out the ammonia, thus producing large volumes of ammonia liquor to be stored and concentrated, before the ammonia can be absorbed in the saturator. All this handling of ammonia liquor results in more or less loss of ammonia, and is the reason why the direct process gives a much higher yield of ammonia per ton of coal. Also, the expense of recovering ammonia is less by the direct process, owing to the large amount of ammonia liquor to be pumped and concentrated in the old system.

A NEW TILE-LAYING MACHINE

Written for Mines and Minerals, by Bert Lloyd.

The accompanying illustrations are self-explanatory of a device that has recently made its appearance in Southern Colorado, which bids fair to reduce the cost of coke-oven maintenance, at the same time increase the production of a battery of beehive ovens.

Fig. 1. Tile-Laying Machine

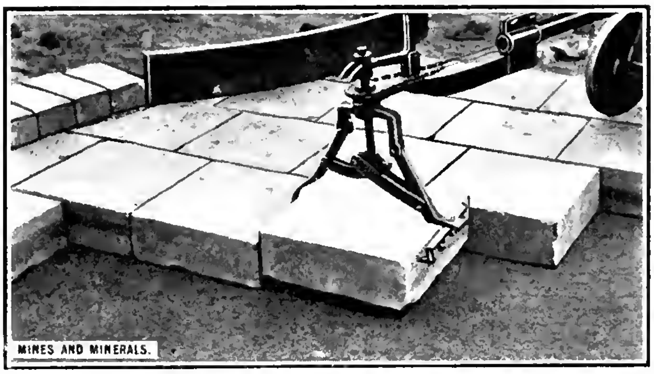


Fig. 2. Laying the Tile

By its use, a mason with a helper can relay the bottom of a 13-foot beehive oven with 12″×12″×4″ floor tile within 4 hours after the oven has been regularly drawn. This time includes the removal of the old tile, sanding the oven floor, and laying the new tile, then if the oven charge is slightly reduced the oven may be drawn in its regular order, and it will produce as high a grade of coke as though the bottom had not been disturbed.

The oven, therefore, loses absolutely no time in its operation; a very important item in cases where every oven is needed to produce coke; also the expense of reheating the oven with wood and small charges of coal is entirely eliminated.

The tile-laying machine as shown consists of a pair of tongs attached to a pipe bar by a clamp. Wires pass through the pipe bar and can be manipulated by the operator by means of levers so as to open and close the tongs as shown in Figs. 2 and 3.

The device is fully protected by patent, and now that its great value has been thoroughly demonstrated, its inventor, Mr. Michele Falcone, is negotiating for its manufacture in quantities sufficient to meet the demands that will undoubtedly be made as soon as its expense-reducing capabilities are known.

Accompanying the machine is a small adjustable metallic diagram, drawn to scale, by the use of which the tile around the edge of the floor may be scribed and cut to any exact fit before being placed. This is also an invention of Mr. Falcone, and has proven an invaluable adjunct to the main device.

Fig. 3. Machine Inserted in Oven

PROTECTIVE VALUE OF HUMIDITY

Written for Mines and Minerals, by James Ashworth, M. E.

Does Moisture Have Any Protective Value Against the Extension of Explosives?

The disaster at the Marianna Mine, in Pennsylvania, particularly directed the attention of mining men to the long and continuous debate which this subject has caused, and it is probable that when the full report of the experiments made at Altofts, in England, by a committee of the Associated Coal Owners is published, it will cause even more interest to be attached to the subject.

Before referring to the lessons taught by the Marianna explosion, the writer proposes to make a few remarks on the long papers contributed to the American Institute of Mining Engineers by Mr. Carl Scholz, and later by Mr. A. Haas. The authors of both papers appear to have based their arguments on very insufficient data. Thus, in none of the cases referred to, do the authors give the weight of water vapor per cubic foot of air, though both refer to "percentages of humidity." Now, the percentage of humidity does not offer any universal datum line from which the value of the percentages can be readily compared. Again, neither of the authors gives a statement of the condition of the outside atmosphere during, or at the time of, any of the disasters; nor do they say whether or not they occurred at times of greatest demand for fuel. Further than that, neither of them ventures an opinion as to the percentage of water saturation which is necessary to make the ventilating air-currents safe during any particular period of the year. As these details are not given, the writer will endeavor to prove that spraying and watering as practiced, and as suggested by Messrs. Scholz and Haas, cannot make a mine safe, and that it cannot control the extension of an explosion after it has once been properly initiated.

Experiments made in Great Britain on the rate of explosion in gases (Prof. H. B. Dixon) demonstrated, and the results have remained unquestioned, (1) that an explosion is impossible in an absolutely dry atmosphere; (2) that mixtures of air and explosive gases require 5 per cent. of water vapor per cubic foot to produce the most violent effects. Now 5 per cent. of water can only be present in the air in the form of steam, and therefore we drop at once into line with those people who concede that mine air ought to be artificially warmed and dampened. But a coal mine ventilated with a steamy air would certainly be a novelty, and the important subject we have to consider is not solely its problematical protective value, but also its effect on the miners' health, and on the roadways of the mine. Wet air is therefore a most serious proposition, and when studied from a really practical point of view, means that there must be present not less than 25 grains of water vapor per cubic foot; that an increased mortality from falls of roof and sides must result; and that the repair costs of every mine will be immensely increased. Supposing that these preliminary difficulties are surmounted, we have then the temperature to deal with. It has been found that 98° F. is the limit of heat in which a strong man can do any work when the air is saturated with moisture (18.7 grains per cubic foot) and therefore, we have at the outset the clearest proof that it is quite impossible to apply sufficient water in this form to restrict the flame of an explosion. From actual experience in English deep mining, it has been found that air having a temperature of 87 degrees, and carrying 13.6 grains of water vapor per cubic foot, is more oppressive to work in than air of 89½ degrees carrying 7.9 grains; and that men working in an atmosphere of 86 degrees containing 11 grains of water per cubic foot complain greatly of exhaustion. In Scotland, a temperature of 80 degrees, with 10¼ grains of water per cubic foot of air was found to be so exhausting that only certain men could work in it. The capability of a miner to work in a hot damp mine depends not only on the weight of water vapor in the air, but also on the velocity of the air, which ought not to be less than 300 feet per minute. This speed gives a sense of coolness to the skin, without being high enough to make the current dusty. Hence, it results that there are three principal factors which must be observed for the sake of the miners' health and efficiency; viz., (1) the temperature of the air-current, (2) its humidity, and (3) its speed.

The arguments of Messrs. Scholz and Haas go to prove that a certain number of grains of water are valuable in the ventilation air-current because their cooling effect disposes of a certain number of British thermal units, if the flame of an explosion comes in contact with them; but the writer considers that the practical demonstrations of the Marianna Mine explosion effectually disposes of this theory. According to the information at hand, this mine was most thoroughly and effectively watered, the intake air being dampened by the discharge into it of the whole of the exhaust steam from the pumping engines. It needs no argument to convince any person who is slightly acquainted with coal mining, that the air on entering a mine at the tunnel mouth, or the pit top, is of varying temperature, according to the period of the year; and for the same reason that its water content varies. On following this air into the mine, we find that it immediately commences to increase in temperature, and therefore its capacity to absorb water increases; but we also find that at a certain point the heat of the air reaches a normal degree all the year around. This fact is perhaps the most noticeable in a mine worked from a deep shaft, as then the normal point is found at or near the pit bottom, and in like manner if the shaft is damp, the air contains a regular charge of watery vapor. There is no doubt that dampness from sprays will cause some dust to clear out of the air more readily than if the air were quite dry; but even then it will not cause all the dust to fall; and the dust which the writer always considers the most dangerous, viz., the very dry and impalpable which is produced every moment by friction during the transit of the coal from the face to the pit bottom, still floats along in the air supported (if from a gaseous mine) in its own balloon of gas. This gas is stored in the pores of the coal, and is dissipating every moment and mixing with the air of the mine. The only place in a deep pit where water sprays can be applied with advantage, is at and about the pit bottom, where the air is cool, and where, when saturated with water, it would not contain more than, say, 4 or 5 grains of water per cubic foot of air.

Opinions differ greatly as to the quantity of coal dust it is necessary to have in the air to form an explosive mixture, and the writer is of opinion that the quantity normally found in a brisk air-current is amply sufficient for the purpose. The fine dust which has already settled on the timbering, sides, and floor, may play a part, but if it does, it is in a secondary sense, that is, after the flame has in passing distilled gases from it, and thus created that deadly atmosphere of gases of which carbon monoxide is the chief and most dreaded ingredient.

During the time the British coal-dust experiments were being made at Altofts, England, no difference in effect produced by an explosion was observed between the times when the air circulating through the testing gallery was saturated with moisture, and other occasions when the air had only 3.7 grains as a water content.

In many explosions, take for instance Udston, in Scotland, the worst coking effects were in rooms where there was standing water on the floor. At the Marianna, the places naturally wet showed evidence of as severe, or even more severe, effects from flame than the drier places. In the case of the Universal Colliery, in South Wales (very full reports of which are available) the whole of the east side districts were visited by the flame, although they were naturally wet, and the return air from the east district was practically saturated with water (nearly 8 grains per cubic foot). In this case the wetness of the strata combined with the dampness of the air did not restrain the flame.

The dampness of the air may also assist in the production of another phenomenon of colliery explosions; viz., that of percussion, and the writer is of the opinion that an excess of dampness in the ventilating current will thus assist in the extension of the area affected by an explosion. At the Universal Colliery, South Wales, this percussive effect was particularly demonstrated, and left indelible traces in the Ladysmith district, where men were struck dead in the positions and attitudes they were in at the moment of the explosion. At the Monongah explosion, Prof. H. M. Payne informs me that there were a very large number of cases where men were thus transfixed by death, with food in their hand or in their mouths, and so little disturbed that the caps were not blown off their heads. The fact of this phenomenon being present was also attested by the fact that in at least three places the indications were such that the initiatory point of the explosion might have been attributed to any of them. Without the effects of air percussion, or, as some prefer to style it, concussion, can be counteracted, the question of the safety supposed to be attained by watering is scarcely worth troubling about, and watering simply becomes a question affecting the sanitary condition of the pit. Wherever watering can be carried out without injury to the miners, health, and to the roadways of the mine, it may be applied judiciously in the main shunts or sidings, by spraying each car of coal as it passes underneath at a slow speed. This is a most effective way of minimizing dust, and a woodcut of this arrangement was, the writer believes, attached to Mr. Scholz's paper on watering.

The question of the prevention of explosions is not one of local or district importance, but is one in which the whole world is interested. It has been clearly demonstrated that the use of explosives is really the most serious danger in a dusty mine. We must therefore look for a radical cure by finding a safer mode of breaking down coal, or in the abandonment of blasting. As this cannot be always economically carried out, blasting ought to be restricted to a charge limit. In North Staffordshire, a leading engineer expressed his opinion that if the charges of high explosive were limited to 10 ounces, and reasonable care was exercised, mining would be practically safe. Then again, the question has arisen as to what constitutes a "dangerous" atmosphere. In England this has been to some extent answered by Messrs. Cadman and Whalley, in a report made to the Royal Commission on Mines, in which they suggest that 2 per cent. of gas in the return air or ventilating current should be considered dangerous. Heretofore, it has been thought impossible for any ordinary man to detect this percentage by a safety lamp flame, but now with the aid of a magnifying glass a trained man with sound eyes may see the cap of 2 per cent. of firedamp on the flame of a double-gauze bonneted Marsaut lamp, fitted with a wick .45 inch broad by .10 inch thick, when the flame is reduced down to $\frac{1}{10}$ of an inch high. It is not, however, at all likely that this percentage will be accepted by mine owners in England as "dangerous," though they might agree to 3 per cent. This is a most serious question for the owners and officials of all gaseous mines, especially where the hardness of the coal or the roof necessitates blasting, and if such drastic regulations were introduced into the collieries of the United States, the effect would doubtless be even more serious than in England, where the sale price can be more readily advanced to meet the extra cost of production. So far as blasting in the coal mines of the United States is concerned, the remedy lies mainly in the use of an increased number of shot holes, with a decreased charge (charge limit) and the usage of such high-class explosives as may be approved by the rules regulating mining in the several states. So far as watering is concerned, it was suggested by the three foreign experts who recently made an inspection of several coal mining districts at the request of the Federal Government, that the places where shots were to be fired should be well watered for 20 yards. It must, however, be remembered that, if a shot blows directly down the center of a road, this watering can only give the minimum of protection, and therefore it is practically certain that a double hurdle of wet blankets would be much more effective and less costly.

In concluding these short notes, the writer expresses his firm opinion, based on practical experience, that as demonstrated by a large number of colliery explosions, the humid state of the ventilation air-current of a mine has no retarding influence on the progress of either a coal-dust or a firedamp explosion.

TRADE NOTICES

The J. C. Stine Co., of Tyrone, Pa., is installing a 10-foot centrifugal fan, complete, with a 50-horsepower medium-speed engine at the mines of the Duncan-Spangler Coal Co., Inc., in the Clearfield, Pa., region. This fan casing is being built entirely of iron and steel, making it fireproof and practically indestructible. The fan will replace one 12 feet in diameter, and is guaranteed to circulate twice the quantity of air. The Stine fan is of the double inlet type, which has proved so satisfactory at many other mines, and we hope, when it is in smooth running order, to publish a report of tests showing its efficiency.

The J. C. Stine Co. has won an enviable reputation in all parts of the country for the successful production of coal-mining machinery, capable of effecting marked reduction in mining costs. A new bulletin just issued by the company is of interest to coal-mine managers, and will be mailed free on request.

The General Electric Co., recently shipped to the Rochester Coal and Iron Co., Punxsutawny, Pa., two switchboards, one of 18 panels for the main station and one of 14 panels for the substation. The entire switchboard equipment, including ammeters, instruments, and circuit breakers, was furnished by the General Electric Co.

The Westinghouse Electric and Mfg. Co. has recently received an order from the La Blanca and Anexas Mining Co. for 40 motors to be used in the company's mill at Pachuca, Mexico. The motors ordered range in size from 5 horsepower on the pulp thickeners, to 75 horsepower on the tube mills. The order also includes seven 250-kilowatt O. I. S. C. transformers, and one 12-panel switchboard.

Kolesch & Co., 138 Fulton Street, New York, have recently published a new and revised form of traverse sheet. The old sheets are used by surveyors, who will be interested to know that they are now in a position to procure a uniform-sized sheet for their calculations. The sheet is divided from left to right in columns headed as follows: Line; Bearing; Distance; Logs of Sines and Cosines; Addition of Logs; N; S; E; W; Coordinates; D. M. D.; N. D. A.; S. D. A.; Remarks. Size of sheet 19 in. × 15 in.

The Jeffrey Mfg. Co., of Columbus, Ohio, has changed the location of its Denver office to the rooms in the First National Bank Building. This company also maintains a corps of engineers at its branch offices in Chicago, St. Louis, Denver, Montreal, Pittsburg, Charleston, W. Va., Boston, New York, and Birmingham. There are also nearly 100 Jeffrey agencies in other cities in the United States and abroad.

SOCIETIES

The American Society of Engineering Contractors held their annual convention in St. Louis, on September 27, 28, and 29, in the Coleseum. The local committee of arrangements was E. H. Abadie, chairman; J. L. Westlake, W. C. Swartout, and L. C. F. Metzger. Papers were read by J. B. Goldsborough and Ed. Wegmann, on "Dam Construction for City Water Supplies," and by George C. Warren on "Work Preliminary to Street Paving and Road Work." A banquet was held, and several sight-seeing trips made to important engineering works in and around St. Louis.

SAMPLING COAL AND COKE*

By E. G. Bailey, M. E.†

Taking Samples from Vessel Cargoes, Mechanical Conveyers, Carts, Stock Piles, Bins, Etc.

It is useless to attempt to take a representative sample of coal from a vessel unless it is loading or discharging, for a sample taken from the hatches of a loaded vessel would probably consist of coal from only as many cars as there were hatches, while there may have been 100 to 300 cars of coal emptied into the cargo. Occasionally it is necessary to take a sample from the broken surface of a cargo that has been partially discharged, but the amount of coal accessible to the sampler is so small a proportion of the total cargo that such samples should be avoided if possible. Sometimes cargo coal is sampled as the vessel is being loaded, in which case the sample is taken from railroad cars as they are dumped on the pier, but the majority of cargo samples are taken as the vessel is discharging. The total amount of original sample to be taken should be governed by the quantity and size of impurities in the coal, rather than the amount of coal to be sampled.

The method of accumulating the sample will depend a great deal upon the local conditions. If the coal is being discharged into a hopper and thence to a mechanical conveyer, industrial railway, carts, or railroad cars, the sample can be taken from them. Taking samples from mechanical conveyers is described later under that subject. If the sample is taken from an industrial railway, a shovelful should be taken from every so many cars, the number of cars per shovelful being estimated from the capacity of the cars, the size of the cargo being sampled, and the quantity of coal desired for the sample. If the sample is taken from railroad cars, the number of shovelfuls per car should be determined in like manner, and in either case the shovel should be pushed deep into the coal in order to bring the portions of the sample from beneath the surface. The shovelfuls should be taken alternately from different parts of the surface, in order to prevent an undue proportion of coarse or fine, which may tend to be separated when dropped or run into the cars. The coal is more uniformly mixed by the time it is discharged into the cars and dumped in larger quantities, thus giving less chance for segregation of slate than when loading railroad cars at the mines, hence many precautions which are necessary to obtain a representative sample from an individual railroad car loaded at the mines may be disregarded in sampling cargo coal under similar conditions.

It is sometimes necessary to hold up the grab bucket every so many trips and take a shovelful directly from it, but this retards the discharging and causes additional labor, especially in cases where more than one tower is being used for discharging.

If it is impossible to obtain a sample by any of the foregoing methods it can be taken from various parts of the exposed surface in the vessel as the coal is discharging.

A special moisture sample should always be taken.

Some authorities claim that more than one sample should be taken and analyzed in order to determine the true quality of even 500 tons of coal, but one sample, properly taken and reduced, will represent 5,000 tons or more just as accurately as one sample will represent a 50-ton carload. The following data, taken from a 6,000-ton cargo during the entire time of discharging, bear out this contention:

Variation between results of one 2,300-pound sample and average results from twenty 135-pound samples: Ash, +.06 per cent.; sulphur, +.01 per cent.

Variation between results of one 2,300-pound sample and the average results from four 560-pound samples: Ash, −.04 per cent.; sulphur, +.02 per cent.

* See also article Coal and Coke Sampling in Mines and Minerals for September, 1910.

† Fuel Testing Co., 220 Devonshire Street, Boston, Mass.

Other similar data taken from 20 consecutive cargoes of 5,000 to 7,000 tons capacity, where three 1,000-pound samples were analyzed and compared with one 3,000-pound sample from the same coal, are given Table 1:

Table 1. Variation Between Results of One 3,000-Pound Sample and the Average Results From Three 1,000-Pound Samples

Cargo	Ash	Sulphur
1	+.03	+.06
2	+.15	+.02
3	−.03	+.01
4	+.09	+.12
5	+.22	+.01
6	−.04	−.04
7	−.02	
8	−.20	−.03
9	−.05	+.02
10	−.04	+.05
11	−.02	+.03
12	−.22	+.02
13	+.02	+.02
14	−.23	+.02
15	−.43	+.03
16	+.01	+.02
17	−.10	+.13
18	−.05	−.03
19	−.07	−.01
20	−.11	+.05

The difference between the results of one sample and the average of several is, with very few exceptions, within the limit of accuracy in making check analyses from the same pulverized laboratory sample. There is no doubt but that a 12,000-ton cargo of coal could be represented by one sample with equal accuracy, if the sample were properly taken and reduced.

Sampling From Mechanical Conveyers.—Sampling coal or coke is simplified when it can be taken from a mechanical conveyer. In many cases the coal has been crushed before it is delivered to the conveyer, so that a smaller sample need be taken than if it were accumulated from the run-of-mine or larger coal.

The two types of conveyers most generally used are the belt and bucket. When sampling from the latter, an entire bucket may be dumped at certain intervals, or a shovelful may be taken from the buckets as they pass. A 3- to 5-pound shovelful is ample if the coal has been crushed to 2-inch or smaller. The portions of a sample may be taken from a belt conveyer by shovel, providing care is used to obtain a true proportion of the coarse and fine. There is a tendency for the fine coal to settle toward the middle of the belt and the lumps to remain on the surface and roll to the outer edge, hence it is better to scrape off a complete cross-section of the strip of coal at certain intervals.

The quantity of coal to be taken for each portion of the sample should be determined from the quantity of coal to be sampled and the size and quantity of the impurities, so the required amount of gross sample will be accumulated by the time the coal to be sampled has all been handled by the conveyer. The time between taking the different increments of sample should be varied according to the rate at which the coal is passing, rather than equal periods of time. This is especially necessary when the coal is coming from a discharging vessel and is unloaded more slowly near the last when the vessel is being trimmed and cleaned up.

Mechanical Sampling.—Whenever coal is being received regularly at any one place and samples are to be taken frequently, a mechanical sampler should be installed if possible. A mechanical sampler is merely an automatic method of accumulating portions of a sample from coal as it is being handled, and is usually confined to sampling crushed or fine coal from mechanical conveyers or gravity feeds.

When coal is periodically dropped from a weighing or measuring hopper, a small opening 4 to 6 inches square can be opened every time the hopper is emptied, and the original sample accumulated in this manner. A moving bucket or spout can be arranged to periodically move across a stream of coal falling from a hopper or conveyer. A miniature bucket conveyer, with one bucket, can be arranged to travel under a stream

of coal and carry the portions of the sample to any desired place before dumping it. In the case of a bucket conveyer, one or more buckets may be arranged to dump into a chute. A scraper may be periodically lowered on to a belt conveyer, and various other devices made to suit the local conditions However, in all of them great care should be used to prevent an undue proportion of lump or fine, and also to adjust the mechanism so the required amount of sample will be taken from the desired amount of coal.

If possible, all subsequent crushing and dividing of a sample should be done mechanically. There are a few satisfactory kinds of crushers, and other cheaper ones will undoubtedly be placed on the market as the demand for them increases. A series of riffle samplers, as shown in Fig. 1, or a modification of it, is undoubtedly the best method of dividing a sample.

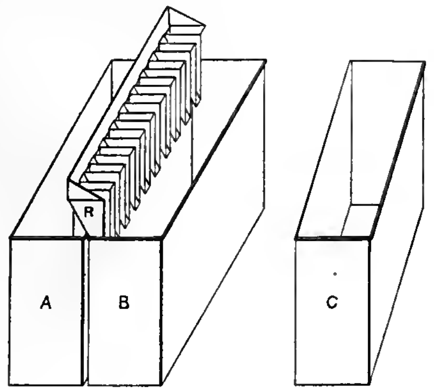


Fig. 1. Riffle Sampler

Sampling From Carts and Wagons.—Coal delivered by wagon is frequently purchased on the analysis basis and samples are usually taken as the coal is delivered. Shovelfuls should be taken throughout the entire quantity as it is being dumped or shoveled out, in about the same manner as in the case of a railroad car. As a rule, the sample is accumulated from several wagon loads, and may represent one day's delivery, or else may be taken throughout a period of one week. In any case, the entire sample should be kept in a hermetically sealed can until it is all accumulated, or else a special moisture sample should be taken. The fact that only a few tons are delivered during the time covered by a sample does not obviate the necessity of taking a sufficiently large sample that it will be truly representative, for no sample at all is better than inaccurate or nonrepresentative figures.

Sampling in Boiler Plants, Gas Houses, Etc.—Samples of coal are frequently taken in boiler plants to represent the coal being burned on a test, in which case the sample should represent the coal as weighed to the boilers. If the coal comes from overhead hoppers, a shovelful should be taken from the spout or hopper at regular intervals. If the coal is brought in by wheelbarrow, industrial railway car, or wagon, the shovelfuls may be taken from the coal as loaded, after it is dumped on to the floor, or from the wheelbarrow or cart itself. The shovel should always be pushed deep into the coal and filled from coal below the surface. It must be remembered that the quantity of coal sampled does not govern the size of sample to be taken, for the sample taken from 5 tons used on a boiler test should be just as large as a sample taken from a 50-ton railroad car. A very common error in sampling coal as weighed in boiler or other plants is due to escape of moisture from the sample. A loss of 2 per cent. moisture will cause an equivalent error in the calculated boiler efficiency. Failure to take special moisture samples has been the cause of many erratic and erroneous results from boiler tests. The loss of moisture from coal between the time it is discharged from vessel or car until it is weighed to the boilers, especially where it is stored in overhead bins, accounts for many apparent discrepancies in weights if the moisture is properly determined at the two places.

Sampling From Stock Piles and Bins.—There are few places where it is more difficult to obtain a truly representative sample of coal or coke than from a large stock pile or bin. There is so small a proportion of the total quantity available near the surface that at best a sample can only approximately represent the total pile. Where coal is dropped from one overhead position, and the coal forms a conical pile, that on the surface can represent only the last of the coal dumped. A pile of this shape containing 1,000 tons has about 25 per cent. of the total quantity within 1 foot of the surface, and larger piles have a still smaller percentage of the coal available. If the coal has been dropped from an overhead track or trolley and has formed a long pile of triangular cross-section and 25 or 30 feet high, there is less than 10 per cent. of the coal within 1 foot of the surface. But such a pile usually has the advantage of representing on its surface some of the coal from practically all parts of the cargo or cars discharged. However, in either case there is the additional difficulty of obtaining a true proportion of lumps and fine coal. When coal is dropped on to a pile, the larger lumps and heavier pieces of impurities roll to the bottom and the finer coal remains in the center of the pile and on the top.

If a conical pile is divided into four parts by passing lines about it equally spaced between the bottom and top, the relative volume and surface between them is as follows:

	Percentage of Total	
	Volume	Surface
First, or top part	1.6	6.3
Second part	10.9	18.7
Third part	29.8	31.2
Fourth, or bottom part	57.7	43.8
Total	100.0	100.00

From this it would appear that 57.7 per cent. of the sample should be accumulated from the bottom part of the pile, because that is the percentage of coal contained in this part of the pile. But it contains some fine coal in the center and only the coarser coal on the surface, so it would not be right to take 57.7 per cent. of the sample from the large lumps alone. It has been found by actual experiment that the true proportion of lumps and fine can only be obtained by taking equal increments of sample from equal areas of the exposed surface, even though they do not represent equal volumes or quantities, but at best it must be remembered that only the coal near the surface is available when the sample is taken by shovel.

From the total approximate area of the surface of a pile and the quantity of sample to be taken, one can estimate the area from which a shovelful should be taken. For instance, if a 1,500-pound sample is to be taken from a conical pile containing 1,000 tons, there will be about 11,000 square feet of surface exposed, and 100 15-pound shovelfuls should be taken, or one shovelful to every 110 square feet, or from points about 10 feet apart.

The best method of taking the sample is to begin 3 or 4 feet from the bottom and take shovelfuls at the predetermined distance apart or along the pile on the same level. Then start a certain distance up the pile and circle it again, and so on, keeping the distance between points of taking shovelfuls as nearly equal as possible. One shovelful should be taken from near the surface, the next one deeper, the next still deeper, the next from the surface, and so on, in order that the sample will be taken from different strata. The depth to which one may dig depends a great deal upon how wet the coal is and how easily is slides By beginning at the bottom of the pile and working around and up, the coal which slides down does not interfere

with that part of the pile not already sampled, as would be the case if the work were started at the top.

Sampling from a bin is even more difficult and unsatisfactory than sampling from a pile, owing to the restricted surface exposed.

Sampling by Pipe.—The difficulty of digging deeply enough into a car, pile, or bin with a shovel when sampling coal has led many people to suggest the driving of pipes or tubes and withdrawing them full of coal. Some advocates of this method claim satisfactory results from it, but a great deal of practical experimenting with pipes 3 or 4 inches in diameter has proven it to be unsatisfactory, especially for coal containing lumps. If the coal packs, it quickly plugs the end of the pipe and very little coal is obtained from any considerable distance below the surface. If there are lumps as well as fine coal, there is great difficulty in obtaining a representative proportion of the lumps, as they tend to split open and permit the pipe to pass between the parts rather than filling with a core out of the lump. As a rule, that coal which is already in the pipe is jarred out when it strikes a lump. A coal-mining auger encased in a pipe or tube has given little better results, for with this arrangement it is possible to fill the pipe with coal, providing the spaces between the flat sides of the auger and pipe are large enough for the largest pieces of coal to pass freely. In coarser coal, the auger tends to push the lumps to one side or glance past them, but if one is hit fairly it may be drilled through; however, most of the fine coal made from drilling the hole drops out as soon as the auger cuts through to the bottom side of the lump. It is practically impossible to drill through a lump of slate or hard bone when lying in a pile of coal, hence the sample taken by pipe or encased auger would tend to be better than is the actual coal. The accumulation of a sample by pipe is much slower than by shovel and it does not obviate the necessity of taking a large sample. In short, the small-pipe method of sampling coal is not practical nor accurate, regardless of how nice it looks theoretically, but the use of larger pipes of 10 or 12 inches in diameter may be possible under certain conditions.

A variety of methods have been used to crush coal in the laboratory from 4 mesh or larger to 8 or 10 mesh. Jaw crushers and other types of machines have been used, but a bone mill has proven to be the best all-around machine for this purpose.

The division of a sample in the laboratory after it has been crushed may be done by thoroughly mixing it on a table or cloth and dividing it into four parts by means of a spatula. Special quartering machines may be obtained on the market but a riffle sampler has proven to be the most satisfactory method of dividing samples, and it has considerable advantage both as to speed and accuracy. A sketch of this sampler is shown in Fig. 1. Its operation is as follows: A sample which has already been crushed and is ready for further division is placed in can *C*. The riffle sampler *R* is placed astride the adjacent sides of cans *A* and *B* and the sample in can *C* is poured on to the top of the riffle *R*. This riffle consists of 20 spouts or troughs ½ inch wide, the adjacent ones pointing into opposite cans, and the stream of coal passing out of can *C* is divided into 20 parts. Half of the sample will then be collected in each of the cans *A* and *B*. Can *B* is then replaced by can *C* and the part of the sample in *B* is poured on to the riffle and can *C* will then contain one-fourth of the original. The operation can be repeated until it is reduced to the quantity which it is desired to pulverize. The riffle sampler shown in Fig. 1 is simpler than those usually described as it does not have any frame nor legs but merely rests on the adjacent edges of the two cans. This form of riffle is more accessible and easier to clean than are some of the other patterns.

It is so often desirable to have a considerable quantity of sample on file that several hundred grams should be pulverized and retained. The final sample should be pulverized to at least 60 mesh, while 80 mesh is preferable, but anything finer than this is both unnecessary and undesirable. The final pulverizing should be done in a ball mill.

There are nearly always some particles of bone, or other hard material, in the sample which are not readily pulverized; however, it is necessary that everything should go through a 60- or 80-mesh sieve, hence the entire sample, after it is thought to be pulverized to this degree of fineness, should be tested by putting all that is fine enough through a 60- or 80-mesh sieve. The remaining quantity, if a large percentage, should be returned to the ball mill, but if only a few grams, it can be reduced in a hand mortar. It is absolutely necessary that all the sample go through the sieve and it should then be thoroughly mixed. Instances have been frequently encountered where a sampler merely dipped out a quantity of coal from that which was crushed or partly pulverized and sifted enough of it through the 60-mesh sieve until the desired quantity of sample was obtained, and all of the coarser material was discarded. In many laboratories there seems to be a prevailing tendency for the sampler to discard the small quantity of material which is not readily pulverized. As errors arising from this cause often exceed 5 per cent. in ash and 1 per cent. in sulphur, it is certainly not asking too much of the chief chemist or man responsible for the work of the laboratory to occasionally follow samples through the complete operation and investigate every detail in order that errors may be eliminated.

Sixteen-ounce cream jars with straight sides and screw tops are preferable to bottles for keeping laboratory samples.

(*To be Continued*)

New Inventions

PATENTS PERTAINING TO MINING ISSUED AUGUST 2 TO AUGUST 23, 1910, INCLUSIVE

No. 966,377. Rotary mining drill, George G. Barker, Upland, Pa.

No. 966,378. Portable mining machine, George G. Barker, Upland, Pa.

No. 966,389. Wet process for the treatment of ores, Henry Thomas Durant, Henry Livingstone Sulman, and Woldemar Hommel, London, England.

No. 966,209. Obtaining zinc oxide from zinc ores and products, Woldemar Hommel and Henry Livingstone Sulman, London, England.

No. 966,521. Ore concentrator, George W. Burnhart, Ward, Colo.

No. 966,008. Ore screen, Frank Franz, Burke, Idaho.

No. 966,772. Method of and apparatus for pumping oil wells, Andrew Stattler, Carpinteria, Cal.

No. 966,651. Rock breaker and crusher, Walter J. Cochran, Los Angeles, Cal.

No. 967,859. Concentrating table, August Ten Winkel, Denver, Colo.

No. 967,741. Combination dredge and conveyer, Franklin P. Eastman, New York, N. Y.

No. 967,481. Drilling rig, Augustus C. Zierath, Longbeach, Cal.

No. 967,200. Electrolyte and method of electrodepositing zinc, Edward F. Kern, Knoxville, Tenn.

No. 967,745. Gold-saving riffle, Carl Erickson, San Francisco, Cal.

No. 967,671. Method of separating minerals, Alexander S. Ramage, Detroit, Mich.

No. 967,159. Method of reducing ores, Frederick M. Becket, Niagara Falls, N. Y.

No. 968,499. Process of coking coals, Leland L. Summers, Chicago, Ill.

No. 968,504. Mine exit, Clarence W. White, Lacon, Ill.

No. 968,100. Mining apparatus, Ajay Washburne, Fruitvale, Cal.

No. 967,885. Mining machine, Charles E. Davis, Chicago, Ill.

No. 967,884. Chain for mining machines, Charles E. Davis, Chicago, Ill.

No. 967,996. Method of extracting or eliminating sulphur, phosphorus, and other impurities from coal, ore, etc., Leland L. Summers, Chicago, Ill.

Mines and Minerals

Vol. XXXI—No. 4. SCRANTON, PA.—NOVEMBER, 1910—DENVER, COLO. Price, 25 Cents

COAL MINING IN OKLAHOMA

*Written for Mines and Minerals, by Willis P. Thomas**

The coal measures of Oklahoma are embraced in an area of approximately 20,000 square miles, of this 1,000 square miles is estimated as workable coal. The production at the present time comes chiefly from LeFlore, Haskell, Latimer, Pittsburg, Atoka, Carter, and Coal counties, all of which are in the coal belt, shown in Fig. 4. Coalgate is in Coal County.

Extent of Development. Quality of the Coal. Methods of Preparing, Mining and Hauling

Geographically the beds are a part of what is known as the Western Interior Coal Fields, the coal-bearing rocks belonging to the Pennsylvania series of the Carboniferous. The different beds, of which there are 12 or more, vary from a nearly semi-anthracite to a low-grade bituminous. The most persistent and valuable beds are the Upper and Lower Hartshorne and the Lower McAlester. There are four others which are of less importance, but commercially valuable; namely, the Upper McAlester, the two Witteville beds, and the Cavanal. The quality of the last four being inferior to that of the first three beds, they have not been developed to any great extent. The Lower Hartshorne bed lies on what is known as the Hartshorne sandstone, and where this sandstone reaches the surface it has been badly broken, due to folding, and besides has weathered until a distinct ridge has been formed, by means of which the Lower Hartshorne is easily traced from near Fort Smith, Ark., across the state line to Lehigh, Okla., a distance of over 100 miles. The upper Hartshorne is separated from the lower Hartshorne by about 50 feet of shale. Where worked the lower bed has an average thickness of 4½ feet, the upper bed about 4 feet. From 1,000 to 1,200 feet above the Upper Hartshorne is found the Lower McAlester bed, which excels in quality all other coal beds in this vicinity. It varies in thickness from 2½ to 4½ feet.

Fig. 1. Rock Island Coal Mining Co., Shaft No. 6

The development and mining have been done principally along the Rock Island; Missouri, Kansas & Texas; St. Louis & San Francisco; and Fort Smith & Western railroads. These roads have opened up quite a field and have made a wide outlet for the coal produced. However, oil and gas for fuel purposes and the competition of adjoining non-union fields have greatly retarded development and production the last few years. The coal mined at the present time is excellent steam fuel, very hard and capable of standing long transportation. The combustible volatile matter runs high, making the coal available for gas purposes. The following table shows that the Lower Hartshorne compares very favorably in quality with the Pittsburg seam of Pennsylvania, which is one of the best gas and coking coals in the country.

	Lower Hartshorne		Pittsburg Seam	
Moisture	1.46	1.30	1.22	1.28
Volatile matter	39.04	38.90	34.30	38.10
Fixed carbon	53.10	52.15	56.95	54.38
Ash	6.40	7.65	6.41	5.44
Sulphur	1.33	1.58	1.07	.79

The miners are paid for their coal on the mine-run basis, with the result that the quantity of slack produced has become a question of great importance. In order to make the slack more valuable, Mr. Carl Scholz, President Rock Island Coal Co., has recently erected a briquetting plant at Hartshorne, Okla. This plant has been running for some time successfully, making about 100 tons of briquets a day.†

This branch of the coal industry has not proved a solution of the slack problem in Oklahoma, as the market for this material is limited. Several of the larger companies are equipped with coke ovens, but these are operated only intermittently. Table 1 gives a comparison of the Hartshorne and Alderson (McAlester bed) coal and coke with that from the Connellsville region.

Throughout the field, wooden tipples, such as are shown in Figs. 1 and 5, are the rule, although the newer tipples will probably be of steel, as the Rock Island Coal Co. has recently had one constructed.

Self-dumping cages of different styles and hoisted in balance are used almost entirely at the shaft mines. The boiler plants are all equipped with horizontal return-tubular boilers, which are hand fired.

To prepare the coal for market, double balanced shaking screens are used, although several operators are equipped with slack storage bins where pea and slack are separated with rotary screens.

Usually four grades of coal are produced termed lump, egg, nut, and slack. In Fig. 1 the lump coal is loaded into box cars; the slack goes to the bin at the left, and from this bin is fed to the boilers or shipped. The waste rock is hoisted up the incline

* Wilburton, Okla.

†See Mines and Minerals, Vol. 30, page 581

to the left of the tipple in a car; the trestle, which rests on the rock pile, is extended from time to time as demanded.

The coal formations in this field have been faulted and folded, making the coal difficult to mine in comparison with most other fields. The mines are opened by slopes or shafts, sometimes by a combination of the two; of necessity the mines are slope mines whether opened by slope or shaft.

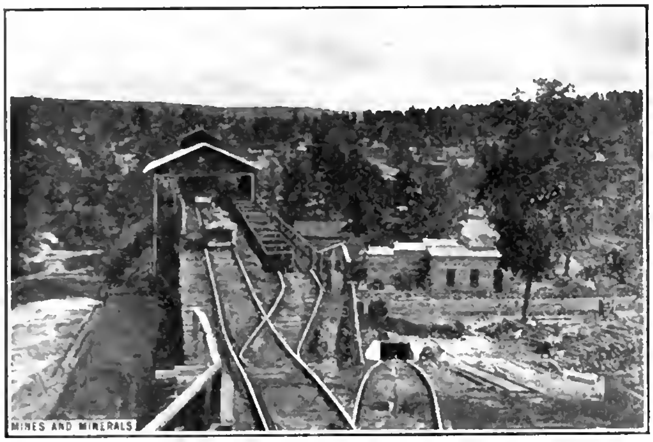


Fig. 2. Tipple Approach, Great Western Coal & Coke Co. No. 3 Slope

Fig. 3. Fan House and Tipple, Slope No. 3, Great Western Coal & Coke Co.

Fig. 2 shows the Great Western Coal and Coke Co.'s No. 3 slope, at McAlester, Okla.; also the approach to the tipple. The brick fan house is shown to the right above the slope mouth. The fan house and tipple is shown in Fig. 5.

Where a shaft is used, a main-haulage road is driven each way from the shaft parallel to the outcrop. From these main entries, cross-entries are driven to the rise and dip, off of which are turned the working entries. These entries are from 150 to 300 feet apart, depending on local conditions. Rope hoists, sometimes underground, sometimes on the surface, handle the cars in trains on the slopes.

Longwall advancing has been tried but never carried on successfully. The method commonly used, and the system now required by the new state mining law, is the double-entry room-and-pillar system. The top entry, being dry, is generally used as the haulageway, the water draining to the lower entry. Rooms are turned off the upper entry, directly up the pitch or at some angle to it; as a rule, rooms are from 200 to 275 feet deep, 20 to 25 feet wide, with room necks 8 to 10 feet in width. Four different methods are in use to place cars at the working face of rooms. (1) Up to 10 degrees the mules pull directly up the pitch. (2) At one mine double rooms are used, 40 feet wide. A track is carried up each side of the room and the rock gobbed in the center to throw the air to the face. At the end of each track near the face a post is placed which carries a sheave through which passes a $\frac{3}{8}$-inch wire cable. When a car is loaded on one track the cable is fastened to the loaded car, the other end of the cable goes around the sheaves and is attached to an empty car at the mouth of the room on the opposite track. The loaded car is then let down, which at the same time places an empty at the face on the other track. (3) At some mines, where the pitch is steep, a sheave is fastened to a post near the face of the room, a cable is then fastened to the car and the mule goes up and pulls down the room. This method gives the mule the advantage of the downhill pull. (4) At Coalgate, Okla., several mines are equipped with electric locomotives, which are provided with a drum and cable; this cable is taken up the room around a sheave and attached to a car. By this means empties are hoisted up and loads let down from the rooms.

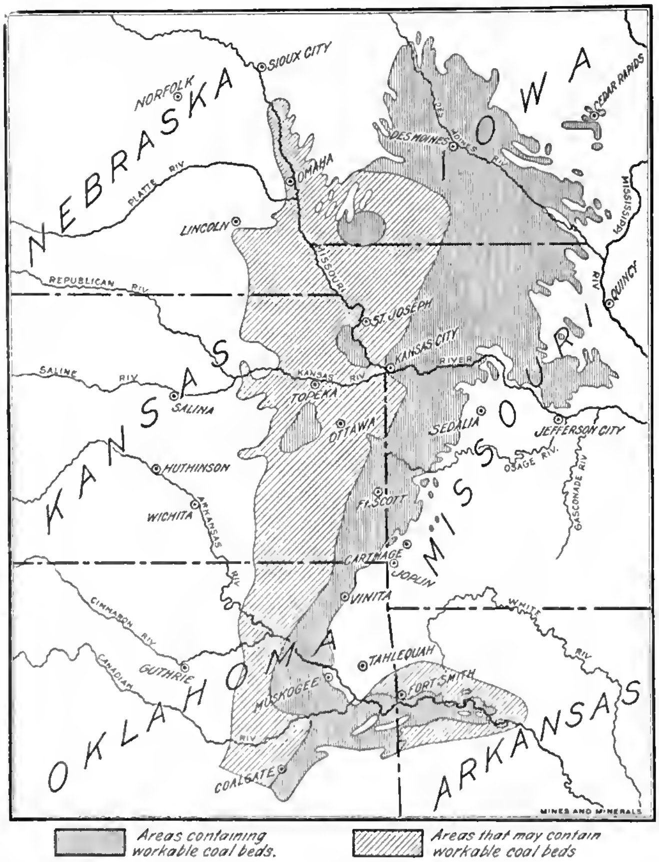


Fig. 4. Map Showing Location of Oklahoma Coal Fields

Where the pitch of the coal exceeds 20 degrees the coal is brought to the entries in chutes.

Pick mining is the rule throughout the state; vertical cuts being preferred to undercutting, the coal being shot from the solid. The steep pitch, the lack of competent runners, and the general antagonistic position of the miners, has worked against the adoption of machines. However, the Sullivan Machinery Co. have apparently solved the chain-machine problem for this field in their low-vein machine, for at Coalgate No. 5 shaft, Coalgate, Okla., they are working successfully up a 13-degree pitch, and cutting across the vein on a 25-degree pitch. Paying the miners on the mine-run basis as at present, will soon force the operators to the more general adoption of machines in order to increase the percentage of lump coal. Machines would also be of great benefit to the mines by reducing the heavy shooting which now takes place, and which proves to be disastrous to the roof at times.

TABLE 1. ANALYSES OF COALS USED IN TEST AT CHOCTAW COKE WORKS, ALDERSON, OKLAHOMA

Locality	Moisture	Volatile Matter	Fixed Carbon	Ash	Sulphur	Phosphorus	Remarks
Alderson lump.....	1.48	35.81	56.71	6.00	1.050	.020	McAlester seam
Alderson slack.....	1.71	34.16	55.69	9.24	1.020	.018	McAlester, washed
Alderson slack.....	1.61	29.50	53.77	15.12	1.100	.018	McAlester, unwashed
Hartshorne lump ..	1.68	38.59	53.91	5.82	2.830	.016	Shaft 1, Grady basin
Hartshorne slack ..	1.64	35.18	53.53	9.65	2.260	.020	Shaft 1, washed
Hartshorne slack ..	1.77	31.82	52.16	14.25	2.250	.022	Shaft 1, unwashed
No. 9 slack........	2.02	29.90	52.64	15.44	1.240	.016	Slack, unwashed
Wilburton.........	1.67	34.50	53.22	10.61	1.340	.018	Mine-run, unwashed
Mitchell basin	.41	19.29	76.53	3.77	1.060	.024	Outcrop, coal
Connellsville for comparison......	1.08	33.31	58.49	7.12	.060	.028	Mine-run
COKE							
Connellsville.......	.79	1.31	86.88	11.54	.695	.015	For comparison
Alderson..........	.95	2.00	84.90	12.15	1.420	.024	Slack, washed
Hartshorne........	1.55	1.34	90.68	6.43	1.840	.024	Lump, unwashed
3 parts Alderson, 1 part Hartshorne.	.98	2.43	85.10	11.49	1.120	.024	Slack, washed
2 parts Alderson, 1 part Hartshorne.	.46	1.77	83.33	14.44	1.770	.026	Slack, washed

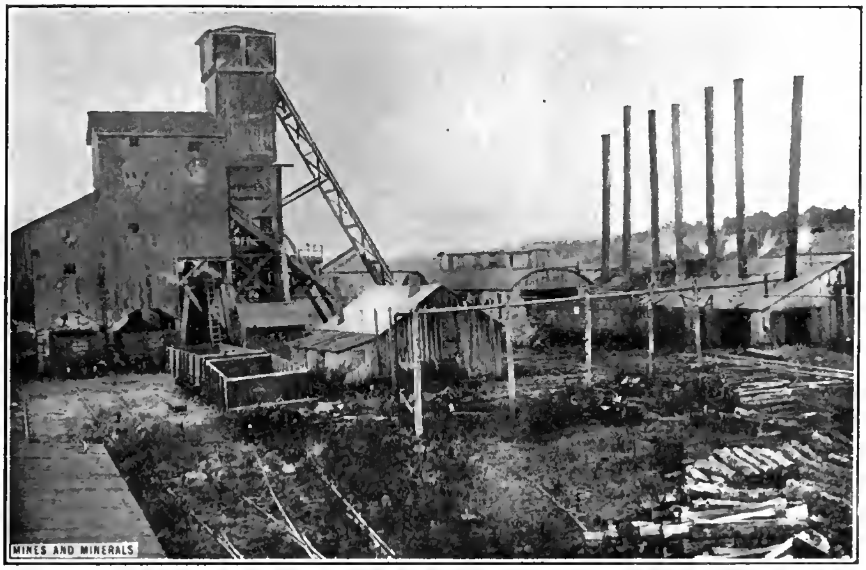


FIG. 5. OSAGE COAL AND MINING CO., SHAFT NO. 5

The amount of coal recovered by present operations will run from 50 to 70 per cent. This is as much as can be obtained under the present methods, drawing pillars is rarely practiced, the mines not being developed with this end in view.

The state mining law requires that all dry mines be sprinkled and kept in a damp condition. This was done by most of the larger companies before the law was passed. The method commonly used was to provide each entry with a sprinkling line of 1½- to 2-inch pipe to which hose connections could be made at regular intervals. These lines also served as a means of fighting fires which are frequent in certain parts of the field.

When the supervision of the mines passed from federal to state control the amount of air required for each man and mule was greatly increased. This increase made necessary the installation of many new fans at mines in different places. The tendency at present is to install fans of small diameter and high speed.

Owing to the many unfavorable conditions under which the mines of this state have to work, the cost of operation is high. The quality of the coal is such, however, that the mines are able to compete with the other operations in the Western Interior coal fields which are able to produce coal at a lower cost per ton.

The large increase in the production of oil and gas has been a serious blow to the coal-producing states of the Southwest, and Oklahoma has probably suffered the most, as the increase has been greater within the boundaries of this state than any other. However, the coal field will prove to be one of its greatest assets, and be one of the prime factors in the building of a great, new state.

AMERICAN *vs.* EUROPEAN COAL MINES

*Written for Mines and Minerals, by Henry M. Payne**

A Discussion of Foreign Mining Methods and Conditions. Points of Difference from American

During the past 3 years a number of the officials of the largest coal-mining operations in the United States have visited the mining districts of Europe, and a diversity of opinions and a multiplicity of suggested reforms have been the result.

The writer spent the summer of 1909 in the principal mining districts of England, Wales, Belgium, Germany, and France, devoting his time to the study of the coal-mining industry in these countries, and its aspect, not only from the standpoint of economic, labor, and general industrial, conditions, but also with regard to cost data, power equipment, methods of prospecting and development, systems of mining instruction in the universities, technical and mining schools, the Government Testing Stations and coal mines regulations, and a careful compilation of such facts as would make possible a valid comparison, as indicated in the title of this discussion.

One important condition to be clearly understood, is the short haul everywhere in Europe. For instance, from Yorkshire or Shropshire in the North of England, or from the famous Rhonddha and Taff valleys, in Wales, is less than 200 miles from London, while much of the product of these fields goes to Liverpool and Cardiff for sea trade, a distance of less than 50 miles. In Belgium, scarcely any of the coal travels 50 miles from the mines. In Germany the greater part is coked at the plant, and in France the entire output is consumed within 200 miles of its origin. On this account, small cars, a plentiful car supply, and limited freight rates (in Belgium and Germany over government-controlled railroads), all render simple, our American problem of the expeditious disposition of the product.

Another matter which attracts the attention of the American abroad, is the character of the labor. There, mining is a trade, almost a profession, and descends from father to son. The mining schools in many sections are supported by a pro rata tax on the output, by the coal companies direct, and the sons of their employes are entitled to free tuition. Returning after this course, the son becomes his father's helper, and eventually his successor. It was stated on more than one occasion that the *good* miners are so well paid and so satisfied that they stay at home, and that with very few exceptions the foreign labor which comes to the United States is in no sense comparable to the real miner in his home country.

Again, the coal, both in its nature and geological formation, is in many cases so different from ours, as to render a comparison of methods or conditions, impossible or unjust. In Belgium and Northern France especially, seams of coal less than 2 feet in thickness, are mined at great depths, with dips exceeding 60 degrees, and large quantities of gas to contend with.

* Mining Engineer, Morgantown, W. Va.

It is true, that because of the superior personnel of the miners abroad, living conditions among them are materially superior to American mining towns. The miner's houses, notably at Broadsworth, England; Gelsenkirchen, Germany; Bois du Luc, à Trivieres, Belgium; and Lens, France, are neat, clean, comfortable brick and stone houses. There are thrifty gardens, casinos, public playgrounds, schools, churches, and hospitals, for the use of the miners and their families.

To sum up general impressions, England and Wales stand preeminent for underground detail and perfection of construction; Belgium for its hospital and rescue equipment, and the things which tend toward the preservation and comfort of human life; Germany for its magnificently equipped mining plants and power houses, and its wholesale attempts to keep down ankylostomiasis by elaborate baths for the miners emerging from the pit; and France for its *finesse* in laboratory research, chemical and electrical experiments. It should not be inferred from this that any of these countries are lacking in other material matters, for in Germany, at the Shamrock mines, in Herne, is one of the most complete rescue stations in the world, and the mines at Gelsenkirchen have been described for their elaborate equipment; but these conclusions are merely indicative of national type.

Belgium, as a country, occupies a unique position. For its size it is the most densely populated country in Europe (being double that of England), and in no other country in the world is the national wealth so evenly and widely distributed. There are neither millionaires nor paupers in Belgium, living is extremely cheap, and wages higher than anywhere else in Europe.

A movement now very general in France is the mutual aid society, by which schools are conducted, the sick are nursed, land and houses purchased and built by the miner. These societies are on the order of American building and loan associations.

In Germany the miner contributes a small amount per week from his wages, toward a general fund, from which, in case of accident, sickness, or death, he draws benefits augmented by duplicate payments by the company itself.

The English "model village" of Broadsworth, located about 6 miles from Doncaster, is extremely interesting. Here there is the cooperative store, the model tenements, the public lyceum, the athletic field, churches, schools, a hospital, and a fine, manly, intelligent and thrifty lot of miners, interested in self-betterment by means of lecture courses and night schools.

Without fear of contradiction, it may be said that Americans do not know what a dusty mine is, until they have visited the deep, thin-seam mines in Belgium, and Pas-de-Calais district in France. Here the coal is such that the air is continually filled with the dust, and may be likened to a whirling blizzard in which coal dust takes the place of snow, and in which the temperature varies from 85° F. to 113° F. The visitor is invariably supplied with a skin-tight skull cap which fits on under the leather mine hat, to keep the dust out of the hair, and this dust decomposes so rapidly that the clothing worn in the mine must be washed at frequent intervals on account of the fearful odor.

Under these conditions, with thin coal, and the dip so steep that room-and-pillar work or longwall are impossible; where ascensional ventilation from level to level, through long and tortuous workings is necessary, and where the amount of gas exceeds that found in any of our American mines, it is obvious that comparisons are useless.

On the other hand, in some parts of Germany, the mines are developed so nearly like those in America, that only the vast amount of timbering, or the amplified systems of rope or gasoline haulage, excite our interest.

In some parts of England and Wales, the coal is very friable, and in non-coking districts the slack is waste, so that the long-wall system is extensively followed, in order that the roof pressure may be utilized. The miner lays the lumps of coal by hand into his "curling box," a sort of sheet-iron scoop, with side handles, and then carefully empties this into the car, to avoid breakage. The purpose of the scoop-shape is to enable him to use it for throwing the slack back into the gob pile. In such mines as these, the percentage of waste exceeds that in this country.

In the matter of haulage and loading the output, there is no doubt that American mines take the lead, but for the amount of coal which they put out, and with the facilities for handling it, women and girls doing much of the tipple work on the continent, the European equipment is ample.

The average miner is considered to have done well if he sends out 1 ton a day, or about one-third the American average.

By-product coke ovens are practically the only ovens in use. Many plants operate a brick-yard in connection with the mining and by-product industries.

The matter of organization in the executive department abroad is excellent, because the miners have lived all their lives in that one district, speak the same dialect, and have grown up under the system.

Summed up, it would appear that in Europe, the matter of timbering is carried almost to an extreme, while the ventilation and roadways are poor, resulting in reduced output and increased cost of operation. The social conditions are uniformly excellent, and the installation of a mining plant is a matter of permanent investment, to be handed down to posterity. Great attention is paid to the gas, dust, safety lamp, and explosives proposition, and the drilling of corps of men for work with rescue apparatus and in fighting mine fires.

But in America, there are, speaking as a whole, larger mines, enormously increased output, longer haul, both in the mines and to the market; a very complex labor problem, involving wages, languages, social conditions and the general question of efficiency; competition to be met in the selling of the product; industrial conditions to be complied with in its preparation; and greatest of all, a series of new conditions constantly arising, and presenting hitherto unsolved problems, involving the production of a maximum output at a minimum outlay, coupled with the preservation of human life, and the conservation of mineral resources.

It may perhaps have been true, a few years past, that in unwonted development, thirst for mineral commercial supremacy, and lack of research along these lines, the American mines were for a time not so completely safeguarded against loss of life as European, nor was such prevention wholly possible until a sufficient period had elapsed in mining development, to learn the individual equation controlling mines and differentiating them from the older collieries of Europe (and even in the past, while the death rate per thousand men employed, has been high, the rate per ton of coal mined has compared favorably with the world. And it should be borne in mind that there have been *recent* disasters abroad, of considerable magnitude).

With the Testing Station at Pittsburg; with the rescue stations already in existence and those soon to be established; with the individual research now being carried on by large coal companies and mining schools; and with the earnest desire of the American coal operators and the American people to safeguard human life, it may safely be said that American mines compare favorably with any other coal mines in the world.

More than 300,000 families in the United States are today living in homes of their own on lands made productive through irrigation. The 13,000,000 acres under water are yielding up annually harvests valued at more than $250,000,000. The 40,000,000 acres of irrigable land for which there is water available will, when reclaimed, provide homes for 1,000,000 families or 5,000,000 men, women, and children, and, together with the towns and cities developed in connection with their settlement, will support a population of 10,000,000 people.

FIRST-AID CONTESTS

Written for Mines and Minerals

Annual Contests by Teams of the Different Collieries of the Pennsylvania Anthracite Region

Pennsylvania-Hillside Contest.—Probably not since Dr. M. J. Shields, the pioneer, organized the first first-aid mine corps in America, at Jermyn, Pa., in 1899, has such a methodical exhibition of preliminary relief work been given as that seen at Valley View Park, Inkerman, Pa., on September 17.

The Pennsylvania-Hillside contests for the cup donated by W. A. May, General Manager of the Pennsylvania Coal Co. and the Hillside Coal and Iron Co., held in the forenoon were preliminary to the annual National Red Cross contests in the afternoon.

Three interesting events formed the program, and apparently there is as much rivalry among the various colliery teams of this company as between the teams of different coal companies. Seven teams competed in the three events for the May cup, and spice was added to the exhibition because the four best teams in this contest were to represent their company in the afternoon contest of the Red Cross Society. The rules governing this contest were as follows:

First-Aid Men Going to Dinner, Philadelphia and Reading First-Aid Field Day

There will be no time limit allowance for any event, but the team finishing a problem will stand erect in the rear of the subject and the captain will announce "Time"—indicating that the problem is completed. The whole seven teams will perform in each event simultaneously.

The marking of this contest shall be as follows: Each team, operator, and operators shall start in each event with a credit of 100 per cent. and shall be marked with the following discounts for each mistake:

For loose or "Granny" knot	5
For loose bandage	5
For loose splint	5
For wrong artificial respiration	5
For not stopping bleeding	10
For not treating shock	10
For not doing the most important thing first	5
For slowness in work	5
For awkward handling of either patient or stretcher	5
For failure to be antiseptic	5
For captain's lack of command	5

In these events certificates were given to the teams ranking second and third, while the winner captured the cup. The first event was the dressing of a fractured jaw and a bleeding arterial ear wound by one man. William Davis, No. 5 colliery, first added 85 points to his team score in this event. The second event allowed three men to operate on a subject who was supposed to have been injured by a premature blast, with the result that flying coal had fractured his nose and caused profuse hemorrhage. There were also cuts on the head, while the upper right arm and the lower left forearm had simple fractures.

This called for the critical work and the expert bandaging in which George Rickert, William Gilmartin, Ralph Frater, of the Barnum colliery team, excelled, and which added 90 points to their colliery score. Chief interest centered in the third and final event as full teams were involved, the cup was to be won, and winning teams entered in the afternoon Red Cross contest.

The Forest City Hillside team won this event and added 95 points to the colliery total. The teams were made up of the following men:

Avoca, Pennsylvania—Ludwig Weichel, captain; Wm. Creeden, Ed. Warren, Chas. Warren, John McDermott; subject, Wm. Semmens.

No. 14, Pennsylvania—Arthur Brown, captain; John Dick, Thomas Yates, Arthur Wilson, David Jones; subject, Arthur Ramage.

No. 5, Pennsylvania—Peter Hoolihan, captain; Theo. Holland, John Holland, William Davis, Bernard Corcoran; subject, George Hindmarch.

Forest City, Hillside—Thomas C. Harvey, captain; Maurice Johns, Jas. Chidinski, George Parry, Edward Walsh; subject, Alex. Jones.

Dunmore, Pennsylvania—Edwin Shopland, captain; William Miller, William Pratt, William Hill, Patrick Haggerty; subject, William Watson.

Mayfield, Hillside—Bloss Leitinger, captain; James Hanophy, Harry Williams, Joseph Kerins, Vincent O'Neill; subject, Edward Powell.

Barnum, Pennsylvania—R. Eugene Smith, captain; George Rickert, William Gilmartin, Ralph Frater, Joseph Jackson; subject, John Lumley.

The order in which the teams finished and their score was: No. 5 Colliery, 250; Avoca, 240; Forest City and Barnum, 225; Dunmore and Mayfield, 205; No. 14 Pennsylvania, 195. No. 5, Avoca, Forest City, Barnum, and Dunmore, entered in the afternoon events.

The full summary showing the points gained in each event and the final score follows:

	1	2	3	Total
Avoca	75	85	80	240
No. 14	70	60	65	195
Forest City	60	70	95	225
No. 5 Colliery	85	75	95	250
Dunmore	65	65	75	205
Mayfield	55	80	70	205
Barnum	50	90	85	225

Dr. J. B. Mahon, of Pittston, who supervised the morning contests, is in charge of the first-aid work of the Hillside Coal and Iron Co. and Pennsylvania Coal Co. collieries. The judges of the morning's work were Drs. C. H. Lake, of Kingston; J. W. Geist, of Wilkes-Barre; and Thomas Monie, of Archbald. Lunch was served on the ground at noon, during which event acquaintances were renewed.

At the close of the exercises H. M. Wilson, of the Federal Bureau of Mines, lectured on rescue apparatus, after which a practical demonstration was given in an improvised mine.

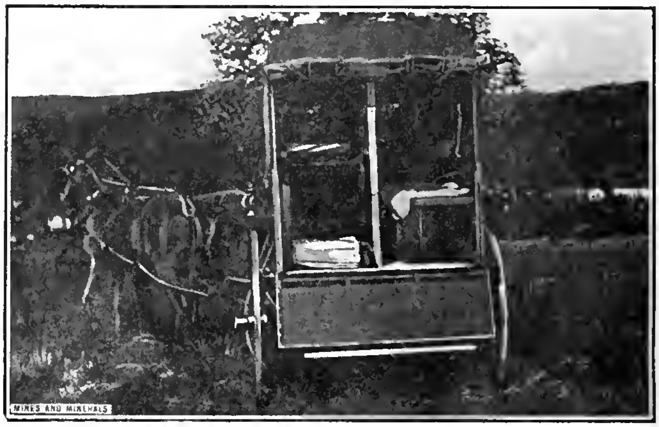
The register of first-aid corps of the company shows that there are 76 corps distributed at the different operations.

The corps from each district competed together and the grounds were arranged so that as many as 10 teams could compete at the same time.

In the district contests, different problems were allotted as follows:

1. Compound fracture of skull and jaw.
2. Compound fracture of thigh. Laceration of hip. Dress with spica.

Philadelphia and Reading Officials Viewing Contest. Beginning at Right-Hand Upper Side of Photo, Dr. Halberstadt, Pres. Geo. F. Baer, Gen. Mgr. Richards



Ambulance, Philadelphia and Reading Company. Will Carry Four Patients

PHILADELPHIA & READING COAL AND IRON CO.

The sixth annual competitive drill of the First-Aid Corps of the Philadelphia & Reading Coal and Iron Co. was held on September 17, 1910, at Lakeside Park, East Mahanoy Junction, Pa. The importance of this work in the estimation of both the management and the employes is shown by the attendance and the interest shown. About 1,600 officers and employes of the company and invited guests were present. General Manager W. J. Richards was in charge and the contest was directed by Dr. G. H. Halberstadt, who is instructor of the corps. The judge of the contests was Dr. J. B. Rogers.

George F. Baer, President of the Philadelphia & Reading Coal and Iron Co., and the Philadelphia & Reading Railway, was present, having, as he said, canceled other engagements to permit his attendance. Other prominent officials and guests were as follows:

Vice-President Theodore Voorhis; General Manager A. T. Dice; Division Superintendents Turk and Keffer; Frank Smink, President of the Reading Iron Co.; Doctor McKee, surgeon for Lehigh and Wilkes-Barre Coal Co.; William Curran, of the Lehigh and Wilkes-Barre Coal Co.; Dr. Joseph A. Holmes, Director of the Bureau of Mines of the United States Government; Robert A. Quinn, General Manager of the Susquehanna Coal Co.; W. A. Phipps, General Manager of the Ashland Coal and Coke Co., of West Virginia; Baird Snyder, Jr., General Manager of the Lehigh Coal and Navigation Co.; T. M. Righter, coal operator, Mount Carmel; Judge O. P. Bechtel, Judge A. L. Shay, Judge C. N. Brumm, Judge MacHenry Wilhelm, County Commissioners Gardner, Crone, and Boyle, and several clergymen, school superintendents, and doctors.

Shamokin District Corps, Subjects Bandaged Ready for Inspection. Philadelphia and Reading Contest

3. Fracture of arm. Laceration of shoulder. Dress shoulder with spica.
4. Fracture of collar bone.
5. Laceration of foot and leg. Dress with spiral reverse.
6. Laceration of left ear with injury to right eye.
7. Rescue from electrical current, and dress the burns of back and shoulder.
8. Crush of pelvis. Laceration of hip. Dress with spica.
9. Handling by one, two, three, and four. Crossing obstruction and artificial respiration.
10. Dislocated hip.

The following teams were winners and entitled to enter in the final competition: Tunnel Ridge, Bear Ridge, Brookside, Knickerbocker (inside), Sterling, Bast (outside), Potts (first lift), Kohinoor (inside), West Shenandoah (outside), Reliance (outside), Wadesville, Eagle Hill, John Veith, and Glendower.

After the completion of the district contests those present, accompanied by three brass bands, marched to a nearby grove where all were guests of the Philadelphia & Reading Co. at a banquet. The large number, over 1,400, were seated without confusion at tables spread with white linen and glittering silver. The menu was as follows: Clam chowder, chicken croquettes, fillet beef, potatoes, peas, cold slaw, potato salad, corned beef, sliced ham, sliced tongue, rasped rolls, chow chow, gherkins, crackers, cheese, ice cream and ices, fancy cakes, coffee, cigars.

In the afternoon the final competitions took place. The problems were as follows:

1. Compound fracture of forearm with laceration of hand and fingers. Dress hand and fingers with spica bandage.

2. Patient caught by belt in engine house. Laceration of head, arm, and leg. Check hemorrhage on the spot and carry patient to hospital, complete dressings prepared to take ambulance.

The winners and the percentages attained were as follows: Wadesville, 100; Glendower, 98; Reliance, 95; Tunnel Ridge, 95; Bast (outside), 94; Bear Ridge, 90; Brookside, 90; Kohinoor (inside), 85; Knickerbocker (inside), 85; Sterling, 85.

After the completion of the final contest the members of all the corps formed in a hollow square and President Baer in a short speech presented the blue pennant and blue badges to the Wadesville team who were the winners.

Dr. J. A. Holmes, Director of the Bureau of Mines, presented a red pennant and badges to the Glendower corps who had the second highest rating, and who were the winners last year. Red pennants and badges were also presented to the other teams mentioned above. After the awarding of the prizes there was an exhibition of the use of Draeger rescue apparatus.

Special trains carried the men to their homes, starting at about 5 P. M.

FIRST-AID CONTEST OF PENNSYLVANIA RAILROAD COLLIERIES

Five hundred members of the First Aid to the Injured Corps of the Susquehanna, Summit branch, Lytle and Mineral coal companies, controlled by Pennsylvania Railroad Co., embracing 13 collieries, employing 20,000 men and boys in the Schuylkill, Shamokin, Lykens, and Luzerne districts, from Nanticoke to Pottsville, participated in the first annual aid to injured contests at Edgewood Park, Shamokin, Pa., September 3, a special train over the Pennsylvania Railroad conveying one-half of the corps from the Nanticoke region. The contests were to have been held in the open field, but rain shifted the exercises to a dance pavilion. Surgeons in charge were Drs. F. L. McKee, Wilkes-Barre; G. M. Stites, Williamstown; J. M. Maurer, Shamokin; J. B. Rodgers, Pottsville. The events were as follows:

Fracture of the lower jaw and a penetrating wound of the right eye

DR. M. J. SHIELDS, ORIGINATOR OF FIRST-AID INSTRUCTION IN COAL MINES OF THE UNITED STATES

Burns of the face and neck.

Fracture of the right collar bone and a compound fracture of the left forearm, with bleeding of bright-red blood in spurts.

An explosion; rescue and carry to dressing room with coat and pole stretcher; examine and find subject unconscious, with a fractured right collar bone and left thigh, also bleeding from a wound on top of the head.

Wound of the lower portion of the left thigh, severe bleeding in jets from the thigh wound; dress injuries, place subject on the hospital stretcher and into ambulance.

The winning teams were as follows:

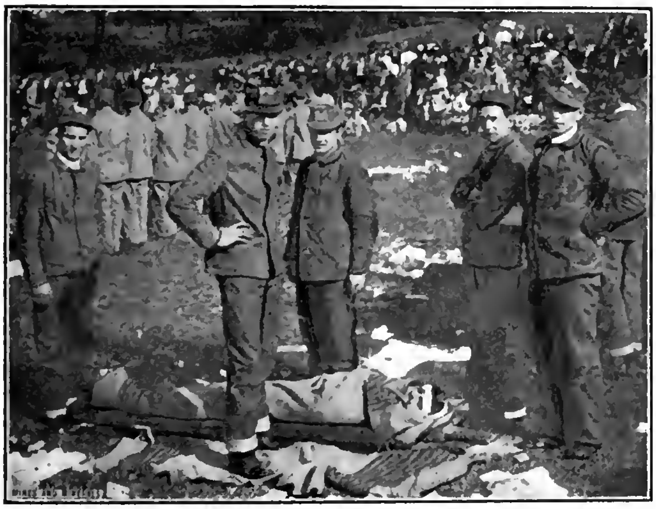
First event, Scott Shaft, Shamokin, William Horn, captain; second, Hickory Ridge, Thomas Howells, captain; third, Nanticoke, No. 7, George McGinnis, captain; fourth, Lykens, A. F. Minnich, captain.

French bronze medals were awarded to the first three, and a silver cup to the fourth. The latter will be subject to annual contest, and must be won three times, to be held permanently.

THE ANTHRACITE RED CROSS CONTEST

In the summer of 1859 M. Henri Dunant, of Switzerland, witnessed the terrible sufferings of the wounded on the battlefield of Solferino. M. Dunant described what he had seen in a pamphlet, which was generally circulated throughout Europe, and aroused such interest that in 1864 a convention of representatives from many of the civilized nations was held in Geneva. From that resulted what is known as the first "Red Cross Treaty." This treaty was subsequently revised at a convention held in Geneva in 1906, at which 43 nations were represented, and provides in time of war for the protection of hospital formations with their personnel of the army, navy, and the recognized official Red Cross societies. Out of compliment to Switzerland, the Swiss flag, with its colors reversed, was adopted to designate these hospital formations and thus the Red Cross on a white ground became the flag of humanity.

The Red Cross societies of the world have a membership of nearly 5,000,000. Japan alone, has a membership of 1,400,000,



FOREST CITY TEAM, WINNERS OF W. A. MAY CUP, PENNSYLVANIA-HILLSIDE CONTEST

WOODWARD COLLIERY TEAM, D. L. & W. WINNERS OF MUCKLE CUP, RED CROSS CONTEST

while the American Red Cross has not 20,000. Since its reorganization, in 1905, the American National Red Cross has rendered active and sympathetic aid, after such serious disasters as the Philippine typhoon, the San Francisco fire and earthquake, the Gulf storm, the Monongah mine explosion, the Mississippi cyclone, the Chelsea fire, the Georgia, South Carolina, and Texas floods, the Michigan forest fires, etc., besides sending aid to the Chinese and Russian famine sufferers, the Kingston, Valparaiso, Calabrian, and Sicilian earthquake unfortunates. It now maintains a corps of physicians who travel around the bituminous coal fields and instruct the miners in the various collieries to apply first aid to the injured. It is well known that many lives and limbs may be saved if the injured are afforded temporary relief immediately or shortly after an accident has occurred, and prior to the arrival of a surgeon. It is with this end in view that first-aid corps have been established in nearly all civilized mining communities.

The anthracite miners have been drilled in first-aid work for a number of years and anually there is a competitive contest under the auspices of the Red Cross Society, for the large silver cup, which was donated as a prize by Mrs. John S. Muckle, of Philadelphia, Pa. The contest was held this year at Valley View Park, Inkerman, Pa., on September 17. While competitive contests are incentives to excel in this special line of work, the fact remains that if any one is trained so as to render efficient aid to the injured, his merit is recognized, and on this occasion Red Cross medals and certificates were awarded to individual winners and a silver cup to the best team. This acknowledgment of proficiency is after all the chief aim, as it is highly probable the holders of the medals can accomplish the same practical results as the winners of the cup.

Whereas, in the anthracite fields, each large coal company is interested in "first-aid work" to the extent that each of the collieries has a corps of trained men it is customary to hold local contests and send the winning teams to the Red Cross contest which is not a local affair but open to all teams.

Considerable rivalry exists between the teams and each watches the moves of the others and comments on the work, down to the outfits. For instance, one team remarked that another team had steel-tipped instead of copper-tipped drills for stretcher bars. In explanation of this, the anthracite miner uses a straight drill which has a cutting edge at one end, while the other end is upset to form a tamping bar. The Pennsylvania Mine Law requires that the tamping end be copper tipped, and the team in question had brought their practice drills instead of their working drills.

The third annual contest of the National Red Cross Society was held at Valley View Park, Inkerman, Pa., on September 17. Twelve teams entered the contest, and to the layman all did equally well, but to the judges, Major Charles Lynch, Captains H. H. Baily and M. A. DeLaney, all United States Army surgeons, there was a difference.

The following teams were entered: Coxe Bros. Derringer colliery; Lackawanna, Woodward; Lehigh Valley, Seneca; Lehigh Valley, Primrose; Lehigh Valley, Wyoming Division team; Temple Iron Co., Lackawanna colliery; Temple Iron Co., Harry E.; Avoca, No. 5, Dunmore, Forest City, and Barnum, of the Erie system.

Previous to the contest, and while the teams were arranging their first-aid kits, Major Lynch handed each team captain a copy of the list of the accidents which were to be demonstrated. The captain then instructed his team mates. At a given signal 12 men were seen trying to lift 12 supposedly injured men and after doing so, carrying them to a place of safety. This is an extremely difficult undertaking, particularly if the injured man is unconscious or unable to aid his assistants, for which reason it is not probable that any one man loaded down with an oxygen helmet can perform this work.

Contest for Muckle Cup, Anthracite Red Cross Contest

Each of the teams treated their subject for the same injury at the same time, and while this kept the three judges moving from group to group, the events were conducted with precision and smoothness.

The first contest was a one-man event. The injured man was lying on an electric wire badly shocked and unconscious. He was to be removed in two minutes and an attempt made to restore him to consciousness. This event was won by the Derringer team of the Lehigh Valley Coal Co. South Pittston district of the Pennsylvania Coal Co. was second, and Seneca colliery of the Lehigh Valley Coal Co. third.

In the second event two men formed the team to treat a man burned about the face. The treatment of a burn of the first degree was applied and the man carried 20 feet to a place of safety. The Harry E. team of the Temple Iron Co. was first, Forest City team of the Hillside Co. second, and Primrose colliery of the Lehigh Valley third.

The third event required three men in the team. The subject had a crushed chest and abdomen with fracture of the lower ribs, due to a heavy weight falling on the body, and was

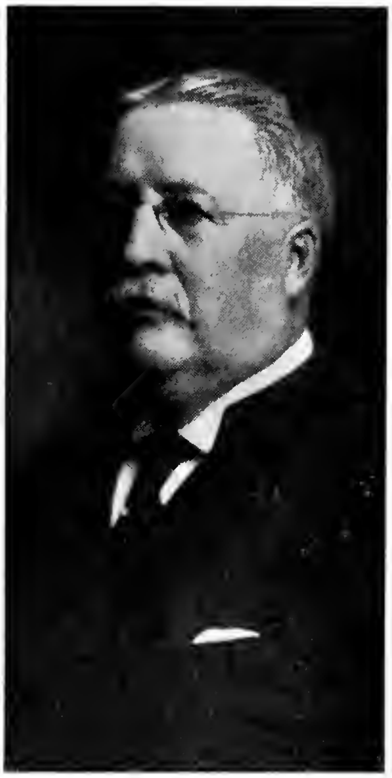
to be carried 30 feet by employing any material at hand for any kind of a stretcher. The Woodward colliery of the Lackawanna Coal Co. was first, Wyoming division of the Lehigh Valley Coal Co. second, and the Avoca team of the Pennsylvania Coal Co. third.

The fourth event called for a full team of five men and was the most interesting and spectacular event of the day. The patient was supposed to have been in an explosion where a fall of rock had followed and he had received a simple fracture of the lower jaw; a severe cut on the palm of the right hand from which blood was spurting in bright-red jets; also a simple fracture of the left thigh high up just below the body. The teams were to dress all wounds and injuries and load the patient on a stretcher. This event was won by the Woodward colliery team of the Lackawanna Coal Co., composed of John Thomas, captain; Neil McCole, John Richards, Benjamin Lewis, David Phillips, and William Martin, subject.

By winning the fourth event the Woodward colliery team of the Lackawanna Coal Co. captured the Muckle cup.

One notable feature at this contest was the large gathering of prominent mine officials, interested friends of the contestants, and people prominent in the coal fields. The audience in number exceeded that of any previous year, which goes to show that the people generally appreciate and approve of this good work.

The first-aid work is being extended to all parts of the country by the American Red Cross Association. Dr. M. J. Shields, who was the originator of the movement in America and is now of the Medical Reserve Corps of the United States Army, is visiting the different coal regions for the purpose of helping to organize the corps under local management, and operators at whose works no such corps are in existence will be afforded valuable assistance by him.



Dr. J. B. Mahon, Instructor in First Aid, Pennsylvania-Hillside Collieries

NEW OHIO MINE LAW

The following notice has been sent out by the State Mining Department of Ohio, to mine operators, miners, oil-well drillers, manufacturers of illuminating oils for use in mines, and all others interested in the operation of mines:

The new mining code passed March 23, by both Houses of the Ohio Legislature, and approved by the signature of Governor Harmon on April 11, 1910, is now in full force.

Many important changes have been made in the old law, and many new provisions, which were necessary to cover new conditions and new dangers.

The duties and obligations of owner, lessee or agent, and all employes in the mines, have been carefully and impartially considered and clearly defined.

The dangers liable to result to life and property in the operation of mines from the hitherto "wild-cat" system of drilling oil and gas wells through workable veins of coal and abandoning them without leaving any record, have been fully provided against. Any person, firm or corporation desiring to commence the drilling of an oil or gas well that will penetrate a vein of coal must first secure permission in writing from the Mining Department, and comply with all the provisions of law pertaining to that work.

The manufacture, sale and use of impure oil for illuminating purposes in the mines is prohibited, and an effective means of detecting it provided.

The sizes of miners' lamps have been regulated, and open torches as stationary lights in mines have been prohibited.

The volume of air has been increased so that better and more powerful artificial ventilation equipments will be necessary in many instances.

Penalties in all cases of violation, and a quick and ready means of prosecution, are also provided. A person cannot longer ignore or violate any provision of the mining law with impunity, and when brought before the lower court waive examination and give bond to appear before the Grand Jury of the county, taking unscrupulous and undue advantage of the interval to prevent the forthcoming of evidence against him. For the first offense his case must be put on record by trial before a Justice of the Peace, while it is new and the evidence obtainable. State and county lines are sometimes convenient landmarks to those who disregard law, and are sometimes used by criminals, as well as witnesses, to defeat the ends of justice.

The new code was the work and recommendations of a Commission composed of three operators, recommended by the operators, and three representative miners, recommended by the miners, and appointed by Governor Harris, and the writer, who was selected by the other six members and also appointed by Governor Harris as the seventh member.

We fully realize that objections may be taken to some of the provisions of the bill, but it could not be expected that an entire code of laws could be drafted covering so many new and varied conditions in mines, that would not have some defects. From time to time, necessary changes can easily be made.

In the consideration of almost every question, the cost of the production of coal, the liberties of the employes, or the increased responsibility of some person, was necessarily affected, necessitating in most instances keen argument, in which the interests of all were zealously guarded and logically protected by their respective representatives. To secure a unanimous and harmonious agreement, and insure a good code of laws, it required reasonable concessions from both sides, which was commendable to both parties, and the fact that Governor Harmon sent the recommendations of the Commission to the Senate and House of Representatives, accompanied by special message of approval, and that both Houses of the Legislature enacted them into law without a single change, or a dissenting voice or vote against them, is the highest compliment that could be paid to the Ohio mine operators and miners.

Sixty days, after the passage and approval of the bill, has been allowed to give time to prepare to meet its requirements. This provision was made so that there could be no misunderstanding on that point, and no room for any excuse of ignorance of the law. All must be prepared to meet with a reasonable compliance of the new law, or the inspectors will be compelled to seek an application of the penalties. This must not be understood to mean, however, that there will be any change of policy on part of the Department, or that the inspectors will pursue any harsh, radical, unreasonable or unfriendly course. It means that the law will be reasonably and judiciously enforced alike to all, and we trust they will have the friendly cooperation of all to that end.

Copies of the entire code can be had by writing the Secretary of State, or George Harrison, Chief Inspector of Mines, Columbus, Ohio.

EXPLOSION AT PALAU No. 2 MINE

Written for Mines and Minerals, by Special Correspondent

Extent and Cause of the Explosion. Rescue Work. Unauthorized Explosives

At about 11 o'clock, Friday night, September 30, an explosion of dust occurred at Palau Mine No. 2 of the Coahuila Coal Co., at Menor, about 20 kilometers (12½ miles) from the mining town of Las Esperanzas, in the State of Coahuila, Mexico. The press dispatches reported that from 150 to 300 men were in the mine; that all of them were killed; and that the mine was the same one in which a similar disaster occurred several months ago. The facts are that there were between 80 and 85 men in the mine at the time of the accident, and this was not the second explosion at this mine. The previous explosion was at Palau No. 3 an entirely different mine,* owned by the same company. It was supposed at first that every man in the mine was killed by the force of the explosion, or was asphyxiated by the afterdamp, but on Tuesday, October 4, four days after the explosion, a group of six Japanese miners were found alive in the sixth south entry, the entrance to which had been closed by an immense fall of rock which prevented the afterdamp from reaching and overcoming them. The latest report places the number of dead at 78. Ten miners belonging to the night shift were saved by the fact that they were in jail for gambling. Had the accident occurred in the daytime the loss of life would have been doubled or trebled.

Fig. 1. Repair Work on Slope, Palau Mine No. 2

Palau No. 2, as shown in Fig. 1, is a slope mine, the slope being driven on the coal which dips about 10 degrees toward the east from the outcrop. The slope has been driven down about 1,100 meters (3,600 feet), with entries to the north and south every 100 meters (325 feet). The side entries extend approximately half a mile to barrier pillars 40 meters (130 feet) in width, separating the workings of mine No. 2 from those of Nos. 1 and 3, to the south and north. As the mines are both gassy and dusty it has been considered advisable to keep the several workings entirely separate, so that a fire or explosion in one mine would not be communicated to the others. Mine No. 2 is the largest of the five mines owned by the Coahuila Coal Co., and is the main source of fuel supply for the National Railways of Mexico. It has a capacity of 1,000 metric tons per day (metric ton 2,204 pounds).

The force of the explosion of September 30 was terrific, blowing down all the timbers in the slope, from the eighth entry to the surface. For 200 feet or more from the mouth of the slope the surface was lifted into the air and falling back crushed the timbers and lagging and entirely closed the mine. Fig. 1 shows the repair work necessary at this slope. Although there were large accumulations of dust in the slope, which was also the intake for the air, there was no evidence of an explosion in the slope above the sixth entry. The dust was, however, forced out of the slope in a great cloud, and after reaching the outside air ignited with a burst of flame that was seen in the town of Muzquiz, 5 miles distant. The fan house was blown down, but fortunately the fan and engine were not seriously injured, and 2 hours after the accident the fan was again in operation, but although entrance was made to the mine through the second opening, about 50 feet to the south of the main entry, it was impossible to penetrate far enough into the mine to rescue the entombed miners, as the ventilating current and air stoppings had been destroyed and the air was short-circuited to the fan. General Manager Ludlow and General Superintendent Jones, summoned from Las Esperanzas by a courier, arrived at 2:30 o'clock in the morning. The first rescue party was organized within 2 hours after the explosion occurred. This party was soon overcome by afterdamp, and a second party sent to relieve the rescuers was also overcome All were brought to the surface, however, and resuscitated after a few hours. It was believed from the first that on account of the destruction of the ventilating currents there was no possibility of any of the men being recovered alive, but the rescuers worked with an earnestness bordering on frenzy to recover the bodies before decomposition should have rendered them unrecognizable. These efforts were rewarded by the rescue of the six Japanese miners 4 days after the explosion. The neighboring mines contributed to the rescue work by sending their superintendents with picked men who rendered invaluable assistance. The company maintains a supply of Draeger helmets and oxygen apparatus and a force of men trained in their use, but owing to the heavy falls and the small spaces left to crawl through, it was simply impossible for the rescue parties to avail themselves of this aid. Fig. 2 shows a rescue party composed of Japanese and Mexican miners taken just before they entered the mine.

On the evening of Sunday, October 2, the rescue parties had penetrated to the eighth lift, and nine bodies, two Mexicans and seven Japanese, which were found in the sixth north entry, were brought to the surface. All had been asphyxiated. Thirty-two more bodies were found on Monday, October 3, by which time decomposition was so far advanced that it was impossible to tell whether the deaths were due to suffocation or to the force of the explosion. At that time it was estimated that it would take at least a month to repair all of the damage to the mine, and it may be that length of time before the last body is recovered.

An instance of oriental peculiarity was evinced during the early part of the rescue work, when General Manager Ludlow, struck by the fervor with which the Japs were removing the debris from the mouth of the slope, suggested paying them by the car of dirt removed instead of by the day. They at once quit work and would not return until he promised not to pay them. They were working for the rescue of their friends or their bodies, and were incensed at having a money consideration placed upon their labor.*

The exact cause of the explosion will probably remain a matter of conjecture. As gas was present in some quantity, particularly in the lower levels, safety lamps (Wolf) were used, and it is against the rules to carry matches into or to smoke in the mines. Evidence seems to point to a windy shot as the initiating cause. All of the coal is "shot from the solid," and paid for on the mine-run basis, and while there are no labor unions, the contracting miner is able to do much as he pleases through the competition for labor in the Coahuila coal-mining district. If he is not allowed to mine his coal according to his own sweet will at one mine he will go to and be welcomed at some other, as the supply of labor is insufficient to produce coal to meet the active demand. Under such conditions it is possible that contractors may permit rules and regulations to be disregarded.

* See Mines and Minerals, Vol. 30, page 462.

* This is not alone an oriental peculiarity. Miners in this country, white or black, will do the same, in fact risk their own lives in the hope of saving their comrades, and without thought of reward.

The contract mining system in Mexico, is somewhat different from that in the United States. A contract miner in Mexico takes the contract for mining so many rooms, or even one or more entries, at so much a ton delivered at the neck of the room or at the parting. The contractor employs and pays the men who actually mine the coal. He will employ as many as 75 or 100 men. The company pays the contractor, and his employes do not appear on the rolls of the company. The company, however, employs a timekeeper who counts the men in the mine after the shift has gone to work. This official was in the Palau Mine at the time of the explosion and was killed. Drivers, car runners, door boys, and other laborers, except miners, are employed directly by the company. The division of authority and responsibility brought about by the contract system is of doubtful wisdom, and operators are as a rule opposed to it, but it is a condition forced upon them by the labor situation in the Republic of Mexico. One of the contractors lost 32 men in the explosion at Palau.

Strong probability attaches to a windy shot as the cause of the explosion by the fact that the miners are in the habit of buying cheap grades of dynamite from the small stores in the town, and smuggling this into the mines instead of buying the safety, or "permissible" explosives which are the only kind provided by the company. Almost prophetic, and to some extent exonerating the management for this explosion, is the following letter of General Manager Ludlow to the agent of the dynamite company. The Coahuila Coal Co. is controlled by the railroad, but the mines are operated under lease by the Mexican Coal and Coke Co., of which Mr. Ludlow is the General Manager. Mr. Ludlow's letter is dated September 28, only 2 days before the accident, and is as follows:

LAS ESPERANZAS, COAH., September 28, 1910

DEAR SIR:—In confirmation of our conversation, I desire to impress upon you the continued risk that we are running at our Palau mines through the use of dynamite against our instructions.

We will not sell dynamite to the miners working in mines where tests have shown that an explosion is liable to be caused by the use of dynamite that would be avoided by the use of safety powders, such as Carbonite and Grisutina.

The miners do not like Grisutina, as they claim it does not do the work, and are therefore buying dynamite outside of our property, from the merchants at the little town of Cuchilla.

These merchants are now doing a large business in this dynamite, and we feel that almost any day we may have an explosion caused by its use that will destroy a great many lives and possibly the property.

I have previously written and asked if you could not arrange with the dynamite company to prohibit the sale of dynamite to miners working in gassy mines.

We care nothing about the profit on the explosive that we sell, but we want to protect the lives of the men and avoid a serious accident.

Both in Europe and the United States, a great deal of attention has been give in the last few years to the use of proper blasting powders, and the laws in some of the states of the union hold the superintendent personally responsible for any powder, except the approved safety powders, being used in his mine.

In this country, with the powder all sold through one company, it would appear to be a matter of very easy regulation if you could impress upon them the danger to the lives and property that is caused by their selling through outside agents to our miners an explosive that is known to be unsafe, for the sale of which a company in the states would be criminally liable.

All the powder that is sold in this town of Cuchilla is sold to miners who can do all the work required of them with safety powders, and it is not necessary that any merchant there should carry any dynamite.

There is one shaft sinker working for us who should be allowed to use dynamite. If the dynamite company does not want us to sell him dynamite, we have no objections to their naming some one else to handle that particular business, but we must insist that the dynamite company assume the full responsibility for all losses of life or damage to the mines that may accrue to their continuing to thwart our endeavors to confine the use, for blasting in our dangerous mines, to the safety powders.

I would appreciate your calling the attention of the powder company to this matter and asking if something cannot be done in regard to it. Very truly yours,

(Signed) EDWIN LUDLOW, General Manager

MINING ENGINEER

The United States Civil Service Commission announces an examination on November 9, 1910, to secure eligibles from which to make certification to fill vacancies as they may occur in the position of mining engineer in the Bureau of Mines, one of the stations of which is located at Pittsburg, Pa., at salaries ranging from $2,400 to $3,600 per annum, and in other branches of the service.

Competitors will not be assembled for any of the tests.

FIG. 2. RESCUE PARTY AT PALAU MINE

The examination will consist of the subjects mentioned below, weighted as indicated: (1) Education and technical training, including postgraduate work, 25; (2) technical and professional experience, 45; (3) publications on mining engineering or allied topics, 30. Total, 100.

It is desired to secure persons having broad training and experience in mining, ore dressing and smelting, and in the treatment and preparation of coal, ores, and other mineral substances. They should be fully qualified to undertake original investigations into the causes of waste in mining mineral products and to suggest remedies therefor.

Applicants must have reached their 21st birthday, but not their 50th birthday, on the date of the examination.

This examination is open to all citizens of the United States who comply with the requirements.

This announcement contains all information which is communicated to applicants regarding the scope of the examination, the vacancy or vacancies to be filled, and the qualifications required.

Applicants should at once apply to the United States Civil Service Commission, Washington, D. C., for Form 304 and special form. No application will be accepted unless properly executed and filed with the Commission at Washington prior to the hour of closing business on November 9, 1910. In applying for this examination the exact title as given at the head of this announcement should be used in the application.

KELLERMAN MINE, KELLERMAN, ALA.

*Written for Mines and Minerals, by Neill Hutchins**

The Kellerman Mine is owned and operated by the Central Iron and Coal Co., of New York, with general offices at Holt, Ala., at which point the company operates one modern blast furnace, a soil-pipe foundry, nodulizing plant, 164 beehive coke ovens, and 45 retort, or by-product, ovens. Holt is situated on the east bank of the Warrior River, 5 miles north of Tuscaloosa; Kellerman is 17 miles north of Holt. Both places are connected with the Mobile & Ohio Railroad, and the Alabama Great Southern Railroad, at Tuscaloosa, by the Warrior Southern Railroad, a branch line of the Mobile & Ohio Railroad, extending from Tuscaloosa to Kellerman. The Kellerman Mine is in the foot-hills east of the Warrior River, where the company has 5,000 acres of coal lands, covered with virgin timber, well drained and easy of access to the coal markets and manufacturing centers of Alabama. Five miles from the western boundary of the property is the Warrior River. This stream, as the result of extensive government improvements in the nature of magnificent locks and dams, will, in connection with the Tombigbee and Alabama rivers, soon be navigable all the year around from a point north of Kellerman to the Gulf of Mexico, at Mobile, Ala. Lock No. 16, west of Kellerman, is in course of construction, and with its completion, and that of locks numbered 17, 18, 19, and 20, the stream will be navigable up its two forks—the Sipsey and Locust —to the extensive Warrior coal basin, at a point within 12 or 15 miles of Birmingham. The Central Iron and Coal Co. is the only company that has so far sent coal into Mobile via the all-water route. This was done last year, the coal being transferred direct from river barges on to vessels in Mobile Bay. The Kellerman Mine is at the terminus of the Warrior Southern Railroad, originally constructed for the express purpose of handling the raw material and finished products of the Central Iron and Coal Co. This railroad from Tuscaloosa to Kellerman runs through virgin forest, numerous tunnels, high trestles, etc., which add to the picturesqueness of the route. The road has a 3-per-cent. down grade all the way from Kellerman to the manufacturing plants at Holt.

Location. Transportation Facilities. Method of Handling and Preparing the Coal

Fig. 1. Entrance to Kellerman Mine

The tract of 5,000 acres, in which the Kellerman Mine is as yet the only opening, contains the Milldale and Brookwood coal seams, which in this territory run together, the Milldale, or bottom, seam being 20 inches high, and the Brookwood 60 inches high. These beds of coal are above the valleys draining this region, thus making a drift mine easy of operation. Test borings have located the Pratt coal seam 600 feet below the Milldale and Brookwood beds, where it shows a thickness of 42 inches. Assuming that it will take years to exhaust the Milldale and Brookwood seams, the fact that the Pratt seam is in easy striking distance enhances the value of the property. As previously stated, the coal outcrops on the hills and ridges above the draining valleys. From the mouth of the mine, shown in Fig. 1, the coal rises gradually until the center of the field is reached. This natural condition makes the drainage of the mine a simple proposition of cutting sufficient drainage ditches to the outcrop at the different points, so that the water may flow by gravity. This method of unwatering eliminates the expense of pumping which is so often a serious factor in coal-mine operations. Only five or six small Cameron pumps are required for raising water from swamps here and there in the mine to the drainage ditches. All of the pumps are operated by compressed air supplied from the compressor plant.

Tipple, Etc.—The coal-mine tipple is situated about 600 feet from the mouth of the mine. It is a frame structure about 210 feet long, equipped with Phillips cross-over dump, Fairbanks scales, and slate-picking belt. The coal is delivered from the tipple to a $36'' \times 42''$ roll crusher operated by $11'' \times 14''$ twin engines. After leaving the crusher the coal is carried by conveyers to the washer plant, shown in Fig. 2, alongside the tipple. After going through the washer the coal is passed through a large revolving screen with $\frac{3}{4}$-inch holes. The fine coal passing through this screen goes into the washed slack coal bin; the rest of the product, being a fine nut coal, is desirable for steam purposes. By this practice about 80 per cent. of the product goes to the washed slack coal bin, and the remainder, or nut coal, goes either to the washer boilers for fuel by means of conveyer, or to nut coal bin for loading into railroad cars for the market.

Washer.—The washer is of the Stewart type with four jigs, having an average daily capacity of 1,100 tons of coal, and is operated by twin engines with $14'' \times 16''$ cylinders. The engines also operate the dry and wet coal elevators. It is proposed to add two additional jigs to the washer and thus increase the output about 500 tons. The refuse from the washer is raised by conveyers to the top of the washer building and passes off to the dump 300 feet distant through an 18-inch cast-iron split pipe, which is made in 5-foot sections and elevated on the trestle shown in Fig. 2.

A stream of water passing through this pipe keeps the refuse in motion. The coal from the washer is carried by conveyer to a 250-ton coal bin and loaded from the bin into railroad cars. All of the washed product of the Kellerman Mine is shipped to the manufacturing plants of the company at Holt.

The steam plant consists of five 80-horsepower and one 150-horsepower horizontal boilers. Nut coal is used for fuel. This has been found satisfactory and as compared with run-of-mine coal has increased the steaming efficiency of the plant. By using nut coal only four of the boilers need be operated to furnish all the steam required for the compressor plant, washer, crusher, etc.

The haulage equipment of the Kellerman Mine is considered to be among the best in the state. The main entry follows a barely perceptible ascent until the center of operations, nearly a mile distant, is reached. The main entry is partly double tracked with 40-pound rails. There are 300 two-ton mine cars, 32 inches high, 44-inch gauge, and equipped with Lobdell wheels and axles. The loaded cars are gathered from the various cross-entries by mules and delivered on the main entry to compressed-air locomotives of H. K. Porter Co. make with cylinders $9\frac{1}{2}$ in. $\times$ 14 in. Sixty mine cars are delivered to each locomotive, making a trip of 120 tons of coal which each

* Ensley, Ala.

locomotive delivers from the gathering point to the tipple. Two 15-ton compressed-air locomotives of the size mentioned are used on the main haulage, while one 10-ton air locomotive of H. K. Porter make, with 7"×14" cylinders, is used for gathering purposes at some points in the mine. The haulage equipment, as well as tracks, is an attractive feature of the mine.

The retreating method of room and pillar mining is employed. Rooms are driven 22 feet wide, and as fast as the entries are driven to the outcrop pillars are pulled. In this manner it is conservatively estimated that 90 per cent. of the developed coal is recovered. This is a feature of the mine peculiarly worthy of mention, inasmuch as very few coal mines afford opportunity for recovering this percentage of developed coal. The advanced workings are all in an area of not over 1 mile from the end of main entry inside, and on account of the natural drift of the bed, the coal after being loaded in cars is very easily gathered. All the product is pick mined. The roof is of good sound rock and accidents from rock falls are infrequent. The height of the bed makes it desirable coal to work, and labor is usually to be had in abundance. The contract system of labor is employed to a considerable extent, laborers being employed by and working for contractors.

The mine is ventilated by three air-shafts driven from the surface down to the coal. The principal intake is the main entry, although the various drainage ditches mentioned serve as intakes. The air circulates freely throughout the mine, and by means of a small furnace at each air-shaft is conveyed out of the mine. The ventilation being almost a natural proposition, the mine is remarkably free from gas. This feature has often been the subject of favorable comment by the state mine inspectors.

The average daily output of the mine is about 1,200 tons, although it has reached 1,600 tons from this one opening in one day. The large output will probably be maintained as soon as additional coal-washing facilities have been provided. The mine employs about 180 miners and 120 laborers both inside and outside of the mine.

The compressed-air plant is on the opposite side of the tipple from the washer, and a short distance from the steam plant. It is housed in a substantial brick structure 44 ft.×80 ft., with hip roof, covered with galvanized iron, and has a concrete floor, making it practically fireproof. This structure contains

Fig. 2. Steam Plant, Tipple and Washery, Kellerman Mine

three Ingersoll-Rand, high-pressure, three-stage compressors, each 18 in.×13 in.×9 in.×3¾ in.×24 in. A 4-inch double extra-heavy pipe line conveys the air from the compressor to the end of the main slope in the mine. A number of stations are provided at convenient points in the mine for charging the locomotives. An average pressure of from 800 to 900 pounds is maintained on this line. The compressor house also contains a small electric generator, which furnishes electric lights around tipple, washer, the company's office, store, and superintendent's residence.

All water necessary for washing is secured from Davis Creek, at a point two miles north of Kellerman, where the company's pumping station is located. One 80-horsepower and two 60-horsepower boilers more than supply steam for the large pump, which is an 18"×27½"×10½" compound duplex pot-

Fig. 3. Commissary, Kellerman Mine

valve type. A large special Cameron pump is also maintained at this point as a spare. The pipe line from pumping station to reservoir is 1½ miles long, and is built of 8-inch pipe. Owing to elevation of the reservoir above the pumping station, a working pressure of 300 pounds per square inch at the pumps is necessary. The pipe line, with its universal, no-lead joints, has given satisfaction on this high-working pressure. The reservoir is located on top of a hill overlooking, and about half a mile from, the washing plant. It is cement lined, with natural rock bottom, and has a capacity of 1,000,000 gallons. The elevation is sufficient for a good pressure at the washer. Two 5-inch pipe lines deliver water by gravity from the reservoir to the washer.

The company has a laboratory and a chemist, who makes frequent daily analyses of washed coal, as well as refuse, and keeps the washer foreman and superintendent fully informed as to results being obtained.

Two lines of standard fire-hose with nozzles are connected up at all times with the line from the reservoir for use in case of fire at the tipple or washer. Casks filled with water and containing fire-buckets are also conveniently located. A night watchman is employed who carries an approved watchman's clock.

All dangerous points where sprocket wheels, etc., operate about plant are protected with railings or other safety devices. The company, in keeping with other large operators in this district, is giving this matter considerable attention now.

The town is well arranged and contains 195 tenement houses, 85 of which are for white, and 110 for negro employes, all on high ground, with special view to natural drainage. The negro houses are on the hill east of the mine, and are separated from white quarters by a valley. The white houses are convenient to and just west of the mine. Most of the white tenements are attractively painted and present a good appearance. In many respects it is a model camp. The management has always been particular to have it so. Visitors frequently comment favorably on the general neat, attractive appearance of not only the camp, but the plant, roadways, and structures about the mine. Among other structures there is a stable with a stable lot for the company's 33 large mules, probably the finest mine stock in the state. Near the mine entrance is a combination roundhouse for air locomotives and blacksmith shop. The commissary, shown in Fig. 3, is attractive and well stocked with general merchandise. The local mine office is

near the store in which rooms are provided for the superintendent, resident mining engineer, timekeepers, etc. Both white and negro employes have their separate schools and churches, erected with assistance of company, on lands donated for that purpose. Eight-months public schools are maintained.

The mine was opened in April, 1901, and was named for C. R. Kellerman, the first chief engineer of the company. During the life of the mine it has had only two superintendents, Mr. C. E. Crandall, who served the company in that capacity until recently, and Mr. S. J. Routledge, the present superintendent, who comes from the Tennessee Coal, Iron and Railroad Co., after long experience in same capacity in the Pratt field. Only three different mine foremen have been employed. The present foreman, Mr. Geo. Burgess, is an ex-employe of the Tennessee Coal, Iron and Railroad Co., with years of experience in the Blue Creek fields of that company. Mr. G. H. Howell is resident engineer; Mr. C. M. Ayers, Holt, Ala., is chief engineer.

As an index to the character of the mine and its present condition the following extract is quoted from State Mine Inspector's report of July 19, 1910:

"Ventilation in good shape. Mine naturally damp in character. General conditions way above the average."

EXPLOSIVES

Investigations have stimulated the powder manufacturers to devise comparatively safe explosives and have resulted in the submission, for investigation by the government at the Pittsburg testing station, of nearly 100 different explosives, of which 45 have now passed the government tests and have been listed as "permissible."

On May 15, 1909, the Geological Survey issued Explosives Circular No. 2, giving lists of permissible explosives. On May 16, 1910, a third list was issued, including all permissible explosives tested up to that date. This list shows that 14 additional explosives have been placed in the permissible class since the issue of Circular No. 2. These are as follows:

Ætna coal powder D, Ætna Powder Co., Chicago, Ill.

Coal special No. 3-B and Coal special No. 3-C, Keystone Powder Co., Emporium, Pa.

Eureka No. 2-L. F., Tunnelite No. 6-L. F., and Tunnelite No. 8-L. F., G. R. McAbee Powder and Oil Co., Pittsburg, Pa.

Titanite No. 3 P, Waclark Titanite Explosive Co., Corry, Pa.

Trojan coal powder A Trojan Powder Co., Allentown, Pa.

Detonite special, Detonite Co., Cincinnati, Ohio.

Monobel No. 2, Monobel No. 3, Carbonite No. 4, and Hecla No. 2. E. I. du Pont de Nemours Powder Co., Wilmington, Del.

Kanite A, W. H. Blumenstein Chemical Works, Pottsville, Pa.

A COAL-LOADING MACHINE

Written for Mines and Minerals, by William Whaley

Apparatus for Use in the Mine to Load Coal into Mine Cars. Loads 1¼-Ton Car in 3 Minutes

For several years the engineering firm of Meyers & Whaley, of Knoxville, Tenn., has been engaged in the development of a shoveling machine that would load coal from the face of a room or entry, or rock from a tunnel face. The first machines built were tried at the mines of the Wind Rock Coal and Coke Co., Wind Rock, Tenn., of which Mr. C. H. Thompson is general manager.

The first machine built was a crude experimental affair, merely intended to ascertain if the peculiar motion given the scoop blade would actually shovel coal from the ground and deliver it to a conveyer. The machine was built on a wooden frame mounted on ordinary mine-car trucks. It had a rubber-belt conveyer to take the material delivered by the shovel back to the car and a motor with necessary gearing to drive the conveyer and the crank-shaft of the shoveling device. This machine loaded several cars of coal and proved by its action that the shovel principle was correct. It was then abandoned and a complete machine designed and built.

Fig. 1. Coal-Loading Machine. Shovel Moving into Coal

Fig. 2. Coal-Loading Machine Loading into Conveyer

This second machine was put to work in the Wind Rock Mine and thoroughly tested at different times extending over a period of nearly 3 years. During this time the rear conveyer was changed to suit larger cars. The machine did excellent work, loading 1¼-ton cars in about 2 to 3 minutes. However, the Wind Rock Co. was unable to keep the machine supplied with cars and the handling of a large number of cars became a serious matter. Owing to these difficulties the Wind Rock machine loaded only 60 to 70 tons per day, whereas it had the capacity and ability to load 100 to 125 tons if the car supply had been properly regulated. Owing to these circumstances the owners of the machine discontinued the experiments at Wind Rock and built another and heavier machine for a somewhat larger seam of coal. In Fig. 1 the machine is shown pushing the shovel scoop into the broken coal at the room face, and in Fig. 2 the scoop is shown subsequent to discharging its load on the traveling belt, and prior to its return to the coal pile. The experimental runs with the Wind Rock machine having furnished data for other machines it was withdrawn from active service, although it was in good condition and at present is on exhibition and running in the main building of the Appalachian Exposition, at Knoxville.

The next machine was designed and built for a 5-foot seam of coal at Middlesboro, Ky. Upon completion of the machine and before installing, it was found that the mine entries and

wires were too low to allow its passage, and as the United States Coal and Oil Co., at Holden, W. Va., had asked for this machine, it was decided to put it in their mines. This company has one of the finest plants in West Virginia and every facility and convenience for handling and caring for mechanical equipment. It also has a very fine seam of coal averaging 6½ feet and not varying more than a few inches from this at any place. The floor is exceedingly hard and smooth, the roof is perhaps the best of any seam in this country. Rooms are driven 36 feet wide and absolutely no props are used in the mines. The company uses large low cars well adapted for loading with the machine.

The coal at Holden is a peculiarly, hard, tough bituminous coal, known as No. 2 gaseous. Its coherent qualities make it somewhat difficult to shoot down and nearly as hard to shovel as large lumps of limestone. However, in spite of the physical

The territory in which the machine worked consisted of seven rooms recently turned off the airway in No. 5 mine, a sketch of which is given in Fig. 4. Much of the work was done on curves. The rooms ranged in width from 21 feet to 27 feet and all but two had two tracks. The rooms were on the butt end of the coal which greatly increased the difficulty of shooting coal down.

The company had no difficulty whatever in keeping the machine supplied with cars, which were hauled to and from the side track on the entry by a mule.

The crew of the machine consisted of four men, as follows: One machine runner, one man in front, and two men to handle cars and pick slate.

The United States Coal and Oil Co. was much pleased with the results obtained. The machine used at the Holden Mine is shown in plan and elevation in Fig. 3.

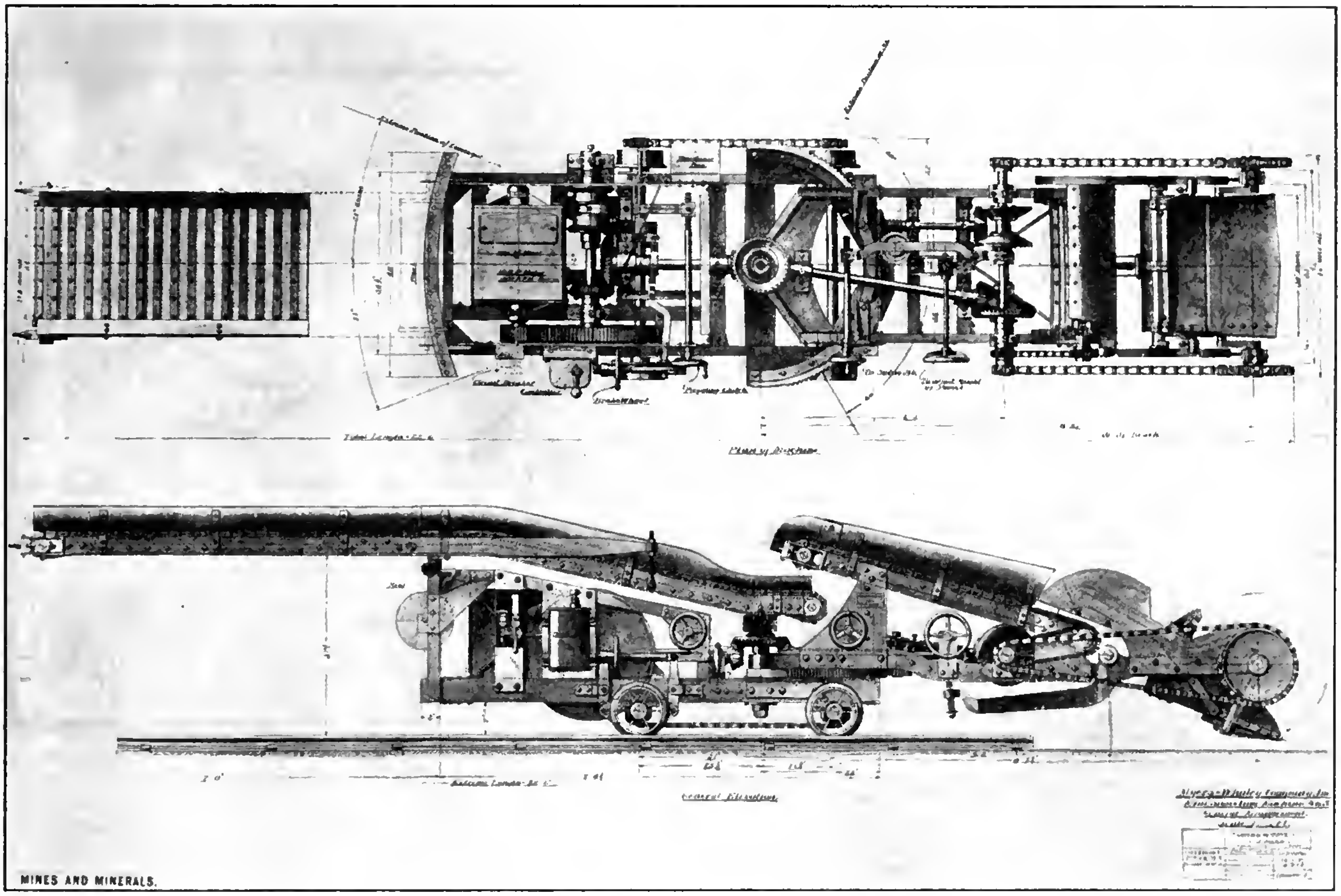


FIG. 3. PLAN AND ELEVATION OF COAL-LOADING MACHINE

condition of the coal the machine was very successful in handling it.

During six days of its operation a record was kept of all features of the run, cars loaded, time required to load and shift, time to change the machine and time lost for any cause. During the six days the machine loaded 768 tons of coal, the time of loading and shifting cars being 36 hours, 6 minutes; time changing the machine 4 hours, 21 minutes; time lost, 14 hours. 33 minutes; total time, 55 hours. The machine loaded out four rooms per day and part of the lost time was due to delay in shot firing, in waiting on smoke to clear out of the room, and other items not chargeable to the machine. A copy of daily report of the last day of this test run is given in Table 1.

In spite of the delays and lost time the machine averaged 128 tons per day. The lowest day's work (due to lack of coal) was 90 tons, and the highest 150 tons. The average time of loading and shifting a car was 8.4 minutes, or 21.3 tons per hour.

The front part or jib of the machine on which the shovel is mounted swings laterally as directed by the operator covering a space 19 feet wide. The yoke supporting the shovel is provided with a vertical adjusting device, operated by a hand wheel, to adapt the shovel to work above the track or to shovel over any irregularities in the bottom. The shovel can be raised to clear the track or lowered to shovel 8 inches below the top of the rail. As the shovel scoop is 30 inches wide it receives lumps up to this size. The rear conveyer can be adjusted vertically, and shifted laterally so that cars can be loaded on sharp curves or even on a track parallel to the loading track. This conveyer is a moving apron 24 inches wide and forms an excellent table from which to pick slate before it goes into the car. The machine is provided with propelling gear and is controlled by hand wheels located on one side and convenient to the operator.

The plan of working face in a Holden Mine room, shown in

Fig. 5, also shows the method of loading out a room from two tracks by means of the machine.

This is the first description of the machine given to the public; up to this time the machine has been carefully developed and tried out and it is no longer an experiment.

There are numerous other applications of the machine which have been considered; for example, as a traction shoveling

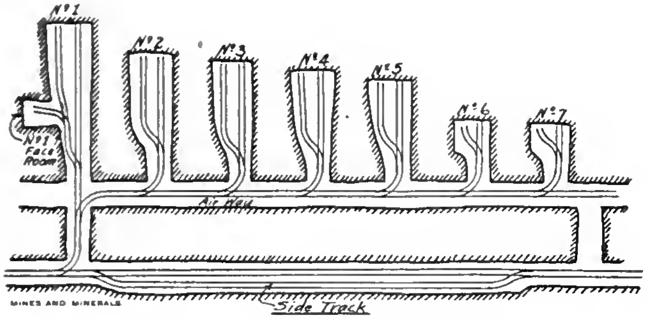


Fig. 4. Mine Plan for Machine Loader

device for stock yards and similar work, to load wagons or railroad cars. A machine has also been designed especially to work on coke-oven wharves to load coke into railroad cars. The machine is adapted to loading broken rock from tunnel faces, although such machines must be specially designed for the purpose.

Table 1. Loader's Daily Report

Holden, W. Va., August 31, 1910

Sixth day of official test run.

Time Started 7:30 A. M.	Time Started 12:33 P. M.	No. of Men Working:
Time Stopped 12:03 A. M.	Time Stopped 5:32 P. M.	A. M. 4
Total Time A. M. 4 hours 33 minutes	Total Time P. M. 4 hours 59 minutes	P. M. 4

Working Place	Time Loading and Shifting Cars	Time Changing Machine	Time Lost	Total Time Consumed	Cars Loaded
No. 6 room......	1 h.	3 m.	1 h. 1 m.	2 h. 4 m.	24 tons
No. 7 room......	2 h. 29 m.	25 m.	23 m.	3 h. 17 m.	54 tons
No. 1 face room (neck)........	1 h. 45 m.	24 m.	26 m.	2 h. 35 m.	33 tons
No. 1 room......	1 h. 25 m.	11 m.	none	1 h. 36 m.	39 tons
Totals......	6 h. 39 m.	1 h. 3 m.	1 h. 50 m.	9 h. 32 m.	150 tons

Remarks:—Time lost No. 6 room — loose setscrew on conveyer sprocket tightened between 7:30 and 8:15 = 45 minutes. Off track 7 minutes; pulling down coal 9 minutes. Total, 1 hour 1 minute.

Time lost No. 7 room—waiting on driver 1 minute; off track 5 minutes; pulling down coal, 17 minutes. Total, 23 minutes.

Time lost No. 1 face room neck—off track, 9 minutes; pulling down coal 17 minutes. Total, 26 minutes.

Time lost No. 1 room—none.

J. B. Haile, Machine Runner.

Report by W. Whaley

TRADE NOTICES

The Denver office of the Jeffrey Mfg. Co. has been removed from 1711 Tremont Place to the First National Bank Building.

The Pittsburg Testing Laboratory has moved its New York office to 50 Church Street, Hudson Terminal Building.

The Ohio Brass Co., Mansfield, Ohio, has issued a New Mine Catalog, No. 9, which is a valuable reference book for mine managers, superintendents, and mine electricians.

The Carnegie Steel Co., of Pittsburg, Pa., has issued a pamphlet on Vanadium Steel which will be found interesting and useful by all steel workers.

The Minneapolis Steel and Machinery Co., of Minneapolis, Minn., have been designing, manufacturing, and erecting mining, milling, and smelting structures for some years. Some of the buildings they have designed have been photographed and incorporated in a neat pamphlet for distribution. This company's structures are all steel.

Owing to the great increase in business in the vicinity of Atlanta, Ga., and Rochester, N. Y., the H. W. Johns-Manville Co. has recently opened a new office in each of these cities. The Atlanta office is located in the Empire Building, in charge of Mr. W. F. Johns, who has been traveling this territory for the company for a number of years, and the Rochester office is located at 725 Chamber of Commerce, in charge of Mr. H. P. Domine, formerly with the Buffalo branch of the company.

The Stromberg-Carlson Telephone Mfg. Co. has had its factory buildings at Rochester, N. Y., photographed from a height of 65 feet. This is in the form of a souvenir postal card, which shows the immense area covered, together with the system of lighting and ventilation.

On September 1, the General Chemical Company of California, succeeded to the business and connections formerly enjoyed by the Peyton Chemical Co. Unfinished transactions, as well as future business, will have the attention of the new company. All

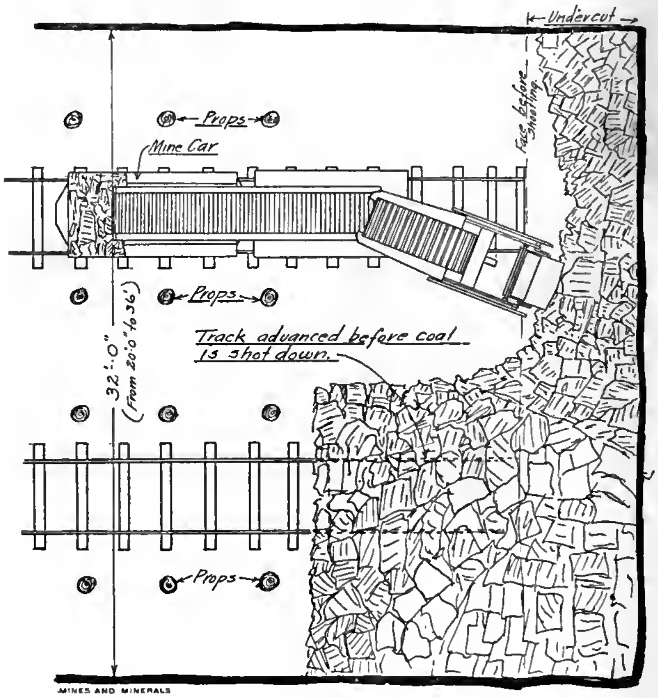


Fig. 5. Working Face in Room, Holden Mine

communications should be addressed to General Chemical Company of California, Royal Insurance Building, San Francisco.

The Ball Engine Co., of Erie, Pa., has recently installed high-class engines at plants of the following coal companies: Bailey Wood Coal Co., Wood Bay, W. Va.; Rosiclaire Lead and Fluorspar Mines, Shawneetown, Ill.; Sycamore Coal and Coke Co., Williamson, W. Va.; West Kentucky Coal Co., Wheatcroft, Ky.; Webb Coal Mfg. Co., Rayford, W. Va.; Barnes & Tucker Co., Barnesboro, Pa.; Gulf Smokeless Coal Co., Tams, W. Va.; Big Sandy Coal and Coke Co., Marytown, W. Va.; Pittsburg Coal Co., Cecil, Pa., and Burdine, Pa.; By-Products Coal Corporation, Solvay, Ill.; Pittsburg-Belmont Coal Co., Neffs, Ohio.

A feature of the Beaver die stock, in addition to not having to change dies, is the universal chuck which centers and grips all sizes of pipe by shifting a cam. It is complete within itself, no loose parts to change or get loose. Since only one set of dies is used only one set must be purchased when dull, and the usual expense from broken grip screws and lost bushings is eliminated, also the lead screw is a small separate piece for cheapness in repairing in case of wear. It is made by The Borden Co., Warren, O., whose No. 11 catalog, showing a complete line of die stocks, may be had on request.

SAMPLING COAL AND COKE*

By E. G. Bailey, M. E.†

(Continued from October)

Sampling Coke.—The same general method can be used in sampling coke, whether at beehive ovens, by-product ovens, gas plants, blast furnaces, or elsewhere. Coke is much more homogeneous than is the average coal and a much smaller original sample than would be required from coal of the same size is usually ample. As a rule, coke is made from slack or crushed coal, and the present tendency in its manufacture is to pulverize all coal or slack to half-inch or finer in order to make a more homogeneous and stronger coke. The handling and mixing of the coal in the bins, larries, and ovens also minimizes errors arising from variation in the quality of coal from different parts of the mine. At some by-product coke plants, where coal from different mines and mining regions is mixed, the uniformity of coke depends upon the thoroughness with which the coals are mixed.

Taking Samples from Vessel Cargoes, Mechanical Conveyers, Carts, Stock Piles, Bins, Etc.

No guide, corresponding to the size and quantity of impurities in coal, can be adopted in determining the size of sample of coke which should be taken to secure any desired degree of accuracy, but practice has proven a 100-pound original sample to be ample in most cases. If extremely coarse coal containing a high percentage of impurities has been used to make the coke, a larger sample should be taken. A safe guide is to take the same weight of coke for a sample as would be required to be taken from the coal of which the coke was made in order to obtain the same accuracy.

While different pieces of coke vary in their appearance, some being very bright and silvery, others dark and discolored, the physical appearance is no indication of the percentage of ash, sulphur, or phosphorus present, hence it is difficult, if not impossible, for one to select either a good or poor sample, so far as chemical analysis is concerned. It is better to select the longest and most slender pieces in taking a sample, in order that the coke from the top to bottom of the charge may be represented, and selecting the more slender pieces enables a greater number of pieces to be taken from more different parts of the ovens, or shipment, for the same total weight of sample. If the coke contains much large slate and bone, these impurities may predominate in the shorter, thicker pieces of coke, in which case the larger sample should include pieces of the various sizes and shapes. In general, samples taken from the coke yard should consist of long slender pieces, selected proportionally from the different ovens. Samples of coke from railroad cars should be taken while the car is being unloaded if possible, but the chance for error in sampling from the top of the car is much less than in sampling coal in this manner.

There is one point of particular importance which should be remembered, especially when crushing the coke by machine or pulverizing it in the laboratory—this is the abrasion of iron or other material by the coke and consequent contamination of the sample with such material. Coke is extremely hard and readily cuts iron or even hardened steel, and errors arising from this cause have been known to increase the percentage of the ash.

Sampling Ashes.—Samples of ashes and refuse from various kinds of furnaces are frequently taken for the purpose of determining the extent of loss in the form of combustible matter. It is very important that such a sample should contain a proportional part of all refuse whether taken from the pit or drawn from the grates when the fire is cleaned. Two or more shovelfuls should be taken from each wheelbarrow or car of ashes taken from the furnaces during at least one cleaning period, or the period covering the total time between fire cleanings.

If the percentage of combustible contained in the refuse alone is desired, no attention need be given to the moisture in the ashes while taking and reducing the sample, as the analysis of this material should be on the dry basis; however, if the ashes are being weighed, a special moisture sample should be taken and the weights corrected to the dry basis. When mixing and breaking samples of dry refuse to be used for the combustible determination, it is better to dampen them to prevent loss of the fine dusty ash, as well as for the convenience of the sampler.

It is often desirable to sample separately the refuse from the ash pit and that removed from the grate when cleaning fires, especially in boiler tests used for comparing the efficiency of different types of grates and stokers, or the sifting tendency of different kinds of coal. The large clinkers, which are generally present, necessitate a larger sample being taken for the sake of accuracy, for it is evident that an extra 10-pound clinker in a 100-pound sample would cause an unallowable error. The necessity for taking a large sample and the possible error may both be minimized by picking out all clinkers larger than 1 inch, and weighing them separately from the ashes, and recalculating the analysis to include this percentage of clinker as ash. The additional information gained by determining the amount of actual clinker is well worth the trouble of picking them out.

The quantity of sample taken from ashes and refuse, free from large clinkers need not exceed 200 pounds, but with large clinkers present the sample should be two or three times this size.

Shipment of Samples.—Samples sent to the laboratory should consist of 3 pounds or more of 4-mesh or finer. The most convenient container is a $500 coin sack, for all but the special moisture samples. The latter, which have been described, should be shipped in hermetically sealed tin or glass jars, preferably the ones in which they were accumulated, unless proper facilities are available to crush and mix them. If the 2 quarts or more of this special moisture sample can be rapidly crushed in a bone mill or similar equipment and returned directly to the original jar, it can then be thoroughly mixed by shaking, and a few ounces put into a small can or jar which can be hermetically sealed and placed in the coin sack with the other sample taken from the same material. If glass jars or bottles are used for this purpose, they should be enclosed in a mailing case or similar precautions taken to prevent the breakage of the bottle in shipment.

All samples should be plainly marked, giving the name or number of the sample, date, and by whom it was sent.

Reduction of Samples in the Laboratory.—No matter how much care is used in taking a representative sample and reducing it to a few pounds of 4-mesh size, it may still be made valueless through improper methods of further reducing and pulverizing it in the laboratory. It is necessary to maintain a definite relation between the size of the coal or coke and the quantity of sample to be divided whether the sample weighs 1 ton or 1 gram. The prevailing tendency in many laboratories is to reduce the sample to a smaller quantity than it should be for a given size, which may cause an error of several per cent.

The variation between the results obtained in different laboratories is more often traceable to error in reducing the samples than to any other one cause. Whenever a sample is to be divided and sent to different laboratories to check the accuracy of an analysis, it should always be pulverized to 60-mesh or finer and thoroughly mixed before dividing, in order to eliminate a chance for inaccurate reduction of the sample. As a guide in reducing a sample the following table should be used:

Size of Coal	Sample Should Not Be Divided to Less Than
4 mesh	1,100 grams
8 mesh	120 grams
10 mesh	55 grams
20 mesh	3 grams

*See also article Coal and Coke Sampling in MINES AND MINERALS for September, 1910.

† Fuel Testing Co., 220 Devonshire Street, Boston, Mass.

DATA OF PETRODYNAMICS

Written for Mines and Minerals, by R. Dawson Norris Hall

The Mode of Action of Shear Showing How It Results in the Breaking of Rock Strata

Shear is the sliding of one part of a body past another part, and the strain caused by the shearing stress is resisted by the shearing strength of the body. This shearing strength is similar to friction for it acts along the breaking surface known technically, as the "surface of rupture." Friction, like shear, is a resistance to sliding, but whilst friction is the result of lack of smoothness in touching surfaces whereby they indent one another as a pinion sets its teeth into a rack, shearing strength arises from the actual sticking or cohesion of the two surfaces to one another, owing to the presence of certain cementing materials. In carboniferous measures these may be bituminous cements or the oxides, hydrates, and silicates of iron and alumina, or many other compounds. In all shearing, when pressure holds the two surfaces together, the shearing strength is augumented by the resistance the material offers to friction when the two surfaces are parted.

Clearly not all breaking is shearing. If an iron bolt is loaded with weights, as in Fig. 1, it will break at right angles to the direction of the pull of the weights. Obviously such a breaking is not a shear, for instead of the pull being along the breaking surface, it is approximately at right angles to it.

As has been said, friction best illustrates shear. Let a block A, Fig. 2, be taken resting on a level surface B. Attach a light string C to the end of the block and pass it over a pulley D. We can increase the weight E, which is hung from the string C till the block moves steadily forward.

The resistance to motion is equal to the weight E (when E is of such weight that the block does not move with increasing speed), and it acts along the touching surfaces of A and B; namely, along $a\,b$, and the force or weight E acts along the line $d\,e$ parallel to $a\,b$. Thus the line of the pull is parallel to the resisting surface (which corresponds to the breaking surface in the case of shear). It is a matter of no importance, within certain limits, where the string is attached to the block A, the pull of the weight will equal the resistance on the surface represented in the drawing by the line $a\,b$.

There may be several light strings like C attached to A; namely C_1, C_2, C_3, as in Fig. 3, passing over pulleys D_1, D_2, D_3, with weights E_1, E_2, E_2, loading their extremities. The strain on $a\,b$ will be equal to the weights E_1, E_2, and E_3, added together. If A be glued to B, the resistance to motion will be increased, but E_1, E_2, and E can any one or all of them be increased till breaking takes place. Only we must remember that, when breaking of the glue occurs, instead of the block A being drawn with a slow motion it will be pulled with a rapidly increasing motion toward the pulleys D_1, D_2, and D_3, for the weights E_1, E_2, and E_2 as now increased will more than overcome the resistance offered by the friction of A on B as soon as the glue is sheared across.

Summing up the principles discovered, they are these:

1. Breaking, as here considered, is parallel to the direction of the breaking forces.

2. The distance of the breaking forces from the breaking surface does not materially affect their action.

3. The resistance to breaking is equal to the sum of the breaking forces, up to the time when the breaking takes place.

4. When once breaking takes place no further resistance remains, except that which is furnished by friction.

Suppose that the block A and the surface B with supported pulleys, be turned at right angles, as shown in Fig. 4, so that the weights E_1, E_2, and E_2 hang down freely. Then if the glue binding A to B be considered unbroken, the resistance of the glue will be a resistance to vertical shear and the shear will equal the weights of E_1, E_2, and E_2, together with the weight of the block A. When the surfaces at $a\,b$ break apart, no further resistance remains, there being no pressure on $a\,b$ to create friction, consequently the block A falls with a rapidly increasing speed and without giving any premonitory warning.

Now, instead of imagining weights E_1, E_2, and E_3, as hung from the block A, we can imagine them resting on the top of the block, as in Fig. 5. (The linear dimensions of the block, it should be noted, have been increased fourfold for greater distinctness.) The weights are there marked E_1, E_2, and E_3, as in Figs. 3 and 4, but they are drawn as circular weights, as is customary in treatises on Strength of Materials.

Imagine the block A cut and glued together just to the left of E_1, E_2, and E_3, at the surfaces represented by $a_1\,b_1$, $a_2\,b_2$, and $a_3\,b_3$. Then, neglecting the weight of the block, the shear on $a_3\,b_3$ will be the weight of E_3; on $a_2\,b_2$, the weight of E_3 increased by the weight of E_2; and on $a_1\,b_1$, the weight of E_1, E_2, and E_3, combined. We have seen this is the shear on $a\,b$. At each of these reglued cuttings must be a resistance, the strength of the glue under shear, equal to the shear at that point, or the glue will break and a part or all of the block fall. Now $a_1\,b_1$, $a_2\,b_2$, $a_3\,b_3$, or any other cuts, real or imaginary, are called "sections" or cuttings of the block A. These principles and definitions enable us to write the following important statement:

1. The shear at any section is the sum of the weights or loads (vertical external forces) to the right of the section, thus the shear on section $a_2\,b_2$ is the sum of the loads to the right of that section; namely, E_1 and E_2.

2. The shear on the section over the edge of the support is as large as any.

We do not need to imagine all these gluings. We can see that the action will be the same if A be a solid block without breaks extending to the left beyond the surface B and so disposed that it cannot topple over to the right. Nor do we need to imagine loads E_1, E_2, and E_3, but may consider them replaced by the weights of the block. For instance, let the block A (let us now imagine it as a beam extending well to the left over support B) be 6 feet long, as in Fig. 6. Let its overhanging portion weigh 6 tons. Then the weight per foot-run, as it is termed, is 1 ton, or 2,000 pounds. Let us represent it by the letter w. Let us represent the length in feet to any section by l and the shear at any section by S_s in pounds. Then we may write

$$S_s = w\,l.$$

If, as in Fig. 6, $w = 2{,}000$ pounds, and if we say that $l = 3$ feet, then S_s, as shown in the figure, $= 2{,}000 \times 3 = 3$ tons. But, it is clear that the weight of the whole beam and all that is put upon it must be supported by the upward lift of the supports or the beam would go down. This lift of the supports we call the reaction, because it opposes the natural action of the beam, which is to fall unless it is supported. Let us call the reaction R, expressing it also in pounds. Then if, as in Fig. 7, the beam be L feet long and loaded with a burden of earth e pounds per foot-run, and if the beam has a weight of w pounds per foot-run

$$R = w\,L + e\,L$$

and we write $R = S_s = w\,L + e\,L$, S_s being the shear at the support. The reaction of the supports together with the loads and weights are the external forces acting on the sections of the beam. The shears and resistances to shear are the internal forces set up in the section to balance the external forces. In Fig. 8 the internal resistances are marked R_1.

When beams are supported at both ends and of equal cross-section throughout, as well as equally loaded, and when both supports are at the same level as in Fig. 8, one support must bear as much as the other and together must uphold the whole weight of the beam. Calling the reaction at one support R_1 and at the other R_2,

$$R_1 = \frac{w\,L + e\,L}{2} = R_2 = \frac{w + e}{2} L$$

The reaction at the right-hand support, that is, equals $\frac{w+e}{2} L$.

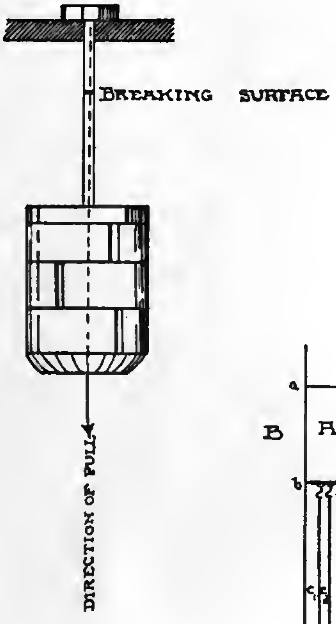


Fig. 1.

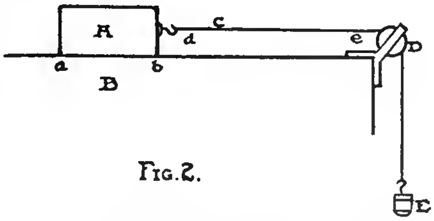


Fig. 2.

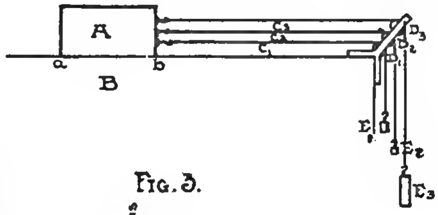


Fig. 3.

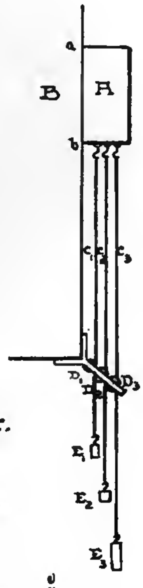


Fig. 4.

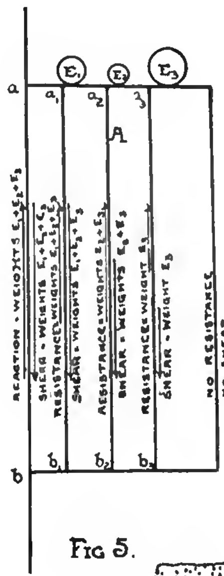


Fig. 5.

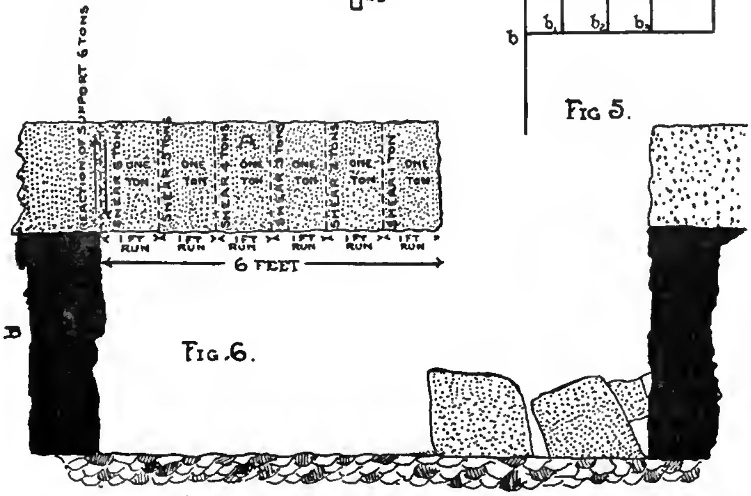


Fig. 6.

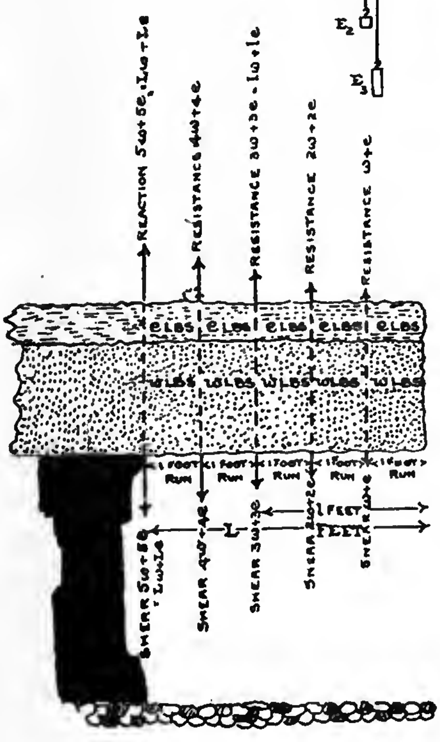


Fig. 7.

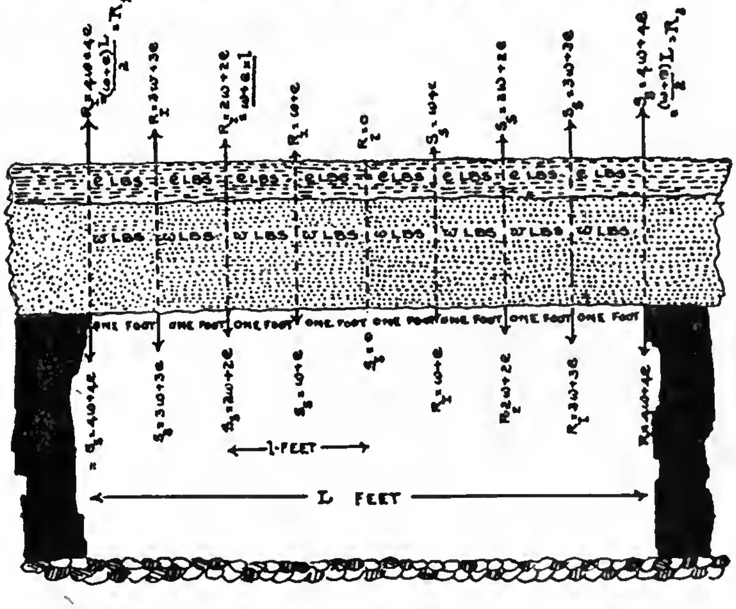


Fig. 8.

This causes a shear upward. A foot away the shear is $\frac{w+e}{2}L$ – the weight of 1 foot of rock (w) and 1 foot of overburden (e).

$$\therefore \quad S_s=\frac{w+e}{2}L-w+e=\frac{w+e}{2}(L-2).$$

The shear l feet away from the reacting support is

$$\therefore \quad S_s=\frac{w+e}{2}(L-2l)$$

when $L=2l$ or when the section considered is half-way between the supports.

Then $$S_s=\frac{w+e}{2}\times 0=0.$$

That is, at the section there is no shear. After the center is passed the shear is increased but the sign becomes "reversed" from plus to minus, the minus member in the equation becoming the larger and overpowering the plus member. To the right of the center the beam breaks so that the part of the beam to the left of any section goes down. On the other hand, to the left of the center, it breaks so that the part of the beam to the right of any section goes down. Thus the sign of the shear is changed from plus to minus. The difference is real but not very important. Restating results we have the following important rules:

1. Shear is zero at the center and at the side of a room or other working is equal to the load on the adjoining rib due to the undermined rock.

2. Shear increases proportionately from the center line of the room out to the pillar.

All this seems contrary to experience. The roof usually breaks in the center, not near the rib. But it must be remembered that only the roof of rooms where the cover is shallow will break from shear and we are now considering shear alone. But shear modifies other forces breaking up a roof mass and grows more important as the roof breaks up and therefore is no less essential a force to be studied, than those of tension and compression. Moreover, just as shear plays an important part on the manner of breaking of present-day mine roofs, so, also, it has played an important part in the fracturing and cleaving processes of past geologic ages.

PEAT BOGS OF IRELAND

Consul H. S. Culver, of Cork, reports on the difficulties that have been encountered in utilizing more extensively the peat beds of Ireland. One-seventh of the entire area of the country consists of bog lands—one bog alone, the Bog of Allen, has an area of 240 square miles and extends across the Central Plain, almost from Dublin to the Shannon.

Although peat has been for ages the fuel of the poorer classes in the remote districts of Ireland, by whom it is used in the same manner as wood in open fireplaces, no process has yet been discovered for converting it into a desirable and economical commercial fuel. Many schemes have been tried in Ireland in recent years, and extensive and well-equipped plants have been erected in different parts of the country at enormous expense in an effort to convert the peat into a profitable fuel for manufacturing purposes, but so far all efforts have been disappointing. It was thought at one time that by applying pressure to the product in its green and wet state and compressing it into a convenient shape, like briquets, it could be successfully used in the place of coal, but in most cases this process has had to be abandoned. So long as the briquets can be kept free of moisture they remain firmly intact, but the least moisture causes them to disintegrate and crumble to pieces. Numerous extensive plants are idle in the country waiting for some enterprising company to take hold of them and through some new process or through further experiments convert these bog lands into mines of wealth.

COAL NOTES

In an interview in New York, just before he sailed for Europe, General F. M. de Sousa Agular, of Rio de Janeiro, who has been in this country as a special commissioner from Brazil on a trade mission, authorized this statement: "Brazil has enormous iron deposits practically undeveloped, but no coal. I came on a special mission for the purpose of establishing closer relations with your manufacturing industries, especially with your great steel companies."

Forty-seven damage suits brought as a result of the Browder Mine explosion on February 1, have been settled. Thirty-four men lost their lives in this explosion, which was one of the most disastrous in the annals of Kentucky coal mining. The refusal of the indemnity company, in which the coal company was insured, to contribute to the settlement made it doubly hard for the Elk Valley Co. to settle; however, they did so, to the satisfaction of the lawyers at least.

Nearly 1,000,000 tons of coal was handled over the piers of the Norfolk & Western, Chesapeake & Ohio, and Virginian railways during the month of August. This tonnage broke all previous records. The Norfolk & Western shipped 391,335 tons from Lambert Point, the Virginian 116,870 tons from Sewalls Point, and the Chesapeake & Ohio 388,048 tons from Newport News, a total of 896,253 tons. Of the tonnage shipped from Norfolk 91,363 tons were foreign cargoes and 22,023 tons for bunker use of foreign vessels, a total of 113,386 tons. While the August tonnage was a surprise, it is predicted that there will soon be shipped more than 1,000,000 tons a month.

During a recent visit to the Pacific coast, William Mackenzie, of Mackenzie & Mann, told representatives of the press of notable improvements to be made at the Dunsmuir collieries, in British Columbia, which he and his associates purchased for $11,000,000 a number of months ago. It is said that $3,000,000 will be spent on development and equipment. W. L. Coulson, an American, became manager of the mines recently. The Dunsmuir collieries have been one of the chief sources of coal supply on the Pacific coast for years past and while the chief intent of the purchase was to secure a fuel supply for the Canadian Northern when it has been built through, the large trade of the present day will be exploited to the fullest extent by the provision of the most up-to-date facilities. Apparently the undertaking will rank well up with the largest companies on the continent in due course of time.

The more it is considered, the greater appears the heavy loss which the miners of Illinois sustained by reason of the late strike. Corresponding gains were made largely by the non-union miners of West Virginia, though the distribution of gains went through a number of other states.

It seems certain that the loss of wages sustained by the Illinois miners covers at least 20,000,000 tons. The annual coal production in Illinois is in round numbers 50,000,000 tons. The men have been idle practically six months. Production during the summer months is on the average considerably below that for the winter season, but a very large percentage of the Illinois product consists of steam coal, the consumption of which is fairly uniform throughout the year. And moreover the mines at the expiration of the six months' idleness are so crippled in various ways that production during the next six months will be considerably below par. One prominent operator says he doubts if much more than half the usual winter tonnage is mined in Illinois. On the whole, 20,000,000 tons is believed to be a conservative estimate of the tonnage decrease in Illinois. The wage cost of this tonnage under the increased scale, adopted at the Cincinnati convention, a wage scale which the miners might have had at the outstart, is about $1 per ton, 90 cents under the old scale and 10 cents additional granted by the Cincinnati convention, a total of $20,000,000 in wages actually lost by the Illinois miners.

To offset this loss the miners have gained for the next

18 months an average increase in wages above the Cincinnati scale of 2 cents a ton, 3 cents in Franklin and Williamson counties, 2 cents in the northern field, and $1\frac{1}{2}$ cents per ton for shot firers in other districts of the state. Assuming that the production in Illinois will be for the next six months 30,000,000 tons and that next year the high record will be reached, 50,000,000 tons, the miners' gains of 2 cents per ton on a production of 80,000,000 tons is $1,600,000, as an offset against the actual loss of $20,000,000. It would take 20 years for the men to make up that loss of $20,000,000, for their increased wages of 2 cents per ton on 50,000,000 tons annually amounts to an even $1,000,000 per year.

It is ridiculous to assume that the new scale will last that long. It certainly will not if history repeats itself. A serious loss to the Illinois miners is therefore the inevitable conclusion as a consequence of the recent strike.—*J. B. M., in Coal Trade Journal.*

A valuable seam of shale recently discovered near the Seafield works of the Pumpherston Oil Co. will be of advantage to the Scotch mineral oil industry, the expansion of which is measured only by the available supply of native oil shale and the products of which are in increasing request. Boring operations have been in progress for some time past in this district, and the shale now revealed is said to be both thick and rich. New pits will be at once opened and a considerable impetus given to local employment.

On the Thorosk estate, near Stirling, a good seam of coal, the existence of which has been known for some time, is now about to be worked, the property having been acquired by Messrs. Archibald Russell & Co., who have the neighboring Fallin collieries. Thorosk lies adjacent to the Bandeath colliery, the coal from which is steam coal. The Thorosk coal, however, is more suitable for domestic consumption.

At Pittston, Pa., the surface has been disturbed by a cave. In one location, where there was a well, "marsh gas" accumulated in such quantities that the owner covered over the top and piped the gas to his kitchen stove. In another case, a pipe which was driven in the ground about 18 inches, tapped sufficient gas to run an argand burner continuously.

The record for freight tonnage on the middle division of the Pennsylvania Railroad between Altoona and Harrisburg, Pa., established about a year ago, was broken on the afternoon of August 18, when a coal train over $1\frac{1}{4}$ miles long, consisting of 120 steel cars having an individual capacity of 100,000 pounds, was hauled over the road. The record made a year ago was 105 cars. Telephonic communication was established between the engine and the last vehicle, and the engine driver was directed by the officials at the rear end of the train. An average speed of more than 20 miles an hour was maintained over the division.

The Illinois Mine Rescue Station Commission has appointed Mr. Richard Newsam, of Peoria, manager of the three stations provided for by the law. The southern station will be located at Springfield, the northern one at La Salle, and another one at Benton. Contracts for the buildings have been let which call for the buildings to be completed not later than January 1, 1911.

The Hacklebernie coal mine about 2 miles from Mauch Chunk, Pa., was the first coal mine opened in the anthracite regions. The mine, which has been operated during the past few years by David E. Pursell and James M. Breslin, under the firm name of the Hacklebernie Coal Co., has been closed by the Lehigh Coal and Navigation Co., its owner, and will probably never be operated again. It was opened shortly after the discovery of anthracite coal by Philip Ginter, a hunter, at Summit Hill, in 1794, and it was from this mine that coal was first taken and hauled to Mauch Chunk, and there loaded in arks and sent by river to Philadelphia to market. This was long before any railroad and many years before the Lehigh canal was built. The Lehigh Coal and Navigation Co. ceased operating this mine as soon as a railroad was built throughout the Panther Creek valley, because coal in that region could be mined and transported more cheaply than at the Hacklebernie Mine, where coal had to be hauled in wagons. There is still coal to be found in abundance at this mine, but the Lehigh Coal and Navigation Co. will not operate it on account of its many collieries at other and more convenient places. This mine is crossed by the famous Switchback, about 1 mile from Mount Pisgah, and is prominently pointed out to passengers taking a ride on that famous road as the pioneer anthracite mine in the entire world.

The people in New England, New York, and New Jersey are greatly interested in the verses of the Springfield *Republican* poet. The New York *Sun* declares that his song, "Talk To Me About Coal" has not been surpassed in the interest manifested by the public since the time when "Drink To Me Only With Thine Eyes" appeared.

A miner, Joseph Tressa, fell down the shaft at the Forty Fort colliery, near Pittston, Pa., and after brushing the dust from his clothes and picking a speck or two of coal from his eye calmly proceeded back to his room in the 11-foot seam and continued his day's work. Just how far Tressa fell is not known, but the shaft is 1,000 feet deep. Tressa boarded a cage about to descend and gave the signal for the 11-foot seam. When the carriage stopped at this landing he alighted and signaled all right. The cage immediately descended toward the red-ash bed, 900 feet below. Just how it happened Tressa does not know, but the moment the cage disappeared he lost his balance, fell, and catching up with the cage held to the top just above the shield. When the bottom was reached he called to the footman and the cage was lowered so that he could jump down from his perch. This method of falling down shafts is highly recommended, besides proving that a man in an office elevator cannot bump his head on the elevator roof no matter how swift the elevator descends.

The Buck Mountain colliery, near Weatherly, which was once run by the late William Spencer, of Pottsville, is to be reopened by the Lehigh Valley Coal Co. It will be remembered that William Spencer was the first to call on the governor for troops during the "Molly McGuire" excitement. After finishing work at Buck Mountain he went to West Virginia and opened the Bottom Creek colliery, which his nephew, Mr. Samuel Patterson, now manages.

Consul Augustus E. Ingram forwards from Bradford a series of English press accounts on the extension of the Yorkshire coal fields, which indicate that they have thirty-five thousand millions more tons of coal available for mining than was supposed.

WORLD'S PANAMA EXPOSITION

The contest between New Orleans and San Francisco for the World's Panama Exposition in 1915 runs merrily on. The New Orleans Exposition Committee is distributing throughout the country a "logical point map," according to which there are 70 cities within an average distance of 900 miles from New Orleans with a combined population of 20,000,000, while within the same average distance of San Francisco there are only eight cities, having a combined population of 1,000,000 people. New Orleans' principal claim is that if the Exposition be given to San Francisco the people of this country would be penalized $200,000,000 in railroad fare over what it would cost to go to New Orleans.

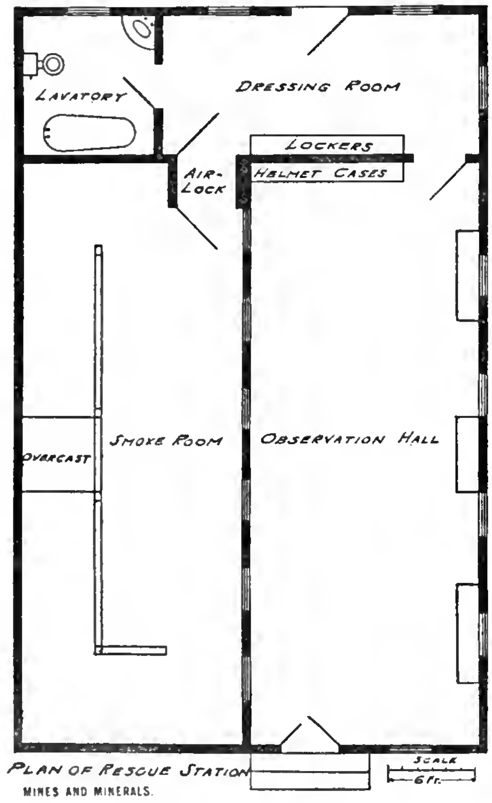
Breccia in the form of rough blocks being similar in texture and uses to marble is dutiable, under the application of the similitude provision of paragraph 481, at the same rate as marble in rough blocks, at 65 cents per cubic foot under paragraph 111, tariff act of 1909.

RESCUE STATIONS IN ILLINOIS

*By R. Y. Williams**

Design and Equipment of a Joint Rescue Station—Plan for Stations in Illinois

The discussion of any problem affecting coal-mining methods applicable to such a state as Illinois, is of considerable interest, because Illinois stands second in the list of the 27 coal-producing states, and holds in reserve more bituminous coal than any other state in the Union. It is also the second largest producer of bituminous coal. This position, however, has been attained by the sacrifice of many human lives and the loss of millions of tons of coal. The indefinite continuance of these conditions would be most unfortunate. But coal-mining engineers are facing these problems seriously. Already the explosiveness of coal dust has been demonstrated, methods of its control investigated, a list of permissible explosives has been published by the government, and a paper on some phase of the efforts now being made for the conservation of that greatest of all natural resources, human life, it is believed will be of value.



Plan of Rescue Station

The investigations in this field are proceeding along two lines—the prevention of mine accidents, and the work of mine rescue. While the first is by far the more important problem, it is nevertheless apparent that calamities such as mine explosions and fires are of such frequent occurrence as to be a constant menace to the safety of the men and of the mine. For these reasons the work of rescuing life and saving of property are urgent problems in present-day coal mining. With this in mind, the following has been written in an endeavor to present a general scheme for Joint Mine Rescue Stations in Illinois.

Early in March, 1909, the United States Survey, in cooperation with the State Geological Survey and the University of Illinois, established at Urbana, Ill., a mine-rescue investigation station, which is a branch of the Central Testing Station at Pittsburg, Pa. This substation is equipped with oxygen helmets, and has a gas-tight room resembling a coal mine, in which miners may test the efficiency of modern breathing appliances. Already this mine-rescue laboratory has been visited by many operators and inspectors; a large number of miners have been trained in the theory and practice of rescue work; and considerable assistance has been given on the occasion of several mine fires and explosions. One station, however, cannot render adequate service in a state having more than 400 large shipping mines scattered throughout an area of 275 by 180 miles. In fact, it was not the intention to establish at the Urbana station a permanent rescue corps to act in the event of mine disasters in ordinary rescue work. This station was established primarily to supply the equipment and trained assistants required for the study, by the Survey experts, of mine explosions in the Illinois-Indiana-West Kentucky coal field, and in the hope that the station would offer a means of demonstrating modern mine-rescue practice to the mining fraternity of this field.

When an explosion or fire occurs in a coal mine, conditions are usually such as to require that the men entering the mine be protected by helmets, which must be supplied as quickly as possible. It is, therefore, necessary, both for rescue and investigation work, that stations should be within easy reach of each important coal field or division of the field, so that the trained experts can reach and enter the mine promptly following the disaster. Were each mine in the state equipped with a complete rescue station and manned with a corps of trained rescuers, we would have ideal conditions for the recovery of life and preservation of property after a calamity. But, unfortunately, these stations are quite expensive; equally unfortunately, the present selling price of coal is so low that the small margin of profit is already a matter of much concern, so, unless some remedy for this condition can be found, but few individual mine owners, I fear, would feel warranted in authorizing such an expenditure of money. In order to overcome this obstacle and still obtain the benefit of the modern mine-rescue methods, the suggestion of joint rescue stations, it is believed, should be of great value.

In general, by a joint rescue station is meant one specially designed and equipped for a particular group of mines; located centrally to each of such mines by natural or special transportation facilities, and financed either on a "per ton" or a "share-and-share-alike" basis by each mine benefited. At such a station, at stated intervals, a squad of men from each mine could report for training in rescue methods under the direction of an experienced instructor, assisted if possible by a physician. From such station the rescue paraphernalia could be furnished without delay in the event of a calamity befalling any mine of the group. There are four divisions of the subject which may be amplified:

1. Design and equipment of a joint rescue station.
2. Character of the training to be given at the station.
3. General advantages that may be derived from the stations.
4. Plan for rescue stations.

Design and Equipment of a Joint Rescue Station.—Breathing appliances were invented abroad and their use established long before we in this country recognized their value. But if we study the design and equipment of these foreign stations, we might be discouraged in an endeavor to copy them. Conditions abroad and in this country are entirely different; and valuable as is a study of foreign methods, we could not afford the luxury of such structures as the Howe Bridge Station in England. There are, however, certain requirements that must be followed in design. The station building should contain a gas-tight room, 40 feet long by 20 feet wide and 10 feet high. The interior of this room should be fitted to resemble a mine and to afford opportunity for the practicing miner to do work similar to that required in the event of an actual disaster. In

* Paper presented before Western Society of Engineers, May 4, 1910.

a number of stations already built in this country, it has been the practice to divide this room longitudinally and to construct an overcast on one side, the aim being to present a passageway about the room, the travel over which would represent the journey of a rescue party through the entries of a mine. This room should be furnished with mine props and a frame consisting of four pieces of 6″×8″ timbers joined together in the shape of a square and tied with two iron rods, in which props may be set and capped with wedges; also brattice cloth, stretchers, and a canvas dummy filled with sand and sawdust so as to weigh about 165 pounds. In order that men may gain confidence in working in the presence of gas, sulphur candles may be burned in this room to form a chokedamp; charcoal may be fired in open salamanders yielding blackdamp; hydrogen disulphide may be generated producing stinkdamp, or ordinary dense smoke may be obtained by burning dampened excelsior.

Adjoining the smoke room and separated from it by a glass partition, should be an observation room where visitors may sit and view the work of the miners. Here, too, the instructor may observe and record the performance of each member of the rescue squad. In this room there should be wall cases in which the rescue apparatus may be hung and protected from dust; there should be work benches to facilitate the cleaning of instruments and the charging of electric safety lamps; and a place to store the cases in which the apparatus is shipped.

Back of the smoke room there should be a lavatory containing toilet, shower baths, and lockers for the accommodation of the miners coming to the station for practice.

The equipment of the station should include oxygen helmets, or other suitable breathing appliances furnishing a dependable supply of pure air, with the aid of which men may safely enter any kind of the foulest and most poisonous atmosphere in order to perform rescue work.

Apparatus and supplies for recharging these machines.

Portable electric safety lamps with a convenient device for recharging.

A supply of some standard make of oil-burning safety lamps.

One or more resuscitating cases for use in reviving men overcome by the afterdamp of mine fires or explosions.

Special cases or trunks, of convenient size for handling, in which the above apparatus may be quickly packed and safely transported to the scene of an accident.

The Character of the Training Given.—The character of the training given should include a general study of the conditions that obtain during and after a mine fire or explosion, with special detailed reference to concrete cases. With these actual occurrences in mind, plans should be discussed for successfully solving these problems according to modern rescue practice. The principles on which the machines used at the station are constructed and operate should be explained; and a thorough first-hand knowledge of the manipulation of the various apparatus should be acquired by the practicing miner. The actual training of the mind and body to do work similar to that required in the actual recovery of a mine and in the presence of deadly gases should be given by means of drills in the smoke room. In this way, men become acquainted with the possibilities and limitations of the machines, gain knowledge as to their own prowess as rescuers, and learn to work in squads under the leadership of one of their comrades. For mental and physical ability shown in the work, a certificate of competency should be awarded. This would tend both to keep up interest in the work of the station, and to be of especial value as a reference card when a disaster occurs.

Advantages That May Be Derived From Stations.—The advantages that would obtain from such stations are in a large measure obvious. It often happens in an explosion that the ventilation machinery is thrown out of commission or totally destroyed. Also, it is often necessary after a gas explosion to stop the fan to prevent a series of subsequent blasts and to control a mine fire by cutting off all ventilation. Previous rescue methods have afforded only a choice between two evils; either close the mine with concrete stoppings and leave it sealed indefinitely, or start the fan, send in the men, and trust to luck, with the result of the loss of many lives and much property.

With the introduction of modern practice, however, rescue work assumes a decidedly different aspect. With the aid of the breathing appliances, trained men may enter the mine at once with comparative safety and begin the task of recovery, without aid of air supply from the fan. As the work progresses, each step may be taken with a complete knowledge of the situation gained from the careful reconnaissance of the helmet men.

Not least among the advantages that would accrue from the employment of rescue stations, is that in cases of emergency there would be available squads of men trained for the undertaking, accustomed to working together and obedient to the commands of their leader.

A further advantage is that such rescue stations may become centers for the dissemination of knowledge among the men. In addition to the usual studies and lectures, local institutes could hold their meetings in the observation hall of the station; and talks and demonstrations on first-aid work could be given by the town or company physician with a view to forming first-aid corps similar to those that are meeting with such success in the anthracite fields of Pennsylvania.

Plan for Rescue Stations in Illinois.—It is clearly out of the question, in view of the destructive competition that at present exists in the coal trade of this locality, to legislate against the very life of the industry by requiring each operator to establish a rescue station or make other improvements not immediately necessary. And yet, considering all that has been and is being done by foreign and domestic stations, and remembering the advantages that would accrue from the establishment of these stations, along the lines of discipline, education, etc., we are compelled to recognize their value.

The legislature of Illinois appropriates annually $193,000 for investigation in agriculture. The mining and metallurgical industry of the state represents an output valued at approximately $150,000,000, and for the aid of these industries the state appropriates only about $25,000. In establishing and maintaining charities and schools, the state annually spends enormous sums; and while rescue stations are primarily devoted to training tending to life saving, they are or may be considered as educational centers. Moreover, the state has appropriated $2,500 for the relief of the sufferers of the Cherry disaster, and the legislature is now considering bills calling for an additional benefit appropriation of $50,000 to $150,000. In view of these facts, it seems reasonable to ask the legislature for a grant of funds sufficient to carry on work which has for its object the saving of life and property, the training of its citizens to be effective agents of a vast enterprise and the reduction of a constantly increasing number of deaths, a special appropriation of $30,000 and in addition an annual appropriation of $30,000, to be expended according to the following plan of operation:

The coal fields of Illinois would be arranged in three divisions, and in a centrally located city or town in each of these districts a central rescue station would be established. For example, LaSalle, Springfield, and Carbondale. Each of these three cities is a railroad center, enjoying exceptional railroad facilities. A station could be built for $5,000, and equipped with a complete line of apparatus for a like sum; itemized as follows: 12 oxygen helmets, or other suitable breathing appliances; 12 portable electric safety lamps; 12 oil-burning safety lamps; 6 oxygen tanks or reservoirs; 1 oxygen pump; 2 oxygen reviving outfits; 200 potash cartridges; 1 chemical cabinet for gas analysis; 15 cases or trunks for transporting the above apparatus; furniture, including chairs, tables, wall cases, etc., tools, supplies, etc.

Thus the three stations could be completely installed for the special appropriation of $30,000.

In charge of each of these three central rescue stations there should be appointed a man whose experience in coal mining has been large and varied—some one who can maintain the interest of the miners who visit the station, care for the apparatus, and keep the records. Over the entire rescue work, with power to purchase supplies, direct the course of training, and assume the entire charge in case of a mine disaster, there should be a man who is a mining engineer by profession, one who has had experience in all phases of coal mining, including mine rescue work, and one upon whom may be thrown with confidence the welfare of the whole proposition. It is proposed that such officer cooperate with the inspector in whose district a disaster may occur. It is also suggested that he report to a board of five persons to be appointed by the Governor, to consist of one inspector, one operator, one miner, the head of the Department of Mining Engineering at the University of Illinois, and one member of the Federal Inspection Force.

Each operator of the state should be asked to send a small number, say 4 per cent., of his employes to the nearest of the central rescue stations at least twice a year for training in rescue work. These men should spend at least 3 days at the station on each visit. In return for this action of the operator in bearing such expense for the safety of the lives of his miners, the men so trained should agree, in cases of emergency, to assist in the work of mine rescue, with the understanding that they are to receive only the "inside wage scale" for time devoted.

These three central stations could be made of great value to both miners and operators. But one further step is necessary

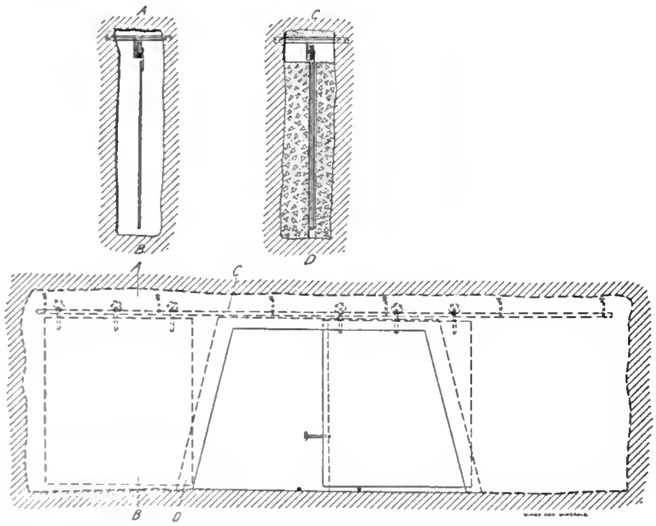


FIG. 1. PLANS AND SECTIONS OF MINE DOOR

to complete the plan and insure its entire success, because, even with the three stations thus located, there would be an appreciable lapse of time before the apparatus could be delivered to a mine in case of trouble. So to overcome this delay, the suggestion is made that all operators combine in what may be called local or private stations, consisting merely of five helmets and five electric safety lamps, charged and packed in cases, ready for transportation, in some town or a mine where there is always some one present who can deliver the helmets as needed. This local station might be in a fire department house of the town. These local stations would serve all mines within a radius of 15 miles. As the station equipment would consist of the five helmets and lamps, without any of the costly apparatus for recharging them, the expense, when divided among all mines within the 15-mile radius circle, would be very small. It would be the duty of the central rescue station people to inspect and charge these helmets periodically and see to it that they are in working condition for an emergency. This might be accomplished by having the local helmets brought to the central station by the miners when they visit it for training The object of the local stations is that the men at the mine may, in case of fire or explosion, have means at hand for preliminary work, or immediate rescue that may be necessary during the period while the men and equipment of the nearest central rescue station are journeying to the mine.

In concluding this paper I can assure you that I am urging the adoption of certain engineering principles that have been of inestimable advantage wherever used legitimately, and which have as their object the conservation of life and property.

EXTRA MINE DOORS

Written for Mines and Minerals

The Anthracite Mine Law, Article 10, Section 12, says: "An extra main door shall be so placed and kept standing open, as to be out of reach of accident, and so fixed that it can be at once closed in the event of an accident to the doors in use."

The intention of this section is to have an extra door on the main airways, so placed that in case of an explosion it will not be injured. The doors are to be kept standing open ostensibly for the purpose of allowing the blast from an explosion to pass through without injuring them. Possibly the law makers had something in their mind's eye that could stand in the way of an explosion and not be wrecked. If that be so, they should have placed a diagram in the article, for as a matter of fact when a good live explosion occurs, the mine entries are like gun barrels where all ahead of the powder moves forward. If door jambs are in a gangway or airway the chances are that if not knocked down by an explosion they would accumulate wreckage, so that the law could not be complied with and the door at once closed in case of accident to the door in use. It may be assumed further that a rock fall might occur to prevent the extra door being closed, particularly if the door frame was moved, and that, too, would prevent its being closed at once. Thomas H. Price, of Wilkes-Barre, Pa., Mine Inspector of the Seventh Anthracite District, evidently thought over this extra main door problem, for he has devised the one shown in Fig. 1, which surely comes as near to complying with the law as any so far advanced. The door frame is made flush with the walls of the entry. A space is then cut in the rib on each side, as shown in Section *A B*, Fig. 1. Suitable bearings for door hangers are then placed at the top of the openings and the forms for the concrete, shown in section *C D*, put so as to leave a space for the doors to slide back out of the entry when not in use. In the illustration, one-half of the sliding door is in the entry and the other in the recess in the rib.

Another case when this door becomes exceedingly useful is where shafts are partitioned for downcast and upcast and the partition lining becomes broken by fall in the shaft, or catches fire. By having one of these doors on each side at the foot of the shaft, and by closing the top of downcast shaft outside, the air will be conducted around the mines as before until all the men are taken out through the second opening required by the mine law of Pennsylvania.

It is evident that doors of this description could be used to advantage in case of mine fires, if judiciously placed with reference to panels and entries, so as to confine the spread of such fires by cutting out the air supply. Considerable more thought is being given to the prevention of mine fires than heretofore, owing to the Cherry Mine disaster and the decided stand the state of Illinois, through the suggestion of its mine commission, has taken to prevent a recurrence in that state. Doors of this kind could make a kind of rescue station to keep back the death-dealing gases until a rescue party reached those behind them. In the account of the Palau mine explosion in this issue of MINES AND MINERALS it is stated that several men were saved from death by a rock fall which held back the afterdamp. The records left by the Cherry Mine victims, also show the need of these doors.

MINE LABORATORY WORK AT GARY, W. VA.

*By Vitus Klier, Chemist**

Coal-mine laboratories should be supplied with samples which represent the large product reduced to small measures, that is, good average samples. If this standard is not maintained, the work in the laboratory, no matter how accurately it may be done, is useless.

Equipment of Laboratory. Methods of Sampling Coal, Coke, and Mine Air

The United States Coal and Coke Co. equipped the coal-mine laboratory, whose analytical room is shown in Fig. 1, at Gary, W. Va., in 1906, since which time 90,000 samples of coal and coke have been tested, or approximately 23,000 samples a year. This number of samples will be increased gradually as the output of the mines and ovens increases. The cost of analyzing one sample is about 10 cents, whether the analysis is for ash or a complete analysis of fireclay. The number of coal and coke analyses made in the laboratory daily varies from 120 to 150, although as many as 214 have been made in a single day. The variation in the number of analyses is due to the number of coke ovens charged and coal cars loaded for shipment.

The working force of the laboratory consists of a chemist, assistant chemist, and two laborers. The assistant chemist must be an ambitious young man with good habits and trained to the daily routine work which remains the same.

FIG. 1. ANALYTICAL ROOM, GARY LABORATORY

FIG. 2. SAMPLING ROOM, GARY LABORATORY

All coal going to the coke ovens is previously crushed. Every tenth larry that leaves the tipple is accompanied by a sampler who takes a sample at intervals, so long as fine coal runs from the larry spout into the oven. To do this he makes use of a flat wooden scoop 3 inches wide and 24 inches long. The coal he catches on this scoop he places in a bag made of some strong material and encloses a memorandum giving the name of the operation, the oven number and the date when the oven was charged. He then ties the bag up tightly and places it where it will be protected until all the ovens have been charged, when all the samples are collected and taken to the laboratory. When coal is to be shipped, the work of obtaining a good average sample is more difficult. To obtain as good a sample as possible a man is put on one side and below the end of the loading apron with a shovel, and as the coal slides into the cars the man holds his shovel under the falling coal. When one-third of the height of the car has been run in, this shovelful of coal is put on a clean steel plate 5 ft. × 5 ft. × $\frac{1}{4}$ in. When the second third of height of car has been run in, another shovelful of coal is taken and placed on the plate. As the car has to be moved three times before it is filled, nine heaping shovelfuls of coal are thus taken from the different parts at different heights of the car and a fair average sample is thus obtained. The coal on the plate naturally consists of varying sizes, from that of a lump as large as one's head down to fine slack. All lumps are broken up to the size of a walnut with a hammer, after which the coal is thoroughly mixed with a trowel; spread out in a circle and quartered down to a sample of about 10 pounds. This sample is then put in a bag marked with the name of the operation, car number, and date, after which it is sent to the laboratory. At the laboratory there is a sample preparing room the interior of which is shown in Fig. 2. This is provided with a quartering machine of the writer's design that reduces a 10-pound sample to 5 ounces in one minute.

Whenever a coke oven is to be sampled it is divided into quadrants. From each quadrant one piece of coke, whose thickness is that of the coke bed in the oven, is taken and placed in a sack which accompanies the coke-drawing machine until all ovens are drawn. The sample is then marked with the number of the operation, date the oven was drawn, and then sent to the laboratory.

To ascertain if the coke has been properly watered a bucket on a rod or a handle is held under the conveyer of the coke-drawing machine at intervals until the oven is drawn. The pieces of coke so caught are broken off and put into a can having an air-tight cover. The sample is weighed, heated in a steam bath up to 100° C., and the moisture determined. All coke samples received at the laboratory during the week are saved until Saturday, when they are combined and a complete analysis made which represents an average analysis of the coke shipped during the week. Samples reach the laboratory about 4 or 5 o'clock in the afternoon and work is commenced in the evening. By 4 o'clock the next day the certificates of analyses are ready to be distributed to the superintendents of the various plants. If this report shows any irregularity the superintendent proceeds to investigate and eliminate the causes that led up to the evil.

There are different sections in a mine and each section has a man whose business it is to see that the coal is cleaned before being loaded into mine cars. The superintendent when informed that his coal is not clean is able immediately to locate the section from which the dirty coal was loaded. In order to do this he takes samples of coal from the different sections as they are run through the crusher at the tipple. These samples he sends to the laboratory and after receiving the analyses he is posted as to which section needs special attention. As stated, all coal is crushed previous to its being sent to the coke ovens, and at the laboratory tests are made to determine the fineness of the slack

* Read at Bluefield Meeting of West Virginia Mining Institute.

in order to keep the superintendent informed as to the condition of his crusher.

The methods adopted at Gary for sampling and analyzing the mine products has up to the present time proved satisfactory. To show the accuracy of this system comparisons between the analyses made at Gary and South Chicago, for ash, are as follows:

Works Number	Difference Between Gary and Chicago	Date
	Per Cent.	
1	.58	March, 1909
2	.32	March, 1909
3	.13	March, 1909
4	.15	March, 1909
5	.32	March, 1909
6	.48	March, 1909
7	.46	March, 1909
8	.45	March, 1909
1	.36	April, 1909
2	.87	April, 1909
3	.23	April, 1909
4	.06	April, 1909
5	.08	April, 1909
6	.65	April, 1909
7	.32	April, 1909
8	.42	April, 1909

The samples at Gary were taken from the ovens and the samples at South Chicago from the cars on arrival. Fig. 3 shows the steel tipple at the Gary No. 7 mine. The lower building is for the crushed oven coal; the small addition on the left-hand side is for domestic coal used at the plant, and the chute is for loading locomotives.

In addition to testing mine products at the Gary laboratory, tests are also made for information in regard to the condition of the mines and to ascertain the quantity of gas and dust in them. This is accomplished by means of an aspirator and a sampling tube. The aspirator is made of tin and has 15 liters capacity. It is provided with a stop-cock and on the upper end is a filter tube with a filter inserted. To take a sample the apparatus is filled with water, the stop-cock is opened, and as the water is run out at the lower end the dust coming in is

Fig. 3. Steel Tipple, Gary, W. Va.

caught by the filter in the tube. The air sample is taken with a glass sampling tube which is filled with water before reaching the place where the mine air is to be sampled. This apparatus is supplied with two stop-cocks, one to permit the water to run out and the other to permit the air to enter as the water runs out. As soon as the water is all out of this tube both stop-cocks are closed and the tube is sent to the laboratory where its air contents are analyzed Air and dust samples are taken at the same time and in the same place regularly So far tests for inflammable gas have not been necessary at Gary, as the amount of gas found in samples of mine air taken at the coal face has never been over 1 per cent., and the dust determined at the same time has never exceeded 5 milligrams per cubic meter. According to Taffanel, 4,500 milligrams of dust per cubic meter will start an explosion. However, the sampling is continued at Gary as a means of precaution. Every superintendent when in doubt as to gas takes samples and checks his Pieler lamp to insure the safety of the mine. The following is the form of report used, and it shows how closely the Pieler lamp checks with analyses, the greatest difference being not quite ¼ of 1 per cent.

Date	Sample Number	Per Cent. Analysis	Pieler Lamp Per Cent.
9–1	1	.7	.50
	2		
9–10	1	.4	.50
	2	.2	.25
9–16	1	.2	.50
	2	.2	.25
	3		
9–30	1	.4	.50
	2	.3	.25
	3		

The laboratory is not only used to analyze the output of the mines and of the coke ovens, but is also used to check the materials purchased, such as miners' oil, blasting powder, brick material, etc., so as to know exactly what is being paid for and its quality.

PERSONALS

Sidney F. Walker, R. N., of Bloomfield Crescent Bath, England, has been elected President of the South Branch of the Association of Mining Electrical Engineers.

W. F. Zimmerman, M. E., is reassociated with the Pittsburg Testing Laboratory, with office at 50 Church Street, New York City.

Prof. John J. Rutledge, of the Bureau of Mines, and Dr. M. J. Shields, have been touring the southern coal-mining states lecturing to miners. Professor Rutledge lectures on mine apparatus and safety in mines, and Doctor Shields on how to render first aid to the injured.

A. E. Borie has resigned from the New Jersey Zinc Co. to accept a vice-presidency of the Taylor Iron and Steel Co., High Bridge, N. J. New York office, No. 100 Broadway.

Wm. A. Evans has been appointed superintendent of the Hartshorne group of mines of the Rock Island Coal Mining Co., vice R. J. Pickett. Mr. Evans will continue to act in the capacity of chief engineer for the mines of this company, located in Oklahoma.

Charles Enzian, Wyoming Division Engineer of the Lehigh Valley Coal Co., has been appointed mining engineer for the Anthracite Division of the United States Bureau of Mines. His duties will be to cooperate with miners and operators and state officials in safeguarding lives and property in and about the mines. He will be expected to investigate thoroughly the many mining problems and will make himself familiar with serious questions confronting the mining officials. He is to go to Pittsburg for a few weeks special work and then will be permanently stationed at Wilkes-Barre.

J. B. Tyrrell, mining engineer of Toronto, Canada, has returned from London, England.

Arthur Lakes and Arthur Lakes, Jr., have moved their offices to rooms 701-703 Gas and Electric Building, Denver, Colo., where they will continue their practice as mining and metallurgical engineers.

Eli T. Conner, Mining Engineer, has opened an additional office in the Traders Bank Building, Scranton, still retaining his office in the Real Estate Trust Building, Philadelphia.

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

(Continued from October)

NOTE.—The following questions, selected from different examinations, are here numbered consecutively. The examination from which the question is taken and the number of the question are found at the end of each.

Selected Questions of the Illinois Examinations for 1909, Held at Springfield

QUES. 1.—How is carbonic acid produced in mines, and how is its presence detected?—**Q. 4, April 12.*

ANS.—Carbon dioxide (CO_2) or carbonic acid gas is produced by the combustion of carbon in a plentiful supply of air. This combustion is called oxidation; such is the burning of lamps, powder, coal, wood, or other organic matter; breathing of men and animals, decay of timber, etc. The oxygen of the air unites with the carbon, in the proportion of two atoms of oxygen to one atom of carbon, forming carbon dioxide (CO_2). The presence of this gas in mine air is manifested by the dim burning of lamps or their complete extinction when a sufficient percentage of the gas is present.

QUES. 2.—What is the quantity of air required by law for the ventilation of a mine employing 150 miners, 21 day men, and eight mules?—**Q. 5, April 12.*

ANS.—The Illinois mining law requires 100 cubic feet of air for each man and 600 cubic feet for each animal—Section 19, (*a*)—to be circulated per minute throughout the mine. This would require in this case $100\ (150+21)+600\times 8=21{,}900$ cubic feet per minute. The mine inspector is authorized by law to require that this circulation be increased whenever, in his judgment, the conditions demand a stronger current.

QUES. 3.—If a mine is generating much firedamp and there is danger of explosion, what kind of material would you use to build overcasts—wood, stone, iron, or brick? Give reasons. **Q. 7, April 12.*

ANS.—Opinions differ in regard to the material to select for building overcasts in gassy mines where explosions are liable to occur. While it is desirable to build overcasts that will resist, ordinarily, the force of an explosion in the mine, it must be remembered that nothing but an overcast driven in the solid strata can resist the tremendous shock of many mine explosions. It is also claimed that the resistance of an overcast, preventing as it does the escape of the expanding gases, causes the destruction to extend further in the mine and penetrate workings that would otherwise not be reached. However this may be, it is a fact that the iron plates and beams used in constructing an overcast are often so badly twisted as to make it impossible to remove them or repair the damage quickly so as to reestablish the air-current. Timbers, if broken or dislodged, can be cut away and removed and a new overcast built temporarily to restore the current. For this reason it would seem advisable to build an overcast of heavy timbers and plank well braced and provide a large trap in the floor of the bridge so as to relieve the pressure and perhaps avoid the destruction of the air-crossing.

QUES. 4.—When making your rounds in the morning you find gas at the face of an entry, and after measuring it with a safety lamp you estimate its volume to be about 21 cubic feet. What volume of flame would you expect if this gas were to be ignited, the temperature of the mine being 65° F.?—**Q. 8, April 12.*

ANS.—The flame volume of marsh gas (CH_4) mixed with sufficient air to insure the complete combustion of the gas is generally assumed to be 10 times the original volume of the firedamp mixture. If it could be assumed that the body of firedamp mentioned in this question was at its most explosive point, and measured 21 cubic feet, its ignition might be expected to produce $21\times 10=210$ cubic feet of flame. This is not, however, the case in practice, since the mixture of gas and air is not uniform, but varies from nearly pure gas at the roof, near the face, to a mixture containing but a small percentage of gas at a point farther away. At the point where the gas is first detected there is probably 3 per cent. of gas, but there is much gas below this point, so that it is not possible in such a case to estimate the volume of flame that would be produced by the ignition of the gas with any degree of accuracy.

QUES. 5.—Suppose there are 10 men and 22 mules in the mine, and on your arrival in the morning you discover the tower on fire and smoke going down the shaft; the fan is running at a speed of 100 revolutions per minute. What would you do? **Q. 9, April 12.*

ANS.—Get word at once to the men, by phone or by messenger. At the same time, give orders to reverse the fan; or if this cannot be done, short-circuit the air-current by setting open the doors at the bottom of the shaft. In the meantime, do everything possible to get water on the fire; use the chemical extinguishers, which should always be at hand. When the safety of the men is insured give attention to the mules.

QUES. 6.—The long hand on an anemometer makes 3.25 revolutions in 1 minute; the airway where this reading is taken is 6.25 ft. × 8.5 ft. What is the quantity of air traveling, allowing 3 per cent. for the resistance of the instrument?—**Q. 11, April 12.*

ANS.—Assuming this is an average reading, the quantity of air passing is $(1.00-.03)\ 3.25\times 100\ (6.25\times 8.5)=16{,}747+$, say 16,750 cubic feet per minute.

QUES. 7.—If the temperature in a mine is 60° F., and the average velocity of the air-current 325 feet per minute at a point where the entry is 6.25 ft. × 8.5 ft., what weight of air is traveling per minute in the airway?—**Q. 13, April 12.*

ANS.—The weight of air passing per minute assuming a barometer reading of 30 inches is

$$325\ (6.25\times 8.5)\ \frac{1.3273\times 30}{460+60}=1{,}322+\ \text{lb.}$$

QUES. 8.—A current of 9,000 cubic feet of air per minute is passing through a regulator having an opening 25 in. × 35 in. How much air will pass through the regulator if the opening is enlarged to 45 inches square, the pressure and rubbing surface (and area of the airway) being the same in each case?—**Q. 15, April 12.*

ANS.—The question is not definite in regard to the pressure that remains the same when the opening is enlarged. If the pressure due to the regulator is the same in each case, the quantity of air passing will be increased in the same ratio as the area of the opening is increased, or $\frac{45\times 45}{25\times 35}=\frac{81}{35}$; and the quantity of air passing will then be $9{,}000\times\frac{81}{35}=20{,}828+$ cubic feet per minute. In that case, however, the ventilating pressure, which is the sum of the pressure due to friction in the airway and that due to the regulator, would have to be increased to maintain a constant pressure at the regulator.

The question probably intends to make the total ventilating pressure constant. In this case, as the opening in the regulator is enlarged the pressure due to the regulator decreases, which causes more air to flow through the entry and incidentally increases the friction and the pressure required to overcome friction. Without knowing the sectional area and rubbing surface of the airway, it is not possible to determine the quantity of air that will pass through the larger opening under the same ventilating pressure. The original area of the opening in the regulator was $(25\times 35)\div 144=6.076$ square feet; and the enlarged area is $(45\times 45)\div 144=14.062$ square feet. Then, assuming the sectional area of the airway as 50 square feet, and the rubbing surface as 100,000 square feet, the quantity of air that will flow through the larger opening, under the same ventilating pressure, is found by the formula

* Mine Examiners' Examination.

$$\frac{q_2}{q_1}=\frac{A_2}{A_1}\sqrt{\frac{a^3+.026\ s\ A_1^2}{a^3+.026\ s\ A_2^2}}$$

or in this case,

$$q_2=9{,}000\left(\frac{14.062}{6.076}\right)\sqrt{\frac{50^2+.026\times 100{,}000\times 6.076^2}{50^2+.026\times 100{,}000\times 14.062^2}}$$

= say, 12,250 cu. ft. per min.

Ques. 9.—If an air-current passing in a mine is divided into two or more equal splits, will the resistance of the mine be diminished or increased, or does it remain the same after splitting as before, the power upon the air being constant? Explain fully.—†*Q. 4, July 12.*

Ans.—Considering primary splits only, or splits starting from one point in the mine, if the power on the air remains constant at this point, the velocity will be unchanged by splitting the current, because the rubbing surface is constant. This is easily seen by observing the formula for power, $u=k\ s\ v^3$. Also, the mine resistance is not altered, in this case, as shown by the formula for resistance $R=k\ s\ v^2$. In all cases, however, where the air-current travels through a deep shaft or main airway before reaching the point of split, if the power is constant at the mouth of the shaft or main airway, the effect of splitting the air-current at some point in the mine is to increase the quantity of air in circulation. The result is an increase of velocity in the shaft or main airway and likewise an increase in resistance here. As a result the power on the air at the point of split is decreased, which causes a decrease in velocity and resistance in the splits. The decrease of resistance in the splits is always greater than the increase of the same in the shaft or main airway, so that in this case splitting the air produces a decrease in the mine resistance. This is always true for all secondary splits.

Ques. 10.—A mine is ventilated by a furnace that consumes 5 tons of coal each 12 hours. The water gauge gives a reading of 1.5 inches. What will be the increase of pressure caused by increasing the amount of coal consumed to 10 tons each 12 hours?—†*Q. 5, July 12.*

Ans.—In furnace ventilation, the power developed is proportional to the coal burned per hour; and, for the same mine or airway, the square root of the pressure varies as the cube root of the power. Hence, in this case, since the coal burned per hour is doubled, the fuel ratio is 2, and the power ratio is also 2. Then, calling the pressure ratio x, we have $\sqrt{x}=\sqrt[3]{2}=1.26$; and $x=1.26^2=1.58+$. If the water gauge reads 1.5 inches the pressure due to burning 5 tons of coal in 12 hours, in this mine, is $1.5\times 5.2=7.8$ pounds per square foot; and the pressure caused by burning 10 tons of coal in 12 hours, in the same shaft, would be $7.8\times 1.58=12.32+$ pounds per square foot. The increase in pressure is therefore $12.32-7.8=4.52$ pounds per square foot.

Ques. 11.—A current of 50,000 cubic feet of an explosive mixture of air and marsh gas is passing in a mine per minute; what is the quantity of gas given off?—†*Q. 6, July 12.*

Ans.—The question as it reads is unintelligible, as the explosive range of marsh gas extends from 7.14 per cent. to 16.67 per cent. of gas in the firedamp mixture when no other gases are present. The presence of carbon dioxide in the firedamp decreases this explosive range, while the presence of dust or carbon monoxide increases the same. Assuming, however, that the question refers to pure marsh gas mixed with air in such proportion that the firedamp mixture is at its most explosive point, or contains 9.46 per cent. of gas, the quantity of gas carried in this air-current, and which is being generated per minute in the mine, is $50{,}000\times .0946=4{,}730$ cubic feet.

Ques. 12.—What do you understand by ascensional ventilation?—†*Q. 7, July 12.*

Ans.—The ventilation of a mine is ascensional when the air is conducted to the lowest point of the workings in an inclined seam first, and from there continues, in general, to rise in its circulation throughout the mine. As far as practicable the circulation in every mine should be so arranged, in order to take advantage of the air-column due to the natural heat of the mine wherever there are rise or dip workings.

Ques. 13.—What is the velocity of air under the following conditions: water gauge 1.577 inches, barometer 29.5 inches, temperature 64° F.?—†*Q. 8, July 12.*

Ans.—From a practical mining standpoint this question is wholly unintelligible, since the velocity of air-currents in mines depends on data not given and is not appreciably altered (except in furnace ventilation) by the barometric pressure or the temperature of the mine. In mine ventilation, velocity depends on the power on the air and the resisting power of the mine as determined by the relation of its rubbing surface to the area of passage; or, in other words the mine potential $X=a\sqrt{\frac{a}{k\ s}}=\frac{Q}{\sqrt{p}}$; whence $Q=X\sqrt{p}$, or $v=X\frac{\sqrt{p}}{a}$. When the water gauge or the ventilating pressure, or power on the air is known, and the size and length of the airways are given, the velocity can be calculated, but these data are necessary in mining practice. The flow of air into a vacuum under a given pressure is calculated by the formula

$$v=\sqrt{2\ g\ h}=\sqrt{\frac{2\ g\ p}{w}}$$

in which, v = velocity of air (feet per second);
g = force of gravity (32.16 feet per second);
p = pressure on air (pounds per square foot);
w = weight of air (pounds per cubic foot).

This formula, however, cannot be used in this case, because the mention of water gauge precludes the idea of a vacuum, as the water gauge always measures the difference between two pressures, or the ventilating pressure. The question cannot be answered.

Ques. 14.—A fan is producing 150,000 cubic feet of air per minute when the mine resistance produces 1.5 inches of water gauge. If the area of inlet of this fan is 100 square feet and the area of the discharge orifice 70 square feet, what is the manometric efficiency of the fan?—†*Q. 9, July 12.*

Ans.—In order to calculate the so-called manometric efficiency of this fan under the given conditions, it would be necessary to know the depression that causes the air to enter the fan, or that which expels the air from the fan, either of which are not given. It has been suggested that the pressure created by the action of a centrifugal fan may be divided into three distinct pressures; namely, the pressure required to blow the air into the fan, that required to blow the same air out of the fan, and that caused by the mine resistance. Since the pressure due to the mine resistance is the effective pressure, the ratio of this pressure to the sum of all the pressures is what has been called the manometric efficiency of the fan. The ratio of the blowing-in to the blowing-out pressure is equal to the square of the ratio of the discharge to the intake area. Hence, it is necessary to know one of these pressures, from which the other can then be calculated, and the manometrical efficiency of the fan determined. Without this, however, the question cannot be answered.

Note.—The manometric efficiency is clearly of no practical value, since it only relates to pressure (not power), and that in reference to a particular mine or circulation where a given quantity of air is passing under a certain pressure. It is practically impossible to measure either the blowing-in or blowing-out pressure required to calculate manometrical efficiency. Again, since it is the mine resistance that causes the pressure it is evident that the same fan working on another mine or airway will present an entirely different efficiency, depending upon the resisting power of the mine. The true efficiency of a fan is the mechanical efficiency which is expressed by the ratio of the power on the air to the indicated power of the engine, making due allowance for the power absorbed by the engine itself.

Ques. 15.—There is 150,000 cubic feet of air per minute passing through an airway 8 ft. × 10 ft.; what would be the

† Mine Managers' Examination.

horsepower of the circulation apart from friction? The temperature of the air is 60° F., barometer 30 inches.—†*Q. 10, July 12.*

ANS.—Ignoring friction in the circulation of an air-current in a mine or airway, there is no pressure, and therefore no power; no work is done aside from the work of overcoming friction.

QUES. 16.—What must be the area of an airway to pass 10,000 cubic feet of air per minute, with a water gauge of 1.7 inches, the length of the airway being 5,000 feet.—†*Q. 12, July 12.*

ANS.—The question should state the form of the airway, as being circular, square, or rectangular; and if rectangular the ratio of the height to the width of the airway should be given; otherwise the question cannot be worked. The following formulas apply to each one of these types of airways:

Circular airway $a=\sqrt[5]{4\pi\left(\frac{k\,l\,q^2}{p}\right)^2}$

Square airway $a=\sqrt[5]{16\left(\frac{k\,l\,q^2}{p}\right)^2}$

Rectangular airway $a=\sqrt[5]{\frac{4(1+r)^2}{r}\left(\frac{k\,l\,q^2}{p}\right)^2}$

For a circular airway 5,000 feet long the sectional area would be 27.62 square feet; for a square airway, 28.95 square feet; and for a rectangular airway in which the width is twice the height of the airway the area would be 29.68 square feet. The diameter of the first airway would be 5.9 feet; the side of the second (square) airway would be 5 47 feet; and the rectangular airway would be 3.85 ft. × 7.7 ft.

QUES. 17.—There is 20,000 cubic feet of air per minute passing into a mine, which becomes charged with 2.5 per cent. of marsh gas (CH_4); what will be the volume of the return current?—†*Q. 13, July 12.*

ANS.—The meaning of this question would be more clearly expressed if it asked, "what will be the volume of the return current if charged with 2.5 per cent. of gas, the temperature remaining the same?" The gas is 2.5 per cent. of the return current, whereas, the reading of the question would lead one to think it was 2.5 per cent. of the intake volume. The gas being 2.5 per cent., the air (20,000 cubic feet) is $100-2.5=97.5$ per cent. of the return current, which is therefore $20{,}000\div.975=20{,}512+$ cubic feet, at the same temperature.

QUES. 18.—Find the motive column that would produce a pressure of 3,906 pounds when the temperature of the upcast is 70° F. and that of the downcast 42° F., the barometer being 30 inches.—†*Q. 14, July 12.*

ANS.—This question is wholly unintelligible. It is difficult to understand what is the true meaning. A motive column in mine ventilation is an imaginary air column, 1 square foot in section and of such height that its weight will produce a given pressure, which is stated in pounds per square foot. The pressure, 3,906 pounds, is too great to be understood as being the pressure per square foot. It is evidently the total pressure due to the difference in weight of air in the two shafts. But the size of the shafts is not given and without this and the depth of the shafts the question cannot be worked. Moreover, the question should state whether the height of the motive column is to be found in terms of the upcast or the downcast air, as the answer will be different in each case The formulas for height of motive column are as follows:

Downcast air, $M=D\,\frac{T-t}{460+T}$

Upcast air, $M=D\,\frac{T-t}{460+t}$

QUES. 19.—If on a pitch of 25 degrees there is a pitch distance of 300 feet between two seams of coal, what would be the pitch distance if the pitch (of the connecting slope) was increased to 55 degrees?—†*Q. 17, July 12.*

ANS.—Assuming the two seams of coal are horizontal, the vertical distance between the seams is $300\times\sin 25°=300\times.42262=126.786$ feet. The pitch distance then for 55 degrees would be $126.786\div\sin 55°=126.786\div.81915=154.7$ feet.

QUES. 20.—With the entry running due east and rooms running N 45° E, what distance apart (center to center) would you mark off rooms on the entry in order to provide for a width of 30 feet in the rooms, and pillars 15 feet wide?—†*Q. 18, July 12.*

ANS.—The distance between room centers, measured on the entry should be $(30+15)\div\sin 45°=45\div.7071=63.64$ feet.

QUES. 21.—What is the angle included between two lines whose bearings are, respectively, N 47° W and S 32° W?—†*Q. 19, July 12.*

ANS.—The included angle is $180-(47+32)=101°$.

QUES. 22.—How tightly should props be wedged when first stood in a room or entry to support the roof?—†*Q. 20, July 12.*

ANS.—The wedging should be only tight enough to hold the post in position and prevent its being accidentally knocked out or displaced by an ordinary jar. This will allow for the initial and irresistible settlement of the roof due to the extraction of the coal and which will tighten the post without breaking it.

QUES. 23.—In wedging a post in a room with a cap piece sharpened 1 inch in 12 inches, or one sharpened 3 inches in 12 inches; which of these would cause the greater pressure on the post?—†*Q. 21, July 12.*

ANS.—The cap sharpened 1 in 12 would cause a greater pressure on the post than the one sharpened 3 in 12, because it is a sharper edge and is more easily driven up.

QUES. 24.—How much blasting powder would you use in a dead hole that was 2 feet on the solid?—†*Q. 24, July 12.*

ANS.—None. The hole is a dangerous hole.

COST OF MINING AND MARKETING ALASKA COAL

The Bering River and Mantanuska fields of the Pacific coast province furnish high-grade steaming and coking coals as well as anthracite, but these fields have not yet been reached by railways, and these will require large investments of capital. The conditions in both fields are in many ways similar. The Mantanuska field will probably have a slight advantage in a lower cost of mining, but this advantage will be more than offset by a greater railway haul. The bituminous coal of the Bering River field can probably be mined for about $2 a ton, and when a railroad to tide water is built should be delivered at Seattle for little more than $4 a ton. The anthracite of this field can probably be delivered at Seattle for $5 a ton. These coals could probably be delivered at Oregon and California ports at an additional cost of not more than 50 cents a ton.—*United States Geological Survey.*

The Heroult electric iron smelter has demonstrated its commercial practicability and is producing about 20 tons of high-grade pig iron per day in California. The Noble Electric Steel Co. has decided to add five more furnaces to the plant, to bring the production up to 100 tons per day. The smelter has demonstrated its ability to produce pig iron at a total cost of $12 per ton, and, as eastern pig iron costs about $30 per ton on the Pacific coast, the success of the Heroult plant means considerable to the industrial enterprises in California. Deposits of iron ore have been discovered in Shasta and other counties, but the absence of coal has prevented the production of pig iron. Now that it can be produced electrically, the iron-ore beds will receive attention. The Noble Electric Steel Co. is mining ore at a short distance from the smelter, and sufficient ore is developed to insure the operation of the enlarged plant for a long term of years. The company has booked extensive orders for the product, most of it going to San Francisco iron works.

BLASTING SUPPLIES

*By F. H. Gunsolus**

Although the explosive naturally demands first consideration when blasting is to be done, the fact is not to be overlooked that it cannot be properly exploded without certain materials and appliances, especially designed for the purpose, known as "blasting supplies." In addition to those articles necessary to develop the energy of a charge of explosive, there are other devices which, although they may not be absolutely requisite, contribute to safety, certainty, and economy in the use of explosives, and these are also included in the category of blasting supplies.

Importance of Proper Apparatus for Firing or Detonating Explosives. Right Methods of Using

The importance of keeping blasting supplies up to the highest standard in every respect cannot be overestimated, for the very best grades cost but a trifle in comparison with the charge of explosives in connection with which they are used. It is poor economy to attempt to detonate an explosive with an inferior article, for it always results in a considerable waste of the value of the explosive. More than a hundred years' experience in the manufacture, sale, and use of all kinds of explosives has taught that the popularity depends entirely on the results which consumers have with them, and with this in view and from a standpoint of "good business," only a high quality of blasting supplies are recommended.

When a high explosive, such as dynamite, is the blasting agent the "electric fuse" is the detonator used when firing by electricity. It consists of a copper shell containing a detonating charge surrounding a fine platinum wire "bridge," which joins the tips of two insulated copper wires of various lengths. The ends of the copper wires are secured in the copper shell by a composition plug which also serves to keep moisture and water away from the charge. The electric fuse is detonated when sufficient electric current is passed through the copper wires and across the platinum bridge to heat the latter to a temperature high enough to ignite the charge surrounding it.

Fig. 1. Series Connection

In certain classes of tunnel or shaft excavating, "delay-action" fuses have been used to advantage. These are made so that a very short space of time intervenes between their ignition by the electric current and their detonation. Two kinds are manufactured: One kind, called first-period delay-action fuses, detonates a little before the other kind, called second-period delay-action fuses, when both are connected in the same blasting circuit. When they are used in conjunction with regular electric fuses, the latter will detonate at the instant the electric current passes through the circuit; the first period delay-action fuses will follow these, and the second period delay-action fuses will detonate a little later. By their use it is possible to fire one section of a blast, sufficiently ahead of the following section, for the material blasted by the charges in the first section to be blown out of the way when the next section is blasted.

* Manager Technical Division E. I. duPont de Nemours Powder Co.

When several charges of explosives are to be detonated at one time with electric fuses, it is necessary (unless the electric fuse wires are long enough to reach between the bore holes) to join them with connecting wire. Insulated copper wire is used for this purpose.

Leading wire, which is also insulated, and in order to reduce resistance is of a larger size than connecting wire, connects the electric fuses in the first and in the last bore hole with the source of the electric current.

The current for electric blasting is sometimes taken from a power or lighting system, but the source of current commonly used is known as a blasting machine. Although manufactured in several sizes and styles, they are usually designed on the same general principle.

For electric blasting in coal mines or other work where not more than three or four shots are to be fired at one time, the pocket battery is used instead of the blasting machine, because it is smaller and more convenient.

In order to have the maximum amount of electric current pass through the electric fuses, all connections must be carefully made and insulated to prevent leakage, insulating tape being used for this purpose.

Owing to the many difficulties generally attending blasting, such as wet or cold weather, water in drill holes, necessity for hurry and so on, it not infrequently happens that electric fuse wires are stripped or broken in tamping; connections are improperly made, poorly insulated and leaky, or the circuit is interrupted in some other way which results in the misfire of the explosives in some or all of the bore holes, causing serious loss and delay. In order to eliminate this trouble as far as possible, a galvanometer has been designed for testing the blasting circuit before firing. It is possible with this instrument to detect a break or a leak of any considerable extent in the blasting circuit; it will also detect leaks of any considerable extent, or serious defects in the electric fuses, rendering it possible to locate the point at which the trouble exists.

The kind of work on which blasting machines are used is largely responsible for rough and careless handling, which often wears them out rapidly. The rheostat is a simple but effective instrument which should be used from time to time to test the capacity of blasting machines, so that there will be no danger of overloading them.

If leading wire is to be kept in good condition and handled easily and quickly a leading-wire reel is necessary.

When blasting powder is the explosive used, electric squibs may take the place of electric fuses. They are made on the same principle as the electric fuses, but cost less, as a heavy paper shell replaces the copper cap of the electric fuse. As the charge in electric squibs does not detonate but burns or flashes, they will not detonate dynamite or other high explosives.

Blasting caps are often used to detonate high explosives, when it is not necessary to fire more than one charge at a time, or when for some other reason electric firing is not feasible. They consist of a copper shell similar to that of the electric fuse which contains the same kind of a charge. The charge in the

blasting cap, however, is not ignited electrically, but by safety fuse on the end of which the blasting cap must be crimped.

Safety fuse consists of a small train of fine grain black powder which forms the core of a rope of hemp, cotton, or tape, generally covered with waterproofing mixture. It is used for detonating blasting caps as described above, or for igniting directly charges of blasting powder into which it carries a spark.

The cap crimper is a very convenient and serviceable little tool, which is used to attach the blasting cap securely to the safety fuse. Some styles are equipped with a fuse cutter.

Most high explosives containing nitroglycerine freeze and become insensitive at temperatures from 45° F. to 50° F., consequently they cannot be used effectively in cold weather, unless they are thoroughly thawed and kept warm until they are loaded in the bore hole. Thawing kettles are used for this purpose. There are several different designs, but all are constructed on the principle of a warm-water jacket surrounding the compartment which holds the explosive.

Tamping bags are paper containers for sand, clay, or other material with which horizontal or pitching bore holes, or uppers are to be tamped. They are also used when making blasting powder cartridges for use in similar bore holes.

Blasting mats are woven mats of rope which are spread on the ground above the bore holes when blasting is done where flying pieces of rock will be dangerous. If heavy charges of explosives are used it is the custom to place logs or railroad ties directly over the bore holes, and the blasting mats on top of these.

Blasting by electricity is generally conceded to be the most effectual and economical system, and to surpass any other in safety, expedition, and certainty. In work where it is possible to blast more than one charge at a time it will nearly always be found advantageous to do so, and this can only be accomplished by electric firing. When several charges are fired simultaneously each tends to help the other, both in turning out and in breaking up the material blasted, with the result that a greater amount of work is done by a given quantity of explosive than if the several charges were fired successively. In addition, it is possible to better protect against water and other causes of misfire the appliances used in electric blasting, thus insuring greater certainty. As delayed explosions or "hang fires" are hardly possible, and as the blaster can always be a considerable distance away from the explosive when it detonates, this system reduces the possibility of accident to a minimum.

No method of blasting in gaseous or dusty coal mines, other than the electrical one, deserves consideration, because in all others the ignition in the open of some burning substance is necessary, even though a device be used whereby the safety fuse or squib can be ignited without exposing an open light or flame in a gaseous place.

It is believed by many authorities that disastrous explosions in coal mines have been caused by a blown-out shot occurring shortly after a number of other blasts have been fired. This cannot happen if the firing is done by electricity, when as many shots as desired are fired simultaneously. In submarine or other very wet work, no other system is feasible. In underground work, where ventilation is not good, burning safety fuse increases the smoke and fumes very materially. It is not uncommon in its use for the fire to break through the side of the fuse and ignite the charge of explosives before detonating the blasting cap, resulting in poor execution and increase in fumes. This cannot occur when the blasting is done by electricity.

The equipment necessary for electric blasting is as follows: Electric fuses, connecting wire, leading wire, blasting machine.

In addition, the following will prove of much assistance and saving: Insulating tape, rheostat, galvanometer, leading-wire reel.

If the explosive used is blasting powder, electric squibs, which are less expensive, should replace electric fuses in the above list.

When the source of the electric current is a blasting machine of the usual type, connect the bore holes in series, Fig. 1, that is, join one wire of the electric fuse in the first bore hole (if necessary, using connecting wire), to one wire of the electric fuse in the second bore hole and the other wire of this electric fuse to one wire of the electric fuse in the third bore hole, and so on until all of the bore holes are connected together with a free electric-fuse wire in the first and the last bore hole, each of which is to be connected by one of the leading wires to the blasting machine.

Blasting machines and pocket batteries are commonly made for series connecting only and connections should not be made in parallel, or any modification thereof, except machines are especially designed for such connections.

When making connections, care must be taken to see that all metal parts joining each other are scraped bright and clean. Another point of particular importance is, that no part of the circuit which is not thoroughly insulated should come in contact with any other uninsulated part of the circuit, or with water, or with wet or damp ground. In order to accomplish this, all bare joints should be covered with insulating tape.

FIG. 2. LOOPED WIRES—WRONG WAY

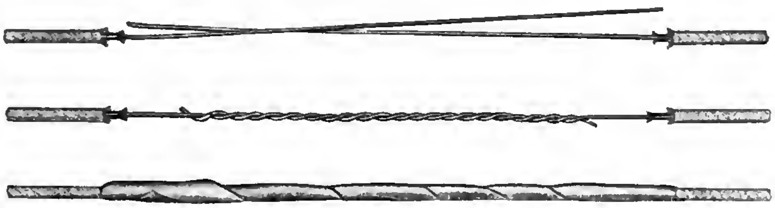


FIG. 3. TWISTED WIRES—RIGHT WAY

When making connections do not loop the wires, but twist them tightly together.

The attempt to use old and damaged leading wire, or connecting wire, is a great mistake and is often the cause of misfires. One of the principal objections to their use is that the wire itself will frequently break inside of the insulation, which will remain intact. When this occurs, the ends of the wire may touch and the circuit appear all right when tested, but a very slight movement of the wires afterwards may pull these ends apart, breaking the circuit, and causing a misfire. A break of this kind is not easily located, and sometimes is responsible for the loss of the time of many workmen waiting for the shot to be fired.

Victor electric fuses are made in four different strengths, No. 4, single strength, No. 6, double strength, No. 7 and No. 8, each of which is put up with insulated copper wires of the following lengths in feet: 4, 6, 8, 10, 12, 14, 16, 18, 20, 22, 24, 26, 28, 30. Special lengths are manufactured to order. The following table describes fully Victor electric fuses of different strengths:

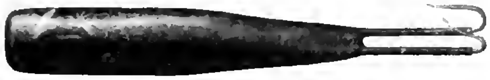
Grade	No. 4 Single Strength	No. 6 Double Strength	No. 7	No. 8
Dimensions of cap	1.25 × .273	1.56 × .273	1.75 × .273	2 × .273
Color of carton	Yellow	Red	Brown	Green

Victor No. 4, single-strength, electric fuses may be used to detonate straight nitroglycerine dynamite, but results are

better with Victor No. 6, double-strength. Nothing weaker than Victor No. 6, double-strength, electric fuses should be used with ammonia dynamite, Red Cross explosives, Semigelatin, gelatin dynamite, blasting gelatin, Judson powder, or the permissible explosives.

The No. 6 double-strength, No. 7 and No. 8 grades, are furnished when so desired for submarine work with a special gutta-percha covering, which is water resisting, Figs. 4 and 5.

SECTION OF GUTTAPERCHA COVERED (FOR SUBMARINE WORK)
Fig. 4



GUTTAPERCHA COVERED (FOR SUBMARINE WORK)
Fig. 5

Fig. 6. Section of Electric Fuse

Fig. 6 shows, in section, an electric fuse. *A* is the shell of copper, having a corrugation thrown out from the inside, which holds the composition plug more firmly in place; *B* is the chamber containing the explosive charge; *C* the copper wires entering the cap, having a cotton covering which is an insulator sufficient for ordinary purposes; *D* the bare ends of the copper wires, projecting through the cement into the charge; *E* the small platinum wire, or bridge, soldered to, and connecting the two ends of the copper wires, which is heated by the electric current; *F* the composition plug holding the fuse wires firmly in place; the bare ends of the wires are tinned to prevent corroding.

Electric fuses are packed in pasteboard cartons which are enclosed in heavy wooden cases. The cartons contain either 25 or 50, depending on the length of the wires. Electric fuses with wires from 4 feet to 16 feet long are packed for domestic trade, 500 to the case, while those with longer wires are packed 250 to the case. The number of electric fuses with wires of different lengths to the case, dimensions, and gross and net weight of case for domestic shipment, are given in Table 2.

Table 2. Electric Fuses

Quantity in Case	Length of Wires in Feet	Gross Weight Pounds	Net Weight In Cartons Pounds
500	4	25	17½
500	6	31	23½
500	8	39	30
500	10	46	37
500	12	54	43½
500	14	61	50½
500	16	68	57½
250	18	45	32½
250	20	48	37½
250	22	52	39½
250	24	55	42½
250	26	58	45½
250	28	61	48½
250	30	64	51½

Outside dimensions of cases are 9½ in. × 17½ in. × 22¾ in.

The storage of electric fuses should always be given careful attention by the consumer. If they are permitted to remain, for a considerable period, in a very warm place, the waterproofing material in the insulation dries out to such an extent that the insulation may break when the wires are bent, and misfires occur if an attempt is made to use them in wet work. The explosive charge in the electric fuses is very easily affected by moisture and if they are stored in a damp or wet place they may deteriorate. As the charge which electric fuses contain is very sensitive, and they may be exploded by a moderately hard knock or jar, they should be handled carefully. Careful handling is also necessary on account of the delicate bridge wire, which may be broken, and when broken it renders the electric fuse absolutely useless. The wires must not be bent sharply, or forcibly separated at the point where they enter the copper cap, as this may break or loosen the sulphur cement and permit water to come in contact with the charge in the electric fuse.

The correct way to prime a high explosive cartridge with an electric fuse is to insert the fuse cap in the center of one end of the cartridge, pointing it directly toward the opposite end, then bring the two wires together up one side of the cartridge, tying them in place with string, an inch or two from each end of the cartridge. The common custom, Figs. 7 and 8, of taking one or more loops or half-hitches around the cartridge with the wires themselves after inserting the fuse cap in a hole made diagonally in the side of the cartridge near one end, is always to be condemned. The principal objection is that the looping of the wires may break the insulation, causing short circuits or leakage of current in wet work, or may even break the wires. In addition, when a fuse cap from 1¼ inches to 2 inches long is pushed into the side of a cartridge 1 inch, 1¼ inches, or even 1½ inches in diameter, it very often occurs that the point where the principal part of the detonating charge is located goes entirely through the explosive, even though it may not penetrate the paper. As it is always the custom when priming in this way to point the fuse cap diagonally toward the end of the cartridge, which will be nearest the outside or top of the charge, it can be readily seen that any pull on the wires hard enough

Fig. 7

Fig. 8

to affect the position of the cap, will tend to bring it more to a right angle with the long axis of the cartridge and thus force the point still farther out of the opposite side. While this does not always cause a failure, it is quite possible that lost shots may be attributed to it, especially when a cartridge of small diameter is used. To make a primer properly with an electric fuse may take a little longer, but the reduction of misfires will more than pay for the trouble.

CORRESPONDENCE

Explosive Mixture

Editor Mines and Minerals:

Sir:—What weight of coal dust and what percentage of marsh gas makes an explosive mixture?

Denver, Colo. Subscriber

Can Water Vapor Be Resolved Into Its Gases in an Explosion?

Editor Mines and Minerals:

Sir:—I would like to know the degree of heat, Fahrenheit, required to resolve steam or aqueous vapor, H_2O, into its original gases? This point of temperature is known as the point of dissociation, and in the study of the effects of moisture upon a mine explosion it would be interesting to know whether or not the moisture or humidity of the air-current plays any important part in the composition of the explosive mixture. The temperature of a firedamp mixture is estimated at about 6,600° F. upon combustion. The question is, whether the hydrogen and oxygen contained in the steam or vapor (H_2O), and mixed with the air of the mine, could combine with any other elements of the same, so as to have an accelerating effect upon the explosion? Could it be possible for the hydrogen of the vapor to become dissociated from its oxygen element in a flame of great intensity in time to combine with something else, such for instance, as the carbon of coal dust, and what would become of the oxygen contained in the vapor in such an event?

Do the resulting gases of an explosion, or the afterdamp, preclude the possibility of such a conclusion?

Montcalm, W. Va. E. W. Bailey

Black Powder

Editor Mines and Minerals:

Sir:—I note the exception taken to my answer to a recent examination question, by "J. S., West Pittston," page 165, October issue. I wish J. S. had given his full name so that I could have written him personally. The reference is to the answer to the second part of Question No. 8, page 52, of the Anthracite Mine Inspector's Examination, Mines and Minerals, August, 1910. The question reads: (*a*) What kind of explosives are used in the anthracite mines? (*b*) What powders are best adapted to various conditions of mining? (*c*) etc.; (*d*) etc. The answer to this question, for lack of space, was not made as full as would be possible in a discussion of this important matter; but was aimed to bring out the chief points of difference between the two general classes of explosives used in coal mining; namely, black powder and dynamite, mentioned in answer to (*a*). The question (*b*) draws attention to the well known "various conditions of mining," with the intent, we presume, of ascertaining a candidate's knowledge of how these two classes of explosives differ the one from the other, and how these differences adapt each class to certain conditions in mining. The question does not require that the candidate subdivide the two general classes of explosives, as such subdivisions are almost endless in their application to various coals.

Powder manufacturers have for several years now been studying to produce powders particularly adapted to coal in different sections. The aim has constantly been to conserve the strength of black powder by uniformity in grinding, mixing, and sorting, and by compression of the mass. Little change has been made in the general formula, but by nitration and other similar means, the various forms of permissible explosives of the black-powder class have been developed. In like manner, also, by varying the dope and the character of the absorbent in the dynamite or nitroglycerine class, the action of this powerful class of explosives has been modified so as to make it of great value in the blasting of the harder coals. In the working of the thin seams of hard bony coal, referred to by our correspondent, "the grade and strength of the powder," as our answer states, "should be adapted to the hardness of the coal." Our correspondent seems to understand the term permissible explosive as describing a different class from the two general types mentioned, whereas the term relates to special powders belonging to each of these classes. We are glad to explain our answer in this respect and hope we have made the same clear.

Scranton, Pa., October 3, 1910. J. T. Beard

The "Howell-White" Furnace

George W. White, son of the inventor of the so-called Howell–White furnace, writes as follows:

"The name was probably copied from the circular and pattern list of the Pacific Iron Works, in 1880, Rankin & Brayton, proprietors. My father, George W. White, of New York City, in 1860 conceived the value of a furnace that would desulphurize, oxidize, and chloridize the base ores, and left for California soon after. A patent was granted him August 23, 1864, also February 7, 1865. A patent was also secured in the Republic of Mexico, 1876. When he reached San Francisco in 1860 he found that base ores were not generally worked, as then there was plenty of free milling ore in sight with placer diggings in plenty. Nothing was done therefore to introduce the furnace until the year 1870, when my father located at the Aetna Foundry, on Fremont Street, where the furnaces were cast, set up, and shipped. The Pacific Iron Works, located nearby, recognizing the value of the White furnace, attempted to secure a one-fourth interest through an agent of the name of Thompson, but before the final payment was made my father learned that they were trying to secure a patent in Mexico without his knowledge. Then suit was commenced against said Thompson and won, and as Thompson was merely an agent of the Pacific Iron Works and had nothing, suit was brought against the Pacific Iron Works and won. It was carried to Superior Court in Equity, and before final decision was reached my father died. At that time mining was on the ebb. Mining-machinery manufacturers were going out of business; the Pacific Iron Works with the rest. My father's lawyer was approached by the Pacific people to settle the suit for $2,000, and as Brayton, the only member of the firm alive, was to make over his possessions to his wife, my lawyer advised settlement. This I agreed to, and the suit, which had been in the courts almost the life of the patent, was settled.

"The name Howell is a myth. Owing to litigation neither the Pacific Iron Works nor my father could collect from the users of the furnace in full, and at the time of his death $60,000 was outstanding which the users of the furnace would not pay, saying that if they did so the other would also try to collect. It seems a pity that the furnace should not be called by its right name—White–Howell Furnace—instead of by the mythical name, Howell–White furnace."

RUST-PROOF FINISH FOR IRON

The so-called "Coslettising" process for producing a rust-proof finish on iron is referred to in a recent issue of the *Brass World*. This process consists in boiling the iron or steel article to be treated in a solution of 1 gallon of water, 4 ounces of phosphoric acid, and 1 ounce of iron filings. By this means a black coating is produced on the iron or steel, which protects it from atmospheric and other corrosive influences. This formula gives good results when care is used, but when carelessly handled a certain amount of undissolved iron filings may be left on the surface of the article being treated. As far as the protection of the coating against corrosion is concerned, it is stated that a piece of steel treated by the process and immersed in salt water for nearly a year has resisted its attacks so that it is practically free from corrosion, while a similar piece untreated has become badly rusted.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

WESTERN EDITORIAL OFFICE: 728 Equitable Building, Denver, Colo.
CHICAGO OFFICE: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to MINES AND MINERALS, Scranton, Pa.
CABLE ADDRESS, "RETSOP, SCRANTON"

RUFUS J. FOSTER...MANAGER
EUGENE B. WILSON, SCRANTON, PA.............................EDITOR
GEORGE F. DUCK, E. M., DENVER, COLO...............WESTERN EDITOR
P. G. MOORE.................................CIRCULATION MANAGER
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., LTD., DRYDEN HOUSE, 43 GERRARD ST., SOHO, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 PITT STREET, SIDNEY.
SOUTH AFRICAN AGENTS: J. AFFLECK ROBERTSON, CAPE TOWN, MANSION HOUSE; JOHANNESBURG, TRUST BUILDING; DURBAN, NATAL BANK CHAMBERS; BLOEMFONTEIN, FLORENCE BUILDINGS.

SUBSCRIPTION RATES

Single Copies..25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada.. 2.50
To all other Countries..................................... 3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always give your FORMER as well as your present address.

TABLE OF CONTENTS

COAL MINING IN OKLAHOMA, by Willis P. Thomas.................... 193
AMERICAN *vs.* EUROPEAN COAL MINES, by Henry M. Payne............ 195
FIRST-AID CONTESTS.. 197
NEW OHIO MINE LAW... 201
EXPLOSION AT PALAU No. 2 MINE................................... 202
MINING ENGINEER... 203
KELLERMAN MINE, by Neill Hutchins............................... 204
A COAL-LOADING MACHINE, by Wm. Whaley........................... 206
TRADE NOTICES... 208
SAMPLING COAL AND COKE, by E. G. Bailey......................... 209
DATA OF PETRODYNAMICS, by R. Dawson Norris Hall.................. 210
COAL NOTES.. 212
RESCUE STATIONS IN ILLINOIS, by R. Y. Williams.................. 214
EXTRA MINE DOORS.. 216
MINE LABORATORY WORK AT GARY, W. VA., by Vitus Klier............ 217
PERSONALS... 218
EXAMINATION QUESTIONS—
Answers to Selected Questions of Illinois Examinations for 1909, Held at Springfield.. 219
BLASTING SUPPLIES, by F. H. Gunsolus............................ 222
CORRESPONDENCE—
Explosive Mixture—Can Water Vapor Be Resolved Into Its Gases in An Explosion?—Black Powder................................ 225
EDITORIAL—
The American Mining Congress—Accidents to Working Men........ 226
BOOK REVIEW... 227
SAG AND STRETCH OF POWER TRANSMISSION ROPE...................... 228
MINING CEMENT GRAVEL AT ALTAR, by Aloysius Coll................. 229
THE AMERICAN MINING CONGRESS.................................... 231
EL COBRE COPPER MINE, by Benjamin B. Lawrence................... 240
CUBAN GOLD MINES, by E. B. Wilson............................... 240
COMBINATION ASSAY OF COPPER BULLION, by S. M. Scott, Jr......... 240
THE PANAMA CANAL, by G. H. Mee.................................. 241
A MEXICAN RIVAL TO THE PANAMA CANAL............................. 244
CUBAN MINING DEVELOPMENT.. 244
IRON ORES OF SANTIAGO, CUBA, by E. B. Wilson.................... 245
THE PEBBLE INDUSTRY OF NEWFOUNDLAND............................. 249
ORE MINING NOTES.. 250
EVOLUTION OF HOISTING, by E. B. W............................... 251
NEW INVENTIONS.. 256
CATALOGS RECEIVED... 256
SQUIBS—
Irrigated Land, 196—Explosives (Permissible), 206—Peat Bogs of Ireland, 212—World's Panama Exposition, 213—Duty on Breccia, 213—Cost of Mining and Marketing Alaska Coal, 221—Electric Iron Smelting, 221—Rust-Proof Finish for Iron, 225—Tungsten Powder, 230—Vanadium Steel, 234—Alaska Mining in 1909.. 250

THE AMERICAN MINING CONGRESS

In common with all meetings of the American Mining Congress held at long distances from the center of population, the attendance at the Los Angeles meeting, just closed, was drawn largely from the surrounding country. While the official figures are lacking, a study of the names of the members present, published in the Los Angeles *Examiner*, indicates that about 50 per cent. of them came from the convention city or within a radius of 50 or 100 miles; some 35 per cent. from other parts of California, from Nevada and Arizona; about 10 per cent. from other states west of Colorado and about 5 per cent. from the farther East. Probably not over 1 per cent. of the attendance was drawn from points east of the Mississippi River, and less than that if the heads of departments and other officials of the Mining Bureau and the United States Geological Survey are omitted. At the last minute many "oil men" came into the convention and took an active part in the proceedings. The policy of admitting oil producers to the Congress is open to discussion and if the expression of opinion voiced by the few members of that calling who spoke publicly is to be followed, they do not care for membership. Although oil is classed as a mineral under our laws, its method of production and handling is so different from any other mineral that the oil producer and miner have little in common except that, for the purposes of acquisition, oil and placer lands are classed together. This unequal distribution of the delegates and members present was not due to any fault of the Congress or the city of Los Angeles, but would have happened had the meeting been held in New York or Philadelphia, and is brought about by the average man having neither the time nor the means to make the round trip of about 5,000 miles required to reach the seat of the convention. The convention at Los Angeles was particularly a western one and largely in the interests of the oil producer; the next, if held in, say, Philadelphia, would reflect the sentiments of the bituminous and anthracite coal operators and of the iron men. Thus, no convention can be considered strictly national in character, and this is contrary to the spirit of the Congress, which is supposed to reflect the opinion of mining men from all sections of the country.

But notwithstanding this, the Congress annually places itself on record as reflecting the sentiment of all the mining men, whereas the fact is that each convention properly reflects the opinion of those nearest the city in which that particular convention is held. It is evident that something should be done to give the deliberations of the American Mining Congress more of a national and less of a local color. It has been suggested that each state in which the mining industry is of sufficient importance to warrant national recognition, shall organize a local chapter which shall elect one delegate to the Congress. This delegate shall be instructed to cast the vote of that state upon all questions of suggested legislation, and by requiring all resolutions to be

presented to the central resolutions committee sufficiently far in advance of the national meeting, each state and its delegate will be able to vote intelligently upon, at least, the most important questions likely to come before the Congress. While members would be welcome at all times and should attend the national convention, they would not be allowed to vote. This seems to us a most excellent plan as placing the more distant states upon an equality so far as voting power is concerned with those within easy traveling distance of the convention hall, and will tend to stimulate interest in the Congress by bringing together in each state a body of men whose interests are essentially identical. But we would go a step beyond this and give each state, not a single vote, but a vote in proportion to the value of its mineral production as determined by the report of the Director of the United States Geological Survey. In this way the vote of Pennsylvania or Colorado would be larger than that of Alabama or New Mexico. By limiting the voting power to those appointed by the state chapters and proportioning the delegates according to the importance of each state as a producer of metals or coal, the affairs of the Congress would be conducted with more dignity and be calculated to inspire more respect as the sentiment of the mining men of the entire nation, and not as now, of those who by nearness to the place of meeting are alone able to attend.

Had some such system as the above been in force at the date of the Los Angeles meeting it is not believed that the Congress would have placed itself on record as opposed to conservation. It is our opinion that a strictly representative congress would have indorsed the opinions of Doctor Pinchot and not repudiated them as did the session at Los Angeles. Engineers are by training unconscious conservationists and we are sure that those who are members of the American Mining Congress will not feel pleased, to put it mildly, at being placed in the position of repudiating themselves.

So much has been said for and against conservation that it is useless to repeat the arguments in the case. There is a strong and growing national sentiment in favor of true conservation, and, as not infrequently included therein, a revision of the federal mining laws whereby all the citizens may profit from the exploitation of the mineral resources.

ACCIDENTS TO WORKING MEN

THE American Museum of Safety, with exhibit hall for accident prevention in the Engineering Building, 29 West 39th Street, New York, issues a monthly bulletin. In the September issue Dr. Ira S. Wile, New York, in an article on "Accidents and the Surgeons," states that while the figures on the prevention of industrial accidents are scarce, it is noteworthy that the investigation of the State Commissioner of Labor in Minnesota revealed the startling fact that 60 per cent. of the killed and injured were working men under thirty years of age. The same Commissioner of Labor declares that one-half of all industrial accidents are preventable. The waste of human life at the period of youth, vigor, and enthusiasm, demands correction. Waving aside all figures showing the economic loss to the working men, their families and the community, losing sight of the immense loss through liability insurance and the costs of damage suits, even dismissing from thought the actual cost in hospital and dispensary expenditures, surgical fees and surgical supplies, it is not too much to ask for the sake of the working man that adequate protection be given to lessen his surgical burden of preventable accidents.

Prevention of accidents is as essential to the welfare of the nation as the prevention of tuberculosis, yellow fever, or typhoid fever, which are occupying the attention of sociologists and humanitarians.

IN the last issue of Mines and Minerals the article appearing under the title "Power Production at Collieries," should have been credited to W. S. Meyers, and not Howard N. Eavenson.

BOOK REVIEW

Esperanto Grammar. Arthur Baker, 700 East 40th Street, Chicago, Ill., has prepared 100,000 brief grammars of Esperanto, and to permit every thinker an opportunity to judge as to the merits of the language, will send one free to any one sufficiently interested to ask for it, enclosing stamp for reply.

Twentieth Century Sheet-Metal Worker is a modern treatise on modern sheet-metal work. The book, which costs $1, bound in cloth, has 86 pages but no index. It contains 76 diagrams explanatory of the sheet-metal workers' art, and numerous practical examples. The book is published by the American Artisan, Chicago, Ill.

Practical Data for the Cyanide Plant, by Herbert A. Megraw, 93 pages, 4¼ in. × 6¾ in., flexible cover, price $2. Published by the McGraw-Hill Book Co. The inspiration for the publication of this volume was in the realization that, while theory and practice of the cyanide process has been ably explained in the standard works on the subject, no attempt has yet been made to collect the practical data, formula tables, usual methods, etc., in one small and convenient volume which might be carried about by the "Man on Shift." As a $2 proposition the book is a joke. E. B. W.

The Third Edition of "Economic Geology," by Heinrich Ries, Professor of Economic Geology at Cornell University, has been issued by the MacMillan Company. The preface states: "Although it is but little more than 4 years since the first edition of this work appeared, our knowledge of the subject of economic geology has expanded to such an extent it was deemed advisable to make a somewhat complete revision of the book." The book is an interesting one on the subject and furnishes book references on the minerals described. There are 589 pages, 237 figures, and 46 plates. The method followed in treating the subjects is as follows: Properties and Occurrence; Analysis; Distributions; Origin; Uses; Production; Bibliography. The price of this book is $3.50. E. B. W.

United States Geological Survey Publications, Washington, D. C.: Bulletin 425, The Explosibility of Coal Dust, by George S. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz; Bulletin 427, Manganese

Deposits of the United States, by Edmund Cecil Harder; Bulletin 430-H, Contributions to Economic Geology, Phosphates, by H. S. Gale, R. W. Richards, and Eliot Blackwelder; Bulletin 432, Some Ore Deposits in Maine and the Milan Mine, New Hampshire, by William H. Emmons; The Production of Graphite in 1908, by Edson S. Bastin; The Production of Glass Sand, Other Sand, and Gravel, in 1909, by Ernest F. Burchard; The Cement Industry in the United States in 1909, by Ernest F. Burchard. Professional Paper 68, The Ore Deposits of New Mexico, by Waldemar Lindgren, Louis C. Graton, and Charles H. Gordon.

The Journal of the Iron and Steel Institute, No. 1, 1910, Vol. LXXXI, edited by George C. Lloyd, secretary. Published by Spon & Chamberlain, 123 Liberty Street, New York, N. Y.

Tenth Annual Report of the Geological Survey of Michigan for the Year 1908, Alfred C. Lane, State Geologist, Lansing, Mich.

Twenty-Eighth Annual Coal Report of the Illinois Bureau of Labor Statistics for 1909, David Ross, Secretary, Springfield, Ill.

Colorado Geological Survey, Bulletins Nos. 1 and 2, R. D. George, State Geologist, Boulder, Colo.

North Carolina Geological and Economic Survey, Joseph Hyde Pratt, State Geologist, Chapel Hill, N. C. Economic Paper, No. 19, Forest Fires in North Carolina During 1909, by J. S. Holmes, Forester.

Eighteenth Annual Catalog of the University of Idaho, 1909-1910, With Announcements for 1910-1911. Published by the University of Idaho, Moscow, Idaho.

Thirty-First Annual Report of the President and Officers of the New York, Ontario & Western Railway Co., With Statement of Accounts for the Fiscal Year Ending June 30, 1910, Richard D. Rickard, Secretary and Treasurer, New York, N. Y.

Proceedings of the Colorado Scientific Society, Denver, Colo. Grinding Wheels—Grinding Machinery—Grinding, by C. H. Norton; Alaska Agricultural Possibilities, by Levi Chubbuck.

Paint Manufacturers' Association of the United States, Philadelphia, Pa. Bulletin No. 29, The Properties and Structure of Certain Paint Pigments.

Report of the Department of Mines of Pennsylvania, Part II—Bituminous, 1909, James E. Roderick, Chief of Department of Mines, Harrisburg, Pa.

Canada Department of Mines, Mines Branch, Ottawa, Canada. Summary Report of the Mines Branch for the Calendar Year Ending December 31, 1909.

New Zealand Geological Survey, J. M. Bell, Director, Wellington, New Zealand. Bulletin No. 9 (New Series), The Geology of the Whatatutu Subdivision, by James Henry Adams.

Department of Mines, W. Dickson, Secretary, Melbourne, Victoria, Australia, Annual Report of the Secretary for Mines to the Hon. P. McBride, M. P., for the year 1909; Memoirs of the Geological Survey of Victoria, E. J. Dunn, F. G. S., Director.

Annual Report of the Department of Mines, New South Wales, for the Year 1909, Sydney, New South Wales, Australia.

Final Report of the Mining Regulations Commission, by F. E. T. Krause, LL. D., M. L. A., Chairman, Charles Porter, M. D., Medical Officer of Health, Johannesburg, and Alexander Heymann, M. A., Consulting Analytical Chemist and Metallurgist, Transvaal, South Africa.

New York State Museum, John M. Clarke, Director, Albany, N. Y. Education Department Bulletin No. 476, The Mining and Quarry Industry of New York State, by D. H. Newland.

The Logical Point, An Illustrated Monthly for the Advancement of the World's Panama Exposition, New Orleans, 1915. Published by the Publicity Department.

Quelques Mots sur la Question des Poussieres au Congres de Dusseldorf, 1910, by Victor Watteyne, Brussels, Belgium.

The Philippine Journal of Science, Nos. 2 and 3. Published by the Bureau of Science, Manila, P. I.

The Minnesota Iron Ranges, is the subject of an article by G. O. Virtue, Ph. D., published in Bulletin No. 84 of the Bureau of Labor, Department of Commerce and Labor. This article is a study of the iron ore mines of Minnesota, and deals with the history of the development of the mines, the amount of ore produced, and the transportation facilities, together with a more detailed discussion of the economic condition of the employes of the mines, a large percentage of whom are of foreign birth. According to figures furnished by the Oliver Iron Mining Co., the principal producer of iron ore in Minnesota, only 23.2 per cent. of the employes of that company, on June 1, 1909, were Americans. The principal foreign-born employes numerically were Austrians, who constituted 32.5 per cent.; Finns, 15.9 per cent.; Italians, 11.1 per cent.; and Scandinavians, 10.2 per cent. Tables are given for 1907 showing the conjugal condition of the foreign-born employes, length of residence in the United States, ability to speak English, and number naturalized. Only 48.6 per cent. of the foreign-born employes could speak English, and 42.9 per cent. of those who had been in the United States 5 years were naturalized; of those over 21 years old reporting their conjugal condition, 51.6 per cent. were married.

The article discusses the characteristics of the various nationalities employed, wages and cost of living, housing conditions, home ownership, educational facilities, and labor organizations. The various methods of mining the ore and the working conditions are described, and statistics are given of fatal and nonfatal accidents, with a discussion of their causes and the provisions made for mine inspection. A brief account is also given of the hospital service and the aid funds and insurance systems provided by the companies.

SAG AND STRETCH OF POWER TRANSMISSION ROPE

The sag and stretch of manila rope cannot be determined to any exact amount, as it varies with the load carried, the speed run, and the initial tension in the rope. In the multiple system, when slack comes on top, the rope is allowed to sag until parts running in opposite directions would nearly touch. It then has to be shortened. The sag on tight side may be determined quite closely if the exact power to be carried is known. Likewise, in the continuous system, where it is possible to keep a uniform tension on rope, the sag may be closely approximated. In general, a rope should be allowed to run as slack as possible without slipping.

The sag of rope takes nearly the form of a parabola. The amount may be approximated by multiplying the weight of 1 foot of rope by the square of the distance from sheave to sheave where the rope leaves, and dividing by eight times the tension on rope. For example, 2-inch diameter rope with supports 90 feet apart, running at 4,200 feet per minute, $\frac{1.34 \text{ pounds} \times 8,100}{8 \times 800} = 1.7$ feet sag on driving side, which would be the same at all speeds. The slack side sag may be determined after the same method, by adding to the initial tension the tension from centrifugal force. Thus, with the same rope as before $\frac{1.34 \times 8,100}{8 \times 199 \text{ pounds initial} + 205 \text{ centrifugal tension}} = 3.3$ feet.

A new rope will stretch considerably for a short time, but after the parts of the rope are firmly bedded together further stretch is slow. The total stretch should not exceed 5 per cent. under reasonable conditions. In cutting the slack out of continuous drives, it must be borne in mind to allow some movement of tightener sheave to guard against shrink caused by atmospheric conditions, and also to allow for future resplicing.

MINING CEMENT GRAVEL AT ALTAR

Written for Mines and Minerals, by Aloysius Coll

Old and New Methods of Recovering Gold from Cemented Gravel in Absence of Water

The success of the recently invented Quenner dry pulverizing machine in the treatment of the gold cement in the Altar gold district of Western Sonora, Mex., has revived operations in a field which was worked more than 100 years ago. Strangely enough, after numerous attempts to mine the conglomerate at a profit, the methods now adopted go back to those employed in the days of the district's supremacy.

Following the discovery of gold in the Altar district by a band of soldiers in 1799, thousands of people flocked into the region from Hermosillo, Chihuahua, Durango, and Sinaloa, establishing camps at Cienega, San Francisco, and other places. Altar is the most arid province in Mexico, and on this account the placers could not often be worked with water. But the ground was so rich in gold that with dry panning, using the Mexican "batea," the operations were profitable. Ten years ago a big stamp mill was erected at El Tiro by Epes Randolph and others, at a cost of $1,000,000, and an attempt was made to extract the gold in the conglomerate by means of the stamp mill, but without profit.

The natives have, year after year, continued to work over the surface of the vast gold field, which extends from a point well up in Arizona to the Yaqui River, in Sonora, rushing into the district after each rainy season, when the gold is found in the top sands.

The richest gold-bearing material is in the lower stratum of the conglomerate, and although gold is almost anywhere, it is only found in profitable quantities in the ancient channels, which are for the most part covered by the wash of centuries of erosion. The field is one requiring prospecting, but when one of these channels is discovered a fortune is in sight. Once found these channels can be followed, whether on bed rock or above, and so far the new operations have failed to discover the petering out of any one of these channel deposits.

The most extensive operations since the success of the dry pulverizer has been established have been those of the American Ore Milling Co., which is working what is known as the Bray lease, a tract of proven ground near Baludo, 60 miles west of Santa Anna, on the Southern Pacific Railroad. A stage line is in operation between the railroad and the camp.

Fig. 1. Monuments for Mining Claims in Mexico

Fig. 2. Old Furnaces at Cienega

The American Ore Milling Co. sank a shaft 130 feet to bed rock, where there is a stratum of gold-bearing cement in a channel that is 3 feet thick and 10 feet wide. The gold in this stratum averages $12 to the ton.

Twenty feet above this is another stratum 20 feet wide and 8 feet thick which averages $10 in gold to the ton. Eight feet above this a third stratum 12 feet thick and 40 feet wide carries $8 in gold to the ton. From the shaft, tunnels follow each stratum. A second shaft has been sunk, up which the material taken from the top stratum is hoisted. The company has now on the dump 14,000 tons of the underground material which averages $8 in gold a ton. This is mixed with the material hoisted from the two shafts, and fed into the dry placer machine, which is now pulverizing 150 tons a day, half from the dump and half from the tunnels.

Labor in this field costs $1 a day gold, and it requires a crew of 60 men and boys to operate one machine and the dry washers used to complete the treatment. The machine eliminates the pebbles and pieces of broken rock, leaving only the dust and black sand with the gold for treatment on the dry washers. It requires six dry washers to handle the pulverized material turned out by one machine. Boys are employed to carry this material from the machine to the dry washers. One man cleans up the dry washers, brushing off the gold and black sand left by the bellows into a common gold pan. The hand panner sits upon a duck sheet and blows off the black sand, which still carries fine particles of gold. It is therefore caught on the sheet and saved for shipment as concentrate.

One of the strange features, is that though the dry-pulverizer had been in operation more than a year, not even those along the borderland in Arizona knew of the operations. As soon as it became known there was the usual rush into the field and during the month of April of this year there were more denouncements made in the Altar district than in all the other districts of Mexico combined. It is now hot in Altar, but the denouncements go on day after day, and the region is swarming with prospectors, who pick out what seems to them promising locations. The Mexican mining laws are so simple and direct that this feature greatly influences activity in the district. When a man finds a piece of ground not already denounced, which he thinks contains gold, he merely has to pay 5 pesos a pertinencia, or unit of mine location, equal to 2.47 acres, or $2.50 in United States currency, and is given his denouncement certificate. He then has a year in which to put up monuments, which must be 50 centimeters square at the base, of

masonry or concrete, and within sight of each other, as in Fig. 1. He may do this work at once, and in 6 months obtain title to the ground. It is then his so long as he pays the taxes upon the land. There is not the same hardship imposed in work upon the claim, as in the United States, nor is there the fear of having the claim jumped because of technical points.

Most of the denouncements so far made have been in and around Baludo, where Quenner has worked his machine, and at Cienega where mining was carried on from 1799 up till 1844. The whole country around Baludo, Cienega, and other villages, is so torn up and gutted out on the surface that a horse cannot be ridden over the ground. It will be understood from Fig. 2 that the district is not new in any sense of the term, and if new mines have not been recorded in years, the loose sand at the surface has been worked summer after summer by the natives. Since the natives had no method of crushing the lower stratum of cement gravel, they contented themselves with screening the surface dirt, as in Fig. 3, and working over the sand and gold particles in it by the dry washer shown in Fig. 4.

Fig. 3. Using Primitive Grizzly Preparatory to Dry Washing

Fig. 4. Primitive Dry Washer Used in Mexico

It is not permitted to work the ground denounced until the expiration of 6 months after putting up monuments, with proper marks, so that there can be no great activity in operations till this fall, by which time most of the ground will have been taken up. Much of the ground has been denounced by persons who have not the capital to install the Quenner plant, simple as it is, and this gives rise to the impression that by the autumn Altar and Hermosillo will witness many stirring scenes in mining deals. Much of the ground is held by men in Douglas and Bisbee, Ariz.

The main body of the machine, Fig. 5, is a screen in the form of a drum which revolves on small wheels set under it, at the rate of 28 revolutions per minute. This drum is made of steel bars 1 inch thick, 2 inches wide, by 6 feet long. The arrangement of the bars, with a small opening between them, makes the drum a revolving screen as well as the body of the pulverizer. Through the center of this drum is a shaft, to which are attached 36 chains of three links each, at the ends of which are suspended manganese hammers. When the machine is in motion these revolve in the same direction as the drum at the rate of 400 revolutions per minute. The hammers, when whirled about the shaft strike a blow of 1,800 pounds and have the advantage over rigid hammers in that if they strike rocks and must give, there is no breakage. The hammers attached to the shaft are set so that they strike edgewise against the material and when new, clear the inside curve of the drum about ½ of an inch. The mill is 6 feet long, and the diameter of the drum 40 inches. It requires 16 horsepower to operate the mill when set to crush to 8 mesh.

One of the successful features of the machine is the arrangement of the chains and hammers on the inside shaft. Quenner's first machine had the hammers arranged in regular order around the shaft, and when the material was fed into the hopper at the end of the drum, the material had a tendency to fly around in circles, with no tendency toward a discharge of the worthless rock in the gold cement. He then solved this problem by setting the hammers in spiral form on the shaft, and these in rotation have the power of passing the pebbles and broken fragments of rock from the feed end to the discharge end of the drum. By reducing the spiral trend, the machine can be made to crush all the material fed into the hopper. The fines drop under the drum, and are then ready for treatment on the dry washers. These are operated by hand. The fines are fed into a hopper, which regulates the flow. The bellows in the dry washers, Fig. 4, blows off the light dust and leaves the black sand and gold ready for hand panning.

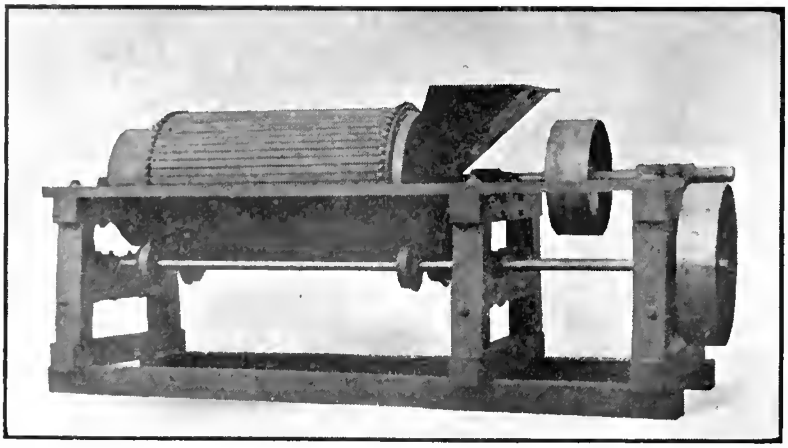


Fig. 5. The Quenner Dry Pulverizer

Tungsten powder, heretofore rarely sent from the Plauen district, was shipped to the United States in 1909 to the value of $34,996. This metal, produced from wolfram ore found in that district, has been used in Germany in a process for hardening steel. It promises to become an important item of export as it is used extensively in American steel works.

THE AMERICAN MINING CONGRESS

Written for Mines and Minerals

Probably the most important meeting ever held by the American Mining Congress was that which commenced September 26, in Los Angeles, and ended October 1.

Report of the Meeting Held at Los Angeles, September 26, to October 1, 1910

Early conventions were not attractive, principally because they were dominated by lawyers, politicians, colonels, boomers, and mining experts who learned their mining from prospectuses and prospectors. Those interested in mining could not afford the time to associate with these people and be regaled with noise. The writer attended one of the early conventions where the effusive outbursts of hot air were possibly never exceeded even at a political convention.

President Buckley in his opening address called attention to former conditions and congratulated the Congress on the beneficial changes that had occurred; hardly had he finished, however, before the lawyers, politicians, corporation henchmen, and promoters commenced the old tactics.

Gifford Pinchot made a few remarks on conservation which were not sufficiently lucid to flag the outbursts of the Antis. He urged that mining men of all classes cooperate with the other people of the country in eliminating waste and control by large interests and in accomplishing the restoration to the people of large areas that through former bad legislation were permitted to slip from the control of the government. He said: "The early grants of the Southern Pacific Railway in the state of California, containing oil, should not permit the people to despair of conservation of the remaining lands of the government. The government will not be doing its duty unless it attacks the title of the Southern Pacific to the lands included in those grants and by legislation or otherwise restores them to the people of the United States."

"Before you condemn conservation find out what the friends of this policy assert and do not condemn upon what the enemies of conservation declare or dispute."

"The fundamental principle of conservation is that natural resources belong to the people of the United States and should be protected for their benefit."

"In the past the few have been benefited. The people of the nation wish to change this condition and they will."

"The rights of state and nation in regard to conservation have been the cause of division among some, but this should not be, for the several states and the nation can work in harmony, as they labored in the past in all great problems concerning the people."

"Every acre should be put to use by the greatest number of people, and excessive holdings by individuals should not be tolerated."

"Settlement of land must be encouraged and the land disposed of in fee simple, but separate from mineral beneath the surface which comes under a different head." (In this clause Mr. Pinchot had reference to agricultural land underlaid with coal.)

"The object of conservation is not to tie up development but to prevent the present generation wasting the natural resources."

"The coming generations have a right to some of the benefits."

"Mining men should accept the proposition and urge it."

"We believe in developing and not in speculation and monopoly."

"The people should supervise regulations concerning the natural resources, and seek to prevent the loss of life, fires, and accidents in mines. That is fair to all mining men."

"The forest service has never attempted to interfere with mining, but as a fundamental fact has endeavored to beat the man who was trying to get control of land under pretense of legitimate mining."

Speaking of the leasing system he said: "Conservation must not and will not interfere with the prospector. Any system that did so would not be sound."

"All that conservationists are trying to do is to make the prospector's work easier, to facilitate and not check his progress."

"Conservation is here to stay whether mining men like it or not."

Mr. Pinchot in his address stated that the attack upon the conservationists is based for the most part upon misunderstanding. The conservationists are working for all the people and the policy of conservation must eventually benefit all the people. He said: "The conservationists want cooperation between state and nation and not discord. Both should work together. He is an enemy to both state and nation who fixes difference between them. The conservationists are striving for the good of all and should receive the support of all the people, and national control or state control should not enter into a controversy. What is needed is both a national conservation commission and a state conservation commission and they should work together in this way. All the resources should be handled equitably and harm will be done to no man."

"Oil lands not now located should remain so. Legislation should be made for the development of these lands, say for 3 years, and if oil is found the government should give a permit to develop for a period of many years, say for 25 years, and the operator should pay a royalty, but small enough not to hamper his development of the land."

Mr. Pinchot, in Pueblo, had more to say on this subject. He said: "There is only one enemy to fight. It is the man in politics to feather his own nest. The loudest cries against the new policy and the most bitter fight against it have come from those individuals and organizations who see their individual profits are in danger."

"The basic principle of this fight of yours and mine is that it is better to help the small man make a living than to help the big man make a profit; to indorse the idea that the natural resources do not belong to a few men, mainly in Wall Street, but they belong, and must be kept, in the possession of the people."

"I believe the contest of states' rights and federal, so prominently before this Congress, would never have arisen if the government had not undertaken to control some of the special interests which believe they could avoid all control by taking refuge behind the states."

E. W. Parker, of the United States Geological Survey, was one of the speakers at Wednesday's session, his subject being "Conservation as it Affects Coal Lands." He voiced the arguments for conservation saying that the people should be protected. He said that in former years coal lands were given away at from $2 to $3 an acre, but that now through the suggestions of the Geological Survey such lands were being sold at from $300 to $400 per acre and this has not retarded development. Following up the argument that the East is seeking to take away that which belongs to the West, he said: "All money that has been obtained thus far from the sale of public lands has been put back immediately into the reclamation service for the development of lands in the West, therefore the West obtains all the benefits of conservation so far as applied and not the East."

Judge Short in his address attacking the policy of conservation claimed in substance that the northern, southern, and eastern sections of the country have had their share of the national wealth, wasted it in riotous living, and now seek to rob the West of its heritage.

Others who opposed the Pinchot policies have also used this extravagant figure.

The chief stumbling block was conservation, and from the various ideas advanced each one had his own. Doctor Buckley said: "There appears to have been some confusion of the

question of conservation with that of ownership of the public domain. They undoubtedly have some relation to each other, but in their fundamental natures they are distinct problems."

"Dr. T. C. Chamberlin, of the University of Chicago, has recently made the point, and rightly, 'I believe that conservation of our natural resources centers in the scientific and technical while the right of ownership and the most desirable distribution of ownership centers in the political and sociological This means that whenever it shall have been determined whether the lands of the public domain shall be owned or leased by the individual, company, or corporation, we will still be confronted with the question of conservation. As far as the conservation of our mineral resources are concerned, it must be conceded that such ownership or distribution of ownership as will result in the least waste, is theoretically the most desirable. Whether the government should use the lands of the public domain as a source of revenue, regardless of whether or not the result should be less waste, is a question involving other fundamental principles for its settlement.'

"Four things appear to be perfectly clear in a consideration of the conservation problem: We cannot afford to have the government enact legislation which will make the occupation of mining more hazardous. That nothing shall be done that will in any way retard the development of our manufacturing industries that depend on the products of the mines for their business. That everything possible be done to increase the percentage of metals that can be recovered from the ore bodies and from deposits of coal, oil, and gas. To bring about as quickly as possible the use of substitutes for the present sources of power and for metals which are supposed to be limited in quantity. If it is desired to bring about a decrease in the consumption of mineral fuels without lessening the growth of our manufacturing commodities, one of two things must happen. Either the efficiency of the mineral fuels must be increased or substitutes for the mineral fuels must be made available.

"The waste in mining is said to approximate 50 per cent. This waste, however, cannot be decreased beyond a point where mining becomes profitable. Under present conditions of the industry there appears to be five ways in which this waste may be lessened; viz., by improved methods of mining, by decreasing the profits; by increasing the price of coal to the consumers; by reducing the price of labor; and by securing markets for the poorer grades of coal."

On Tuesday morning a list of resolutions was offered as follows: Charles F. Potter, Colorado, recognizing conservation and providing for a leasing system for the disposition of oil lands; Colonel E. F. Brown, Colorado, providing for more simple means of location of claims in forest reserves, and again a resolution protesting against the segregation of surface and mineral; C. P. Fox, California, recommending that the American Mining Congress refuse to commit itself on any policy of handling public oil lands. The remaining portion of the morning was devoted to a discussion of the Federal Government in the oil industry. S. C. Graham, Los Angeles, strongly favored conservation and believed in such legislation as would secure for the government as large a revenue as possible consistent with the prevention of monopoly. T. A. O'Donnell, Los Angeles, favored a return to the laws that have been in force for the past 30 years, characterizing as ruinous the handling of affairs of the West from Washington by the leasing system. The former was an individual operator on patented land; the latter manager for a large corporation working on unpatented government land.

In the afternoon J. W. Dawson, West Virginia, recommended such changes in the Sherman Anti-Trust Law as would provide for the conservation of coal lands. George E. Baker, Bakersfield, Cal., urged that the Mining Congress take action on the subject, set forth the injustice of recent decisions by the land office, and express the belief that the situation could be cleared by simple amendments of present laws governing the disposition of oil lands. F. H. Short, Fresno, Cal., spoke in favor of leaving public lands open to entry and location. T. E. Gibbon, Los Angeles, spoke in favor of conservation that will give the government the benefit of lands still owned by it, but that will not withdraw land that has already been entered upon in good faith. W. E. Clark, Arizona, read the report of the Committee on the Revision of Mineral Land Laws as follows:

REPORT OF COMMITTEE ON MINERAL LAND LAWS

General Revision of Mineral Land Laws.—Submitted by E. B. Kerby, Chairman; Frank G. Tyrrell, Los Angeles; Will L. Clark, Jerome, Ariz.; W. H. Dickson, Salt Lake City, Utah; M. Baumgartner, Spokane, Wash.

Your Committee on General Revision of the Mineral Land Laws begs leave to report as follows:

Our work has been carried on with some difficulty, owing to the impossibility of having a meeting of the committee. But by telegraph and mail various matters have been discussed and agreed upon.

With few individual exceptions, the feeling is general among mining men, that a general revision of our mineral land laws is imperative. A generation has passed since the subject was last dealt with in any comprehensive fashion, and conditions and methods have radically changed.

On June 13, 1910, your committee sent the following message to Secretary Callbreath, who was then at Washington, D. C.:

"San Francisco, June 13, 1910.—Committee recommends that our President ask Congress to undertake promptly a general revision of the Mineral Land Laws which, in view of the difficult problems presented, should be in cooperation with the mining industry. The plan adopted for this cooperation should give all sections opportunity for public hearing and discussion of remedies. Mining Congress will suggest a practicable plan later on, if desired."

It was hoped to secure favorable action by Congress at once; this, however, was impossible, but your committe has seen no reason to modify this recommendation, and now reports to your honorable body, again recommending:

That we appeal to the Congress of the United States to undertake promptly a general revision of the Mineral Land Laws; that a Committee or Commission of Congress be appointed, and properly supported to enable them to carry out the work, and instructed to hold public hearing in the various mineral land sections of the country, and cooperate as closely as possible with the people actually engaged in the mining and oil industries in framing amendments, new legislation, etc.; that whatever be the general plan of the Commission, all sections be given opportunities for public hearing and discussion of remedies.

The reasons for this recommendation are:

1. The Mineral Land Laws of the United States and Alaska framed in 1872, and interwoven with a mass of supplementary state legislation, differing more or less in every state, fail to meet the present requirements of the industry. Moreover, evils have developed, the injurious effects of which are steadily increasing. These have become so serious as to create great dissatisfaction and complaint everywhere, and it is confidently believed that a general revision will be welcomed and supported by all the mining communities.

2. Of the numerous evils due to the present inadequate code, those most generally recognized are: The monopolization of valuable mineral land, and its speculative holding, without any effort at utilization, to the consequent exclusion of prospectors and miners. The result is a paralysis of productive industry, and the indefinite postponement of the development and settlement of the country. Among the many points that will come up for consideration in a general revision of the laws are the following:

1. The apex law, with its vexatious uncertainties and costly litigation. Here we shall have to consider, not only the

conflicts due to the extra-lateral right, but also those occasioned by the resulting contours of claims, and the overlapping of lines.

2. The present non-enforcement of the law of discovery.

3. The partial, or complete, evasion, by various ingenious expedients, of the law of assessment.

4. A clear, definite, and practicable procedure for acquiring rights to those mineral claims on which, from the nature of the deposit, discovery must be long deferred.

5. The problem of tunnel locations, and the uncertainties of title caused in neighboring claims.

6. The location of an unlimited number of claims by one person.

7. Location by proxy.

At almost every session of the American Mining Congress the prevailing dissatisfaction has found expression in various resolutions, asking for the correction of this or that or the other feature of the existing law. Such resolutions have been altogether resultless, for two reasons: In the first place, the United States Congress has paid no attention to them. In the second place, those who study the matter closely, find that the laws are interdependent, and it is difficult, and in some cases impossible, to correct one fault without going through the entire code. We must abandon the idea of patchwork, and concentrate our efforts upon the task of securing a general revision.

As the problems involved are peculiarly difficult, a revised code will not be satisfactory, unless it be the result of the most experienced judgment, and of full discussion by the mining communities. It is believed that the National Congress will welcome the active cooperation of the mining and oil men of the country, upon any practicable plan.

Congress must first assent to the general proposition, and agree to undertake a general revision; to do it with the cooperation of the industry and to give all sections an opportunity to be heard upon the problems involved. As to the details of the plan to bring this about, your committee is of the opinion that they can and should be left to the National Congress itself, with the understanding that this body is ready at all times to assist if desired. We further recommend

That the president and other officers of the American Mining Congress, and its members and committees, exercise every effort to secure the prompt attention and favorable action of the National Congress; and

That the people of every mining state and territory be requested to bring the matter before their Senators and Representatives, and enlist their active support.

A motion was carried that the report be referred to the Resolutions Committee.

A resolution approving the holding of a Panama Canal Exposition at San Francisco in 1915, was submitted and adopted.

The resolution providing for the investigation of freight rates on ores was reported favorably. As a part of the discussion on the motion for the adoption of this resolution, T. C. Becker, of Los Angeles, read his address entitled "Railroads and Mining Development." Mr. Becker went fully into the matter of rates and pointed out the injustice of the present basis of charges. He stated that the rates were based on what the carrier thought the traffic would bear, and that long and short haul features were given little thought. The resolution was adopted.

The following is the report of the Committee on Resolutions:

REPORT OF THE COMMITTEE ON RESOLUTIONS

Your Subcommittee on Conservation, having had under consideration the various resolutions referred to it, do respectfully make the following report:

We recommend the adoption of the following resolutions:

As a substitute for Resolution No. 10:

Resolved—That, in common with citizens of the United States engaged in other industries, we approve the theory and practice of true conservation, which means utilization and developing with the least possible waste the natural resources of our country.

We recognize, as men engaged in one of the most important industries of our country, the value of true conservation and its intimate relation to the mining interests, and recommend the enactment of such legislation, both state and national, as will bring about a beneficial development of the mines, the public lands, the public water rights, and the timber contained within our great western country for the best interests of the present and future generations without unnecessary waste.

We condemn, however, as unwise, as opposed to the best interests of the American people, and as wholly unnecessary to the success of any plan of true conservation, legislation or proposed legislation which tends to make the miners and other citizens of the public land states who invest their time, labor, and capital in the development of the natural resources contained within such states, lessees of, or tribute payers to the national government.

We believe that every legitimate means should be adopted in the control of public lands to eradicate or lessen the evils of monopoly, but fail to find in any of the remedies suggested by the advocates of the leasing system how this can be accomplished by changing the present laws so as to take from the citizen a clear title and substitute therefor a lease.

As a substitute for Resolution No. 12:

WHEREAS—The laws relating to forest reserves provide that nothing therein contained shall prohibit any person from entering upon such forest reservations for all proper and lawful purposes, including that of prospecting, locating, and developing the mineral resources thereof, and

WHEREAS—Reports have been made from time to time to this Congress that in the administration of the rules and regulations of the Forest Service in many instances, mining and prospecting have been discouraged within the forest reserves and miners and prospectors have been hampered and interfered with in the exercise of their lawful vocation, and

WHEREAS—Assurances have been made to this Congress by representatives of the Forest Service that in their administration of the laws relating to the public lands under their jurisdiction they will in no manner discriminate against but will aid and assist them in the development of the mineral resources within the Forest Reserves. Therefore,

Resolved—That in reaffirming the right of every prospector who is a citizen of the United States, or who has declared his intentions of becoming such, to enter upon and prospect every part of the public domain, we suggest and recommend a spirit of greater harmony and cooperation between the prospector and miner upon one side, and the officials of the government on the other.

We further recommend that the laws relating to the public domain and particularly that part of the public domain lying within the forest reservations, be administered without burdensome and discouraging departmental rules and regulations and in such a manner as to foster and encourage the mining industry.

We condemn the actions of any class of citizens who go upon the forest reserves, or other public lands, for the purpose of locating fictitious mining claims in order to obtain the timber contained thereon, but we ask Congress to protect the miner and prospector in the development of the mineral resources by the enactment and enforcement of such laws as will give them the right to use such timber and other products of the soil from the public domain as may be necessary in the proper development of their mines.

As a substitute for Resolution No. 30:

Resolved—That, recognizing the right to appropriate to a beneficial use, waters upon the public lands, is fully recognized by Congress and the courts to be wholly governed by the laws of the state wherein such waters are located, and

That the development of mines and the mining industry in many sections of our country is largely dependent upon the

use of water-power, and that there are large amounts of undeveloped water-power now running to waste on the public domain which in the interest of true conservation should be utilized and put to a beneficial use, and

Recognizing further, that the National Government is the owner of large quantities of lands bordering upon and adjacent to streams, the waters of which belong to and are under the control of the states, and that by reason of such diversity of ownership and the different constructions placed upon existing laws relating thereto, water-power development in the West has practically ceased, and

Recognizing further the great expense necessarily incurred in the construction of water-power plants in the mining region, which involves in most cases an expenditure of several million dollars; the uncertain markets for power presented by the mining districts; and the great benefits derived by the mining industry by the construction of such plants,

We therefore recommend that laws be speedily enacted which shall definitely and accurately define the rights of citizens to utilize and put to a beneficial use the waters of the streams and the rights-of-way on public lands adjacent thereto for water-power purposes, and that such laws provide for the concurrent use of the rights of way so long as the water rights are put to a beneficial use under the laws of the state, or for a sufficient length of time to enable those who engage in their development to secure a reasonable return upon their investment.

We believe and therefore recommend that for the purpose of harmonizing all interests and bringing about a proper development of water-power freed from monopolistic influences, all water-power sites upon the public domain should be under the control and supervision of the respective states, wherein said sites are located.

As a substitute for Resolution No. 32:

Resolved—That any conservation policy which places obstacles or restrictions in the way of the free and unrestricted prospecting and location of metalliferous mineral lands, or which favors any lease or royalty upon the future tenure and production of such locations, is inimical to the development of our country's resources, represents a step backward, and strikes directly at the welfare and prosperity of the entire mining industry.

Resolved—That we indorse the work of the Forest Service in its efforts to preserve and maintain a national consumption of the nation's timber resources, but we are unqualifiedly opposed to any and all withdrawals of metalliferous mineral lands from public entry, and to any legislation which will in any way interfere with the free prospecting, location of or production of mining claims.

As a substitute for Resolution No. 33:

Resolved—That this Congress believes that the best interests of all the people will be conserved by state regulation and control of all natural resources within the boundaries of each and every state, in the very largest measure compatible with present Federal statutes.

We have had under consideration Resolutions Nos. 11, 13, and 16 and do respectfully recommend that the same do not pass.

Respectfully submitted,

Charles F. Potter, Chairman, Subcommittee on Conservation.

Edward F. Benjamin, Chairman, Subcommittee on Oil.

The Chairman announced that 10 minutes would be allowed each delegate in discussing these resolutions. Charles F. Potter, chairman of the Conservation Subcommittee, spoke in favor of Resolution No. 10. He reviewed the practice of conservation since the days of early ancient history. He condemned the leasing system; showed that the plan of making the miner a tenant and tribute payer to the government is not a new one, and furthermore that it has been a failure in most of the cases where practiced. He quoted President Polk, who in 1845 condemned the leasing system then in force.

At noon adjournment was taken until 2 P. M.

THURSDAY AFTERNOON

Resolution No. 42 was read, providing for the establishment of government smelters in various mining districts.

The 10-minute discussions of Resolution No. 10 were resumed. Dr. H. Foster Bain, California, speaking in favor of some form of leasing system, pointed out that such systems are now in force, and with great success, in the Mississippi Valley; Missouri has a system of leasing and it does not discourage prospecting. T. W. Hull, Arizona, made a strong appeal for a continuation of the old way of acquiring mineral ground—by location and patent. After remarks by several other delegates, chiefly unfavorable to conservation, Resolution No. 10 was put to a vote and was passed. The substitute for Resolution No. 12 was then taken up and passed without discussion. Substitutes for Resolutions Nos. 30 and 33 were adopted after short discussions.

In the adoption of the report of its Committee on Resolutions the Congress puts itself on record as follows:

While indorsing the policy of conservation in general terms, it is opposed to the leasing of mineral lands by the Federal Government.

Less burdensome restrictions in connection with forest reserves are demanded so that prospectors may have greater freedom in going on to forest reserves to prospect for minerals.

State control of water-power sites on the public domain is urged as tending to more rapid development of the various states.

Withdrawal of metalliferous mineral lands from public entry is condemned.

State regulation and control of all natural resources to the largest degree compatible with Federal statutes is urged.

After the reading of Resolutions Nos. 42, 43, and 44, the Secretary read the report of the Credentials Committee. This was adopted without comment. The report of the Committee on Standardization of Electrical Equipment in Coal Mines was read.

ELECTION OF OFFICERS

The newly-constituted Board of Directors held a meeting at 3 o'clock on Friday afternoon and elected the following officers for the ensuing year: Mr. John Dern, Salt Lake City, Utah, president; Mr. Samuel A. Taylor, Pittsburg, Pa., first vice-president; Mr. D. W. Brunton, Denver Colo., second vice-president; Mr. E. A. Montgomery, Los Angeles, Cal., third vice-president; Mr. James F. Callbreath, Jr., Denver, Colo, was re-elected secretary.

A resolution was passed that the next meeting of the Congress should be held at either Phœnix or Douglas, Arizona.

In the evening the Congress resolved itself into a "smoker," given under the auspices of the Sierra Madre Club, held on the roof of the Hamburger Building.

VANADIUM STEEL

For axles, the superiority of vanadium steel over other alloyed and plain steel, is due to its high elastic ratio (which is the real strength of steel), its wonderful ductility with this increased strength, and its resistance to failure under repeated alternation stresses. The full advantage of vanadium steel is obtained only after heat treating. To demonstrate the ductility of vanadium steel axles: it is possible to bend double without cracking a vanadium axle treated to show an elastic limit of 85,000 pounds per square inch with 19 per cent. elongation in 2 inches, while a plain axle treated to this high elastic limit will invariably break in attempting to bend double. Vanadium bits specially treated for coal-cutting purposes have under actual test cut two-thirds more coal than ordinary cutting bits.

EL COBRE COPPER MINE*

By Benjamin B. Lawrence, New York

It may surprise many to learn that the first copper mined in quantity by civilized man on the Western Hemisphere came from the El Cobre copper mines, at Santiago de Cuba; and much information of historical interest relative to this property may be found in the government archives at Santiago.

History of the Oldest Copper Mine in America. Rich Ore Mined Under Difficulties

It seems that the first discovery of copper was made by Spanish adventurers in 1544, 9 years after the founding of the town now called Cobre. The mine lies 12 miles northwest of Santiago Bay in the valley of Rio del Cobre, in the Sierra Maestre Mountains, 400 feet above sea level.

In the early days, this town lay in the center of rich coffee and sugar plantations, and the mountains in which El Cobre Mine is situated were the summer resort, as it were, of merchants residing in Santiago.

In these days El Cobre was by far the most important town in Oriente Province. From the time of its discovery the mine attracted wide attention in Spain, and the Crown was richer by vast sums from its operations, the Queen of Spain herself being personally interested in the railway which was subsequently built to bring the ore to the coast. The property was worked for a century by the Spaniards, who employed for the purpose slaves, both African and Chinese

The most important feature of El Cobre and one that has made it widely known throughout the world, is the Sanctuary of the Virgin of Cobre. This stands upon the very apex of the Cobre vein, and its pink dome and square blue towers may be seen from any point in the mountains for miles around. The incident of its erection forms a part of the history of the mine itself.

Alonzo Ojeda, from San Domingo, wrecked upon the shores of Cuba early in the fifteenth century, was saved from death by an Indian chief, Cacique by name. Ojeda had with him an image of the virgin, and, in fulfilment of a vow conditional on the saving of his life, built a chapel to her in the town of Cuieba. This was the first christian chapel in the island of Cuba. The Indian chief, however, stole the image to try the Lady's charm against his enemies, and it is said that so great was her power that he never lost a battle while he had the image in his possession.

In 1600, so legend has it, Juan de Joyos and Rodrigo de Joyos, Indians, and Juan Moreno, a negro, found the image floating in the bay. On a board upon the image was written: "Yo soy la Virgin de la Caridad"—"I am Our Lady of Charity." These three men took the image to Hato, where the mayor erected an altar for her, at the same time notifying the manager of the Royal Mines at El Cobre, Don Francisco Sanchez de Moya, that he had done so. Francisco built a chapel for the Virgin, first at Hato, and afterwards she was brought to Cobre, and a chapel was built for her in the town.

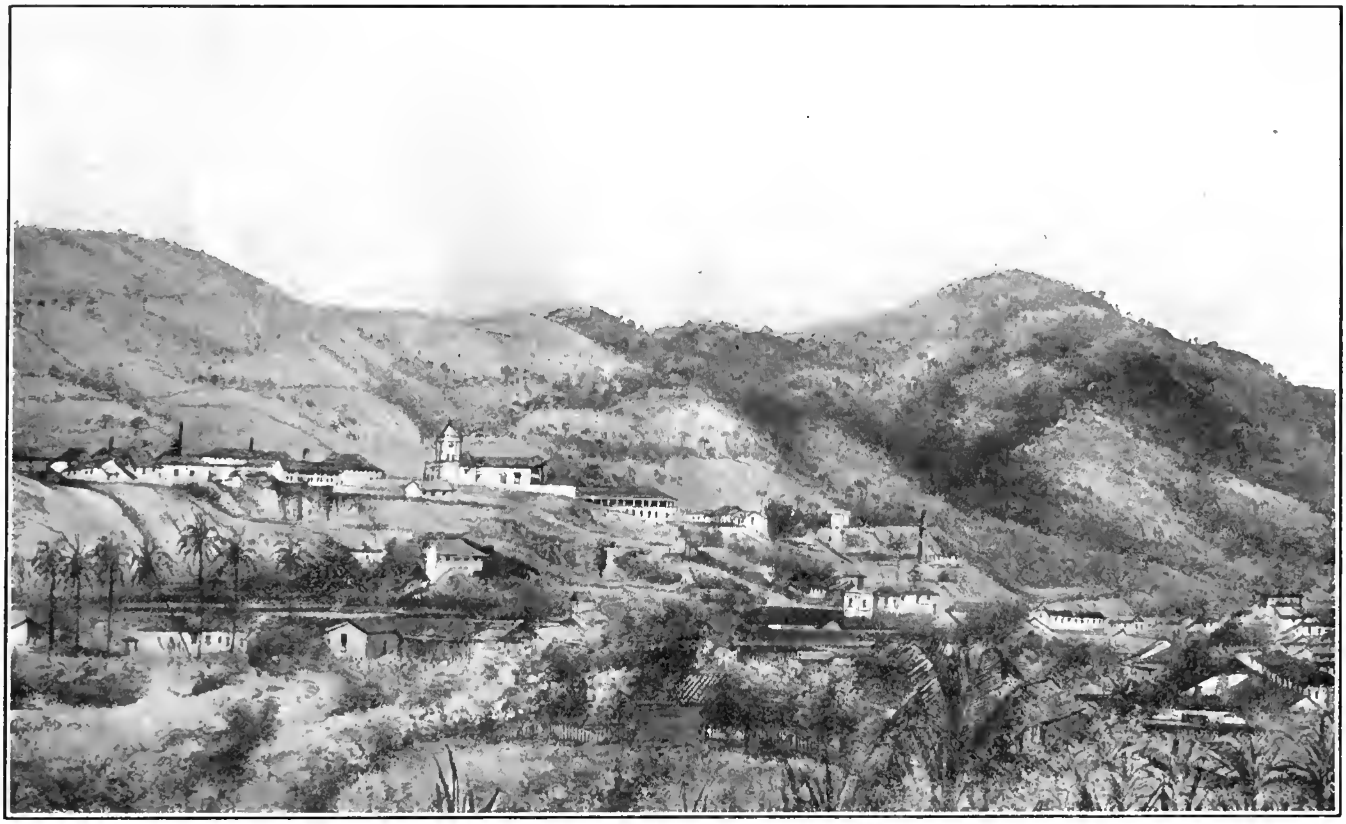


Fig. 1. Part of Town of El Cobre

As the mine prospered, Our Lady of Caridad became, as it were, the patron saint of the mine, and the sanctuary was built for her protection on the present site. It is a handsome building, 150 ft. × 90 ft., supported by massive columns, and is reached by 430 steps from the village below. Up these steps on their knees the pilgrims passed in thousands to pay their "promesas" to the Mother of Charity, and many wonderful cures of the sick are mentioned. Back of the chancel is a pile of crutches and canes, testifying to the fact that cripples had left the sanctuary without these aids to locomotion.

Heroic figures of the saints stand on either side of the entrance to guard the Virgin, who, in her cloth of gold, stands on the altar under the great dome. The figure is about the size of a

*Journal of the Canadian Mining Institute. Part of Vol. XIII, "Two Cuban Mines."

child's doll, and holds a small image of the Holy Infant in her left hand, who in turn holds a sphere symbolizing the world. The Virgin's right hand is raised in benediction.

Today Our Lady of Caridad reigns disconsolately in the old station of the Cobre railway, at the foot of the old steps. The sanctuary, however, still stands as a monument to days gone by.*

Fig. 2. Modern Boiler Plant, El Cobre Mine

In 1700, Francisco Sanchez de Moya was succeeded at the mines by Juan de Equilus, who obtained a concession from the Spanish Crown, agreeing to pay 200 tons of copper yearly to the Arsenal in Havana. Failing to pay his tribute, he was put in jail, and his successor fared no better. The mines were then worked by the Crown until 1830.

As early as 1623 the English displayed much interest in the yield of copper Spain enjoyed from the island of Cuba, and had already cast somewhat longing eyes in this direction. It is recorded that Oliver Cromwell sent a commission to Cuba to investigate this deposit; but there is no record concerning the report of such a commission.

The first appearance of the English on the scene was in 1830 when the Consolidated Company, an English syndicate, purchased some of the claims from the Crown of Spain, and mined actively until the necessary suspension of operations occasioned by the 10-years' war in 1869. Meanwhile this period of 39 years of operation seems to have been the most active in the history of the mine. Tall chimneys of rounded stone rise from among the rusted shells of old Cornish boilers, and the great low-pressure cylinders of the Cornish pumps are still on the ground, testifying to the energy which characterizes the English race. Even block houses were built to defend the operators from attack. There is still on the ground a cast-iron pump bob weighing 60 tons, made in Glasgow, which it is alleged occupied the energies of 500 men for 6 months to transport it from the coast to the mine.

Forty old shafts are recorded, and remnants of old hoisting machinery litter the ground. One flat woven wire rope, 6 inches wide and 1,000 feet long still remains upon the reel.

* The sanctuary was injured by the mine cave in 1906.

Many of the buildings were of massive stone, and provided with port holes indicating that they were used not only for dwellings, but for defense purposes in cases of attack.

On the topmost point of one of the highest hills, known as Hardy Hill, are the remains of the former manager's house, to which a zig zag trail leads. At this height a magnificent panorama presents itself of Santiago Bay, the entrance guarded by Morro Castle, and beyond the sweep of ocean. From this height the stirring events of the Fourth of July, 1898, could have been watched, when Cervera's fleet was defeated and captured just outside Santiago Bay.

The underground labor during English operations was performed by black slaves and Chinese, supervised by Cornish mine captains. Much in the records with reference to the operations at this time is tragic in the extreme, for the mine managers had under their command a corps of armed men, and the treatment of the slaves is said to have been cruel. Slaves were paid $15 per month, and an old letter from one mine manager mentions that more Chinese labor must be introduced since slaves were scarce and free labor commanded the prohibitive wage of $24 per month. Such a wage nowadays would certainly not appear excessive.

Rock and ore were hoisted in buckets and stull timbering was exclusively used. Much ore was in consequence left in the mine, since no method was devised for mining the ore under conditions when the ore body bulged.

When the mine was pumped out, many handsome mahogany stulls of large size were found, and these beautiful stulls, 24 in. × 24 in., 30 to 40 feet in length, were again used in retimbering dangerous places. A wooden pump rod 18 in. × 20 in., and 63 feet long, was removed from the shaft. These timbers were perfectly preserved by the copper sulphate waters, and, as already remarked, were used again after an interval of 50 years.

In 1869, as the mines became deeper, greater difficulty was experienced in keeping the workings dry, especially in the rainy season. The breaking of the Cornish pumps caused serious delays, out of which lawsuits arose between the Spanish and English companies operating the mines. A very complete history of this litigation is given by one Quintana, a mine inspector employed by the Spanish government.

Fig. 3. Old Stack at El Cobre, Sanctuary in Distance

As shown in Fig. 6, much of the surface of the property is covered by fine tailing dumps, the result of hand jigging. There are also the remains of an old stamp mill, Fig. 4, all of which is proof that much of the ore was concentrated before shipping. And indeed the concentration of this ore was necessary, as the freight rate to the coast and to England was excessively high.

Previous to the year 1844, when the railroad was built, the ore was "packed" to the coast on mules, and camels were also for a time employed, but in such a rocky country proved to be of little use.

In 1844 the Cobre Railway, shown on the right in Fig. 7, 8 miles in length, and replacing the use of 2,000 mules, was built. The Queen of Spain was the principal stockholder in this railway, and a freight rate of $6 per ton for down and $4.50 for up freight, was established. The railway was built, as our manager, Mr. Emerson, says: "with a fear of curves and a disregard of grades, which necessitated five important bridges and one incline, up which the trains were hoisted by cable!" The railway became, from its commanding position, the controlling feature of the property, and ultimately such high freight rates were charged as to leave no margin of profit to encourage the reopening of the mine at the conclusion of the 10-years' war. For example, no ore carrying less than 13 per cent. copper could be shipped with profit.

In these days the ore was sent to Swansea, Wales, and was treated in reverberating furnaces.

The old companies had the advantage of a high price for copper and cheap labor, but were handicapped by inadequate pumping machinery, inferior methods of mining, and high freight and treatment charges. In 1869-70 all operations were suspended, chiefly by reason of the war which centered around Santiago. Some 30 years later, in 1901, after the conclusion of the war with Spain, a New York syndicate was organized for the purpose of unwatering the mines.

In order to install modern machinery in the mines it was first necessary to rebuild the railroad. As it had been entirely destroyed, this was no light undertaking. The mine was equipped with heavy pumps, and the first difficulty encountered was occasioned by the copper in the water, which had a disastrous effect on the pipes and pumps. Again, the shafts proved to be full of miscellaneous debris—old pump rods, timber, boxes of cartridges, cannon balls, and human bones

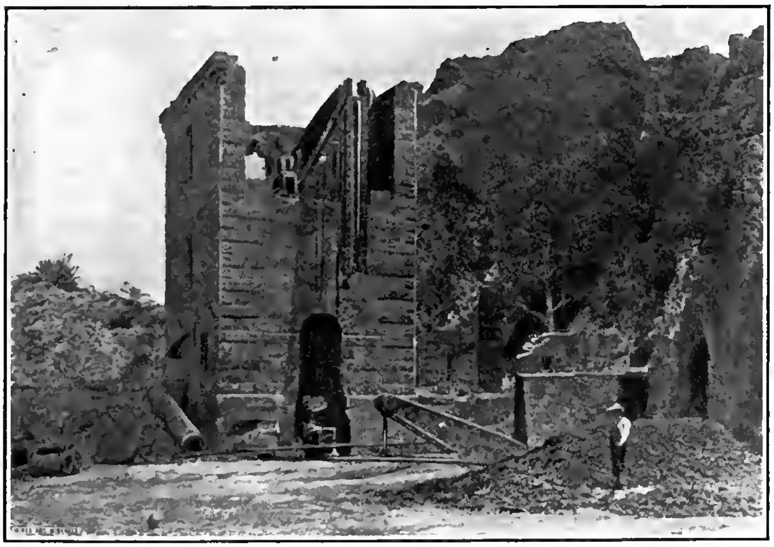


Fig. 5. Old Cornish Pump—El Cobre

were removed. These difficulties delayed the unwatering for more than 2 years; and when, finally, a depth of some 200 feet was reached, the company had the misfortune to experience an exceptionally rainy season accompanied by unprecedented floods, which latter carried away a large part of the town of Cobre and five railway bridges, while some 35 people were drowned.

The following year, 1903, the 500-foot level was reached, and some mining was commenced in the unwatered area. The railway was repaired, and a 250-ton smelter and 400-ton concentrator built at Punta Sal, the port. The company smelted and concentrated some 85,000 tons of ore, averaging 7 per cent. copper in this year. The water taken from the mine for the first

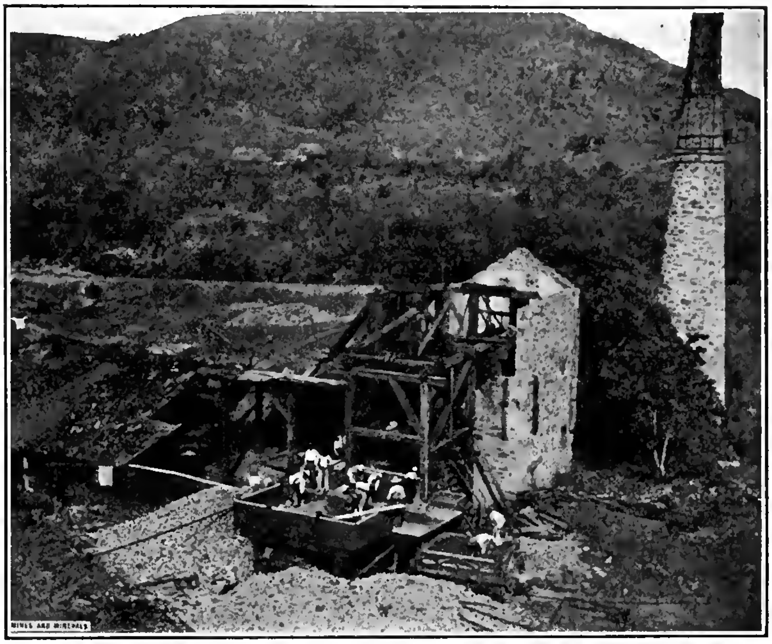


Fig. 4. A Modern Head-Frame in an Old Shaft House

6 or 8 months contained some 400 grams of copper per cubic meter. This water was run over Californias, as is the practice today. After the first year, however, the copper contents in the water rapidly decreased, and now only about 50 grams of copper per cubic meter is recovered.

The large quantity of low-grade oxide ore in and about the surface workings led to the erection of a leaching plant using sulphuric acid; but owing to the loss of acid and heavy cost of transportation, this undertaking was unprofitable.

As greater depth was obtained the old workings were found to be badly in need of timbering. They were, moreover, filled with a soft mud, and it proved a most dangerous and expensive task to lower to the 800 feet. Bad air from rotting timber and sulphides caused much annoyance and some deaths, and caves of old stopes were a constant menace.

In spite of all these difficulties, work was continued at enormous cost, and a considerable production was being maintained, when in May, 1906, a cave-in occurred which for a time practically rendered the lower portions of the mine inaccessible. At the time of this disaster 60 miners were confined in stopes filled with bad air; but through the heroic work of the superintendent, M. B. Yung, and his assistant, who is now superintendent of the mine, Edward B. Nagle, only two men lost their lives.

The pumps were pulled and the old mine temporarily abandoned. This was in 1906.

The management of the company now gave attention to working the upper portion of the property and to the outlying or adjoining property, which they thoroughly prospected. In the latter part of 1906 a huge blow-out, or surface deposit of copper ore was found, from which 64,000 tons of 6 per cent. ore was mined and shipped to New York. In addition, systematic exploration resulted in the mining of 100,000 tons of 8.7 per cent.

ore from the area above the 200-foot level. This ore has all been shipped to the United States for smelting, where remarkably good rates were secured and substantial profits made.

The mine is at present producing from 3,000 to 4,000 tons of ore per month, and exploration is once more being actively prosecuted with the object of again attempting to unwater this wonderful old mine in order to reach the 1,200-foot level, supposed to be the lowest level in the property.

The character of the ore deposit has been very fully described in an excellent paper written by the manager of the property, Mr. E. H. Emerson, and this description I present as a part of this paper.

Character of the Ore Deposit.—The strike of the ore bodies is with the valley, northeast and southwest. A series of rocks, from a coarse agglomerate to a fine-grained rhyolite without definite boundaries, constitutes the country rock of the deposit. The entire region is volcanic, traces of sediments occurring only on the summits of the surrounding mountains. One main fault, with several lesser parallel faults, striking with the formation, is responsible for the deposit. The main fault dips slightly to the south and carries a mud seam 3 inches to 5 feet in width. It presents a straight, smooth surface in many places in the mine. Another nearly parallel fault 300 feet to the north, called the

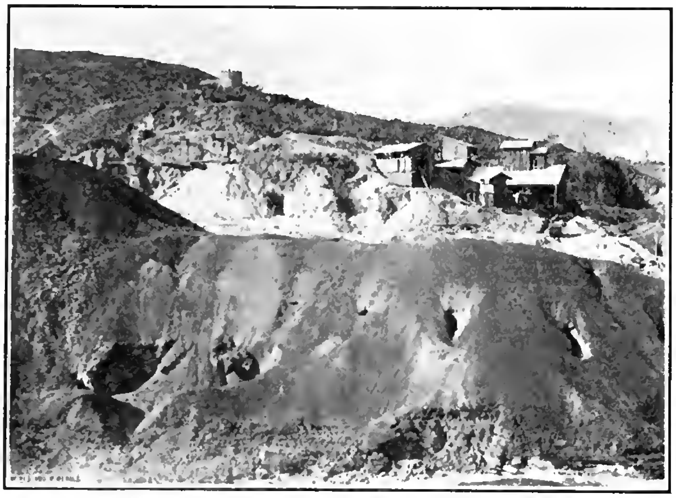


Fig. 6. Old Dumps, El Cobre

North vein, dips to meet the main fault at 1,000 feet in depth. To the south is a series of lesser faults that diverge from the main formation at the east end of the property, and extend along the side of the mountain range for a mile.

The main fault formed what has been called the Middle vein. This is a series of lenses of ore in brecciated zones on both sides of the fault, sometimes touching it and sometimes separated from it by many feet of rock. These lenses are irregular and are cut off by slips and joints without presenting regular walls, while the other faults have produced veins with clean walls, though varying greatly in width. A transverse fault zone has caused a 50-foot throw of the Middle and North veins. Along this zone, and near it, occur the greatest bodies in both veins. The widest lense is 125 feet, and they are almost continuous for a half mile with varying widths.

The veins, with clean walls and rarely exceeding 20 feet in width, were well worked by the stull timber; but caving and slice filling have recovered much ore lost by the old methods in the lenses.

The gossan extends, pretty generally, to a depth of 100 feet, carrying pockets of very rich oxides; but it is noticeable that there is a barren zone of gossan for 10 to 30 feet above the rich sulphides.

Near the surface, and close to the transverse fault between the veins, a brecciated zone was found in the agglomerate. This was filled with oxides, cracks being filled with a network of criss-cross stringers averaging an inch in width, and the mineral also impregnated the leached, almost kaolinized, agglomerate. The rotted rock was easily separated from the harder stringers of oxides, and 10 to 15 tons of barren rock was handled and hand picked, producing 1 ton of 10 to 12 per cent. ore at a good margin of profit. This was worked from open cut and by slice-filling from below.

Pyrite, in perfect separate crystals, is common in great areas of the agglomerate, and much of the country rock shows stringers of gypsum.

The North vein, to the east of the transverse fault and at the 500-foot level, showed a very abrupt change, from the enriched tarnished chalcopyrite with silicious gangue, to a clean bright chalcopyrite with a pure gypsum gangue. Extending with lessening values to the 700-foot level, the vein became pure white gypsum with only an occasional piece of ore. At the same time the lenses along the main fault continued with enriched sulphides and quartz without a trace of gypsum, and the North vein, to the west of the transverse fault, showed no gypsum.

The depth of the enriched zone of the sulphides has never been determined. It occupies at least the area from the 100-foot level to below the 650-foot level. In this area large lenses of fresh 3½ per cent. ore contain chutes of very rich sulphides. A stope on the 550-foot level was supplying 22 per cent. ore at the time of the cave-in. The depth of the lowest workings of the old companies is variously estimated at from 1,000 feet to 1,200 feet, and, as only rich ore could be handled at that time it may be assumed that the enriched zone extends to that depth.

All ores of copper are found, from the red and black oxides and native copper in the gossan, through native copper in considerable quantities in the top of the sulphides, and all the sulphides down to clean chalcopyrite.

The ore is highly silicious (from 45 per cent. to 65 per cent. excess insoluble) carrying only 20 cents gold and ½ ounce silver. It is very free from impurities.

The veins become tight to the east, and although joined by the southern veins of the Santiago Hill, have shown no values of importance to the east of the Cobre River. At the west end they are cut off by a massive igneous intrusion. Two agglomerate hills extend, with the same strike, to the west as though the vein formation has been faulted to the north. Pyrite is very prevalent; but no copper of importance is met with.

The Santiago veins are narrow and appeared to pinch in depth. The only important ore on the Santiago Hill, worked by the Cuba Copper Co., was a peculiar pocket of ore formed by the intersection of the Santiago vein with a band of leached pyrite in the porphyry. The deposition of copper oxides and carbonates followed the spherical lines of weathering of the porphyry, forming an unusual and striking ore of spotted and striped black, red, and green in a white weathered porphyry matrix, which was well called "Tiger ore" at the smelters.

It is peculiar that at the west end of the main Cobre veins, which show only a trace of gold and no impurities, a small vein of zinc blende, chalcocite, and pyrite, carrying $15 in gold, should occur. This vein has the general direction of the others and is only 175 feet deep, 200 feet long, and about 10 feet wide in its widest place. It is a little to the north of the main veins and cut off, as they are, to the west.

The total production of the property can only be approximated. It is, however, known that a million tons of ore and 140,000 tons of copper have been shipped from the property since 1830.

El Cobre has always been handicapped by the extremely silicious character of its ore. A systematic attempt is now being made to develop a basic ore on the island, for utilization in conjunction with the Cobre ores. Should this attempt be successful, El Cobre may again take her place among the important

producing mines and add another page to her varied history, which already covers a period of 375 years.

ROCKS OF EL COBRE

Agglomerate.—Sharp angular pieces from pebble to fist size, cemented by a bluish material very silicious. Blocks have the appearance of concrete. Easily rotted and changed to kaolin near surface.

Breccia.—Rhyolite and porphyry breccia, occurring throughout the agglomerate without defined boundaries. Fine grained, light gray and reddish.

Rhyolite.—Gray, fine grained, and in depth glassy.

Tuff.—Rhyolite, tuff occurring with the rhyolite. Easily rotted to white near surface.

Massive Intrusive.—A coarse-grained porphyry. Iron-stained in weathering from contained pyrite.

DISCUSSION

Dr. Douglas.—Mr. Lawrence has touched on the recent history of copper mining. I think he might have gone back to even pre-Columbian time in his story of this interesting old copper mine. There is every reason to suppose that these copper ores were utilized before the discovery of America. Mr. Clarence Moore, who had been doing a great deal of archeological work in Florida and elsewhere, wrote to me some years ago to ask if I knew anything of a native alloy of lead and copper in New Mexico. I told him I did not think there was any such thing in the world—I was very sure that there was no such thing in New Mexico. He said that he had been informed that such a native alloy existed only in New Mexico; adding that he had discovered some copper implements in the mounds of Florida, that had been made from such an alloy of lead and copper; hence the deduction that there was a commerce in those pre-Columbian days between New Mexico and Florida. I told him I thought it was a mistake, and that his chemist had somehow or other blundered, and I advised him to send his samples to Doctor Ledoux, who would probably form some other theory to account for the copper found in those mounds. He sent the samples to Doctor Ledoux, and Doctor Ledoux of course found that there was absolutely no lead in the copper, but that it was remarkably pure copper.

The question, therefore, was where did the copper come from? I told Mr. Moore that I thought the copper came from whatever region produced copper of exactly that description. It was extremely free from arsenic, and free from antimony. If it had come from the lakes it would contain a certain amount of gold and silver. Of course we all know that copper was mined in Michigan by the natives before the discovery of America, and that it was transported almost over the continent. I suggested to Mr. Moore that possibly the copper may have come from those old Cuban mines, because Columbus on, I think, his third journey, mentioned having seen some natives, from what he supposed to be Asia, who were navigating a very large canoe and carrying on some commerce between the island of Cuba and the Asiatic shore. I advised Mr. Moore to get some samples of native copper if he could from the outcrop of this Cuban mine at Cobre. He did so, and Doctor Ledoux found that the analysis corresponded absolutely with the copper that he had exhumed from these old mounds.

The conclusion, therefore, I belive, is that these mines were worked in pre-Columbian time, and that there was a commerce in copper between the island of Cuba and the continent of America and that now, after so many vicissitudes—political and economical—we are only reviving an old industry that was active more than three centuries ago.

Dr. J. D. Irving.—Mr. Lawrence's reference to these mines has been very interesting to me—the first reference particularly, because it deals with a question to which I have lately devoted considerable thought and concerning which Mr. Ransome read a paper—that is, in regard to secondary enrichment of ore deposits in their relation to climatic conditions.

In the course of some experimental work that I had an opportunity of doing in Alaska, I found that a great many outcrops of cupriferous pyrite were absolutely unoxidized at the surface. The waters of that northern climate were so cold that there had apparently been no opportunity for deep oxidation to occur. Sulphides extended actually to the surface, and there had been apparently no opportunity for leaching to go to any depth. Even in the stream beds themselves there were present particles of sulphides of iron and to some extent cupriferous pyrites, that were not oxidized by the running water of the streams.

Coming farther south to a region of temperate climate, we find a great contrast in the deep secondary enrichment at Butte, Mont. Finally, in tropical countries such as Cuba, we find the conditions that Mr. Lawrence has encountered in the mine which he has just described. I am especially anxious to know the details of oxidation here because it has been very difficult for many of us geologists to get much information as to the depth of oxidation under tropical conditions. Generally the chemical changes may be assumed to extend to much greater depths than farther north, provided the level of standing water is itself sufficiently deep. I should like to ask Mr. Lawrence if he can give me any details on the level of ground water in the mine at the

Fig. 7. The Old and the New at El Cobre Mine

time it was pumped out and the relation in depth, of oxide, secondary sulphide, and primary sulphide zones to this line.

Mr. Lawrence.—I suppose the effect of climatic conditions upon copper ore in Cuba can best be illustrated by one of the numerous catastrophes that seem to have followed the operations there.

A stock pile of copper amounting to some 35,000 tons was placed at the harbor. The stock pile was uncovered and subjected to action of rains for a year. It was partly oxidized ore and partly sulphide ore. Some of the ore that came from the lower levels was clean chalcopyrite, and some from the upper levels was oxidized. When the ore was finally shipped to New York it showed that it had lost ½ of 1 per cent. in copper. We found on investigation that oxidation had been going on in this stock pile rapidly all the time. On digging a hole in the stock pile you could hardly hold your hand in it on account of the heat generated by oxidation. Every rain that fell on that stock pile carried away a substantial amount of copper.

The same operation, I think, happened in the upper portions of this copper mine, and the oxidized area extends approximately 200 feet to the enrichment zone. The gossan extends generally to a depth of 100 feet, carrying pockets of very rich oxide, and it is noticeable that there is a barren zone of gossan 10 to 30 feet from the surface above the rich zone.

The geology of this mine has still to be studied from its lower levels and as yet this history has not been written.

Doctor Irving.—There is just one point on which I should like information: What was the average level of the surface of the water as it stood in the mine when you commenced operations?

Mr. Lawrence.—Well, I cannot positively say as to that. They have been holding the water at the 200-feet level for many years, and that would have been fairly well up in the oxidized zone.

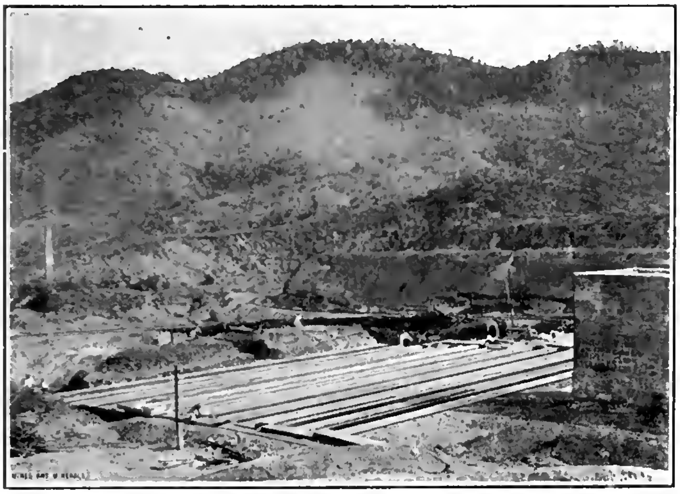


Fig. 8. "Californias" for Recovering Copper from Mine Waters

CUBAN GOLD MINES

Written for Mines and Minerals, by E. B. Wilson

Santiago Province, Cuba, is undoubtedly the mineral spot on the island, for it contains workable deposits of copper, iron, manganese, and gold, not all of which by any means have been developed. About 20 miles south of Gibara, in the vicinity of Holguin, gold is said to have been mined off and on since the island was discovered, mostly by washing, although there are four places where shafts indicate more progressive mining. Tradition states that royal grafters took tolls from the miners which were excessive, and that between buccaneers, wars, and official grafters, there was no incentive to mine. It appears, however, that a French apothecary living at Holguin found a vein-like deposit of magnesium carbonate at Agua Claras, which was about 18 inches wide, between serpentine walls. To manufacture some kind of product he calcined the material, slacked and floated it. In the residue he found considerable gold, of which he made no mention to the authorities. It was not long, however, before he was suspected of being an alchemist, and to avoid complications he withdrew from the island. The writer, under philanthropic emotionalism for "Cuba libre" and as a collector of curios, took one-quarter interest in this magnesium carbonate deposit.

About 4 miles south of Agua Claras, J. S. Black, who had prospected from the Yukon to Panama and Cuba, by systematic research discovered a rich deposit at what is now termed the Santiago Mine. According to the mineral laws of Cuba, when one finds minerals he may denounce a claim even though it be on another's land, and as no work is demanded, the ownership trails along from generation to generation and then some, when an outsider from the States starts to work. The Santiago land was held by well-to-do Cuban people who readily agreed that Black could mine on a 20-per-cent. gross royalty basis. Black thought his place was so rich he could afford this, and taking a quart fruit jar of coarse gold under his arm he sought capitalists that would help him develop. He found a young New Yorker who agreed to furnish $75,000 for a mill and for development provided Black would give him about four-fifths of the company stock. The young man had a friend in the mining machinery business who knew just what was needed and the mill was equipped as follows: Crusher, Chilean mill, shaking plates, and tables, all of a kind which a practical metallurgist would not have ordered, and, with the exception of possibly the tables, unsuited to the ore. The mill, which would not have cost $20,000 in New Jersey, cost $60,000 in Cuba; and with the waterworks, pumps, hoists, and developed work, absorbed all the money and more, so that when the mill did start there was a big debt to overcome. No arrangements were made to empound the tailing, consequently the people on the next lower properties were able to make good wages by washing the waste. In fact far better returns than the stockholders.

The rock carrying the gold has been leached beyond any semblance to the original, which judging from adjacent properties, approximates a quartz-felsite porphyry. This gray rock has been intruded between walls of serpentine wherever in the vicinity the rock had been shattered. Although subsequent movements have faulted the felsite dyke locally, the line along which the gold is found for several miles is practically northeast and southwest.

In the gold belt in the vicinity of the gold-bearing outcrops and in ditches after each rainstorm, gold colors can be obtained. The soil is sharp, and wide areas of talus cover the fields, so that beyond grass and shrubs vegetation does not flourish. If the ground in this vicinity was colored reddish brown the formation could be termed "laterite." Mining men have undoubtedly noticed that near gold deposits there is a gritty feeling on the shoe soles, and in the Santiago gold field this geological peculiarity is properly developed. While $250,000 has been taken from the Santiago Mine in a short time, the combination of debt, royalty, dry weather, high-priced fuel, mine timber, supplies, and labor, absorbed the proposed dividends. There are those who believe that with suitable machinery and systematic mining, the gold deposits in this vicinity could be worked at an exceedingly good profit. The locality is healthy, Holguin being about 500 feet above sea level and one of the oldest towns on the island. Asbestos of the variety known as chrysotile was found near the mines. The life of the material had been sapped from it by the climate and solutions so that it was reconverted into brittle rock at the outcrop, although it retained the fibrous appearance.

COMBINATION ASSAY OF COPPER BULLION

Written for Mines and Minerals, by S. M. Scott, Jr.

Weigh 1 assay ton of ground or sawed sample of copper bullion into a 24-ounce beaker, add 25 cubic centimeters of mercuric nitrate solution, made by dissolving 12 grams of metallic mercury in nitric acid and diluting to 2 liters with water. This is to prevent the solution from climbing on the sides of the beaker. Then add 75 cubic centimeters of sulphuric acid, 1.84 specific gravity; and place the beaker covered on a hot plate or sand bath until it goes down to fumes. Take off, allow to cool. Dilute to about 500 cubic centimeters and stir up the heavy precipitate of copper sulphate in the bottom of the beaker to prevent bumping; add enough salt to precipitate the silver, then boil to get the copper sulphate into solution. Now take off the bath and dilute until the beaker is about full. Allow to stand until cold and filter through double thickness (2) 597 filter paper, using a corrugated funnel to hasten filtering. Wash precipitate four or five times and place papers and precipitate into 10-gram crucible. Burn paper at low heat and flux as follows: 15 grams soda; 7½ grams potash; 1½ grams borax glass; 1½ grams silica; 1½ grams flour; 30 grams litharge. Place the crucible in a muffler at low heat to start and finish at high heat. Fusion should take from 20 to 25 minutes. Cupel, weigh silver, and part for gold in usual way.

THE PANAMA CANAL

Written for Mines and Minerals, by G. H. Mee

The efforts made to discover a waterway across the continent date back as far as 1515, at which time Balboa and other adventurous Spaniards succeeded in crossing the dense jungles and swamps and reaching the Pacific Ocean. The first roadway across the isthmus was constructed in 1521 during the reign of Charles the Fifth, the Atlantic terminal being Nombre de Dios, and on the Pacific, Old Panama. This roadway was constructed with almost insurmountable difficulties, and was completed entirely by manual labor and whilst there was no other permanent line of communication; it was finally abandoned owing to the turbulent condition of the country.

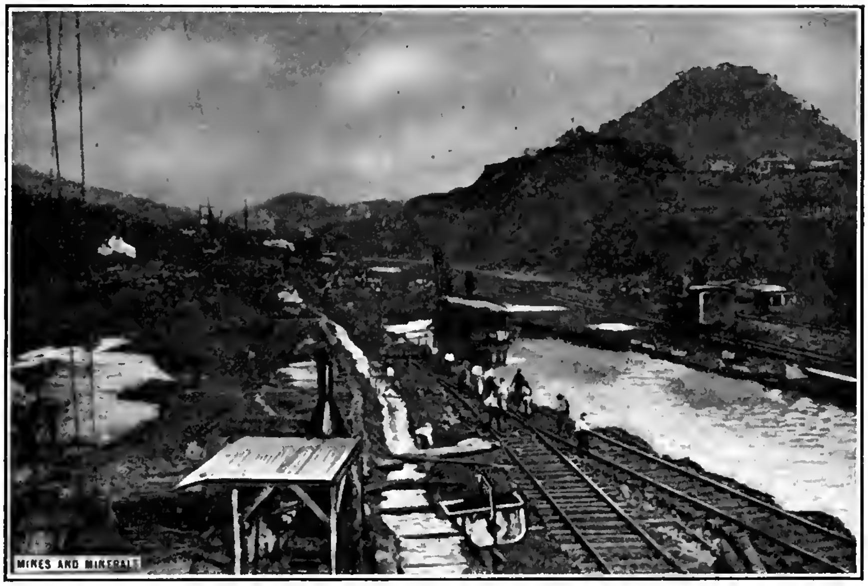
Early Plans and Attempts at Construction. Modern Methods and Conditions

Portions of this wonderful road still exist and can be followed for many miles, although it is overgrown with dense jungle growth.

The Republic of Central America, in 1825, requested the cooperation of the United States in the construction of a waterway through Nicaragua.

This route was favorably considered and the scheme thoroughly investigated, but the project seems to have been tied up on account of inability to secure sufficient funds at that time.

In 1826 a British corporation sent a Mr. J. Bailey to Nicaragua to secure a concession with the government there, which concession he failed to arrange, and it was shortly following this attempt that the Colon-Panama route was again under consideration. Very little of the geology of the Isthmus was known in 1827; in fact, the heights of the tides and the altitudes of the mountains had never been ascertained, and it was at this time that a Mr. J. A. Lloyd was commissioned by President Bolivar to survey the Isthmus of Panama, with a view to discovering the possibilities of rail or water communication. In 1831 the Republic of Colombia was again disrupted and the canal regions became part of New Grenada, and no action was taken on the plans made by Mr. Lloyd.



PEDRO MIGUEL CUT AFTER HEAVY RAIN

The Republic of New Grenada then granted a concession to a French company in the year 1838, which company made an extensive examination of the region and presented the project as a comparatively easy undertaking.

Five years afterwards the statements made by the company were rigidly investigated by the French Government and the claims made were then heavily discounted and led to its failure.

The United States Minister in Nicaragua negotiated a treaty with that Republic in 1849 by which Nicaragua agreed to confer upon our government the exclusive right to construct and operate roads and canals between the two oceans.

A concession was then granted to the American, Atlantic and Pacific Ship Canal Co. (of which Cornelius Vanderbilt was a member) and in 1852 Colonel Orville Childs completed a thorough instrumental survey of the entire region, on behalf of this company, and the plan was pronounced entirely practicable. Nothing became of this work and from 1852 until 1872 practically nothing was done toward opening up this line of communication.

In 1872 the Interoceanic Canal Commission was established, by whom explorations were conducted in various parts of the Isthmus, and following all the ineffectual attempts, the concession was transferred to La Compagnie Universelle de Canal Interoceanique de Panama, or more generally known as the Panama Canal Company, promoted in May, 1879, by Ferdinand De Lesseps.

The colossal failure of this concern is too well known, and it could hardly have been otherwise in view of the mismanagement and insane extravagance of its officers. The residence of one of the engineers is said to have cost some $100,000 and the salary of this gentleman was $50,000 per annum. A Pullman car that cost $42,000 was assigned to his use when on tours of inspection, whilst another man built a private bath house that cost the company the small sum of $40,000. Graft, or rather wholesale robbery, existed everywhere in forms too numerous to mention, and it is said that some of the foremen arranged to have their gangs meet the pay cars in two different places each month and receive double pay, and no check or tally seems to have been kept of either stores or equipment, and that half the names on the pay roll were of persons who never existed. An effort was made to reorganize the company without success.

In 1899, the Congress of the United States authorized the President to make an exhaustive investigation as to the most feasible isthmian route, and the work was placed in the hands of a body known as the Isthmian Canal Commission. The report of the Commission resulted in the taking over of the canal from the French. The remnants of their equipment were also taken over, and although the bulk of the machinery was entirely unsuitable for the work, some parts of it have been used to advantage and the remainder can be seen today rusted and decaying, almost hidden with jungle growth. In 1906 the United States was engaged in active operations on the Isthmus. The question of a sea-level or lock canal was not settled for some time afterwards, and a lock canal was finally decided upon.

The necessity of owning the Panama Railroad was recognized by the French company as an essential to conducting the work, and the road was purchased by the United States Government at the onset. The Panama Railroad Co., composed of William Henry Aspinwall, John Lloyd Stephens, Henry Chauncey, and others, obtained the exclusive right of building a railroad in 1849. The engineers were on the ground in the fall of 1849 and worked waist deep in mud and water, making surveys and cutting trails. The construction was practically completed in 1857 at a cost of $8,000,000; it is now 45 miles long and has a double track. Nothing but the indomitable will and grit of Americans could have ever overcome the terrible difficulties

encountered at every step. The whole Isthmus was a wilderness, pestilential and death dealing, the jungles and swamps were alive with poisonous snakes and insects. It was reported that in some of the swamps the engineers under the late Colonel George M. Totten failed to find bottom at 180 feet, which report was verified by the trouble had later in the Black Swamp, where it is said that in some parts no bottom can be found. It has cost thousands of dollars to maintain the right of way across this swamp, and sometime in 1907 the trestle sank some 20 feet without warning and was entirely submerged. A number of trains had passed over during the day and it was a marvel that no accident occurred when a passenger or freight train was crossing. Hundreds of old French cars and machinery were dumped into this hole and piled over with rock until a solid foundation was laid and piles were then driven in. The greater the quantity of rock and junk dumped into the swamp, the higher the mud oozed up out of the water on each side.

A great lesson on the labor question was learned during the building of the Panama Railroad, when it was found that the natives could not be counted on to any great extent. A large number of Chinamen were introduced with very poor results; they fell sick almost immediately after their arrival, and within a few weeks hundreds were unfit for work. They finally began to destroy themselves by whatever means came nearest at hand, and today the town known as Matachin derived its name from the fact that hundreds of Chinamen hung and otherwise killed themselves to put an end to the misery and suffering caused by unfitness for the climate, and lack of opium and suitable food and accommodations.

Bas Obispo Rock-Crushing Plant and Steam Shovel During Flood

Irish laborers were imported at a big expense, and although the mortality among them was less than that experienced with the Chinamen, it is reported that the company failed to get a good day's work out of any one of them. A great number died on the Isthmus and the remainder were shipped to the States, where many more died from fevers contracted.

The late Colonel George M. Totten, engineer of the road, always discredited the statement made as to the excessive mortality, and said that at no time the number of employes exceeded 7,000, and that the total mortality in the 5 years of construction was not over 1,200. It has frequently been stated that the Panama Railroad cost a life for every tie laid, which is untrue, although thousands of lives have been lost since the first attempt was made to run the canal through.

The Panama Railroad was completed with the labor of some 3,000 men of mixed races, chiefly negroes from Jamaica and West Indies.

The working force on the canal at present consists of Spaniards from the agricultural districts in Spain, Panamaians, and Colombians, the greater part of the laborers being recruited from Jamaica and Barbados; and although the negroes do not seem to be so susceptible to fever as the Spaniards, it is thought that the Spaniards are by far the superior class of laborers.

The total cost of the canal, including the $50,000,000 paid to the French company and Republic of Panama, is now estimated at $375,200,000. This figure is considerably larger than the original estimate, but the results obtained have justified the expenditure.

One of the most important stations on the Zone is Gatun, where the great lake will cover a space of 165 square miles in area. The dam extends between two hills and measures 9,040 feet along the crest, including the spillway or overflow, and is 1,900 feet wide at its base.

The crest of this dam is 115 feet above sea level and is 100 feet broad and will be some 30 feet above the normal level of the lake.

The entire structure has been made 16 times stronger than was originally recommended by the engineers.

The spillway has a channel 300 feet wide over which some 140,000 cubic feet of water from Chagres River will flow every second.

The present sea-level canal from Cristobal to Gatun is about 6 miles in length and was commenced by the French company and will enter the locks at Gatun.

These locks consist of three pairs, each 1,000 feet long by 110 feet wide, and all west-bound vessels will be lifted 85 feet to the surface of the lake. The locks are reinforced by scrap steel rails set in a horizontal position in the concrete.

The cement sheds are located on the banks of the old French canal, and the crushed rock and sand from Nombre de Dios and Porto Bello are stored nearby where they are easily handled at the mixer. The mixed concrete is loaded on little cars operated in trains by electricity from a third rail and is distributed by the cableway conveyers to the locks. These concrete trains move in a constant chain for 10 hours each day distributing the mixture to the conveyers.

Another important station is Culebra, where the big cut extends for a distance of 9 miles and follows the route of the old French survey.

The cut passes between Gold Hill and Contractors Hill some 650 feet high, and on to Paraiso, and will be about 300 feet wide.

The French dumps, which are immediately on top of the banks of the canal at Culebra, have caused considerable trouble owing to the soil descending into the prism in the wet season, and to prevent this as much as possible a large amount of the old dumps has had to be removed a second time by shovels. This action was also necessary, due to the widening of the canal at that point.

During 1908 some 37,000,000 yards of dirt was taken out and 35,000,000 cubic yards in 1909, being an average of some 3,000,000 cubic yards per month in the last 2 years.

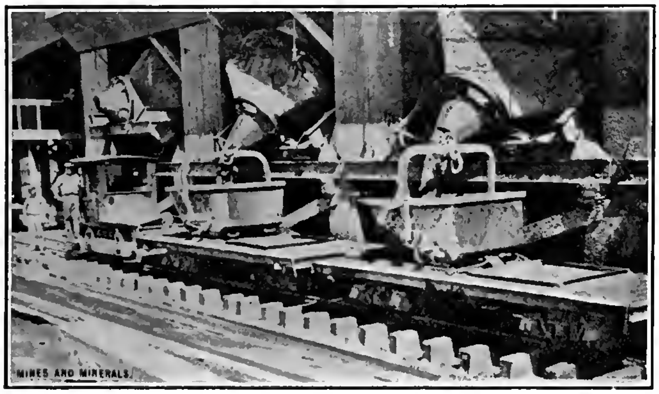
The other important locality is Pedro Miguel where the locks on the Pacific side are being constructed. Miraflores is the town adjoining.

These locks have a chamber 1,000 feet long by 110 feet wide

and are being constructed somewhat upon the same lines as the Gatun locks, except that the lock is single. Vessels passing through this lock from the Pacific will be lowered 30 feet into a a basin 1 mile wide, which is 55 feet above sea level on the Pacific side. This lock is 5 miles from the Pacific entrance at Balboa and the entrance is naturally protected.

An interesting feature at Corozal are some concrete barges, made to carry hydraulic pumps for excavating on the Pacific Division.

These barges are 64 feet long and of a 24-foot beam, being 5 feet 8 inches in draft. The walls are built of ½-inch mesh No. 12 wire cloth, which is stretched on ½-inch iron rods that are placed 8 inches apart. The concrete in the form of a plaster was applied to the frames, and the interior of the hulls suitably protected. This form of barge was experimented with by the Italian Government some years ago with great success.



Concrete Mixers Dumping into Buckets, Gatun Locks

On the Pacific entrance the channel is completed almost up to the lock site at Pedro Miguel and vessels from the west have plenty of water, and at Gatun in the big lake there will be anchorage for the whole navy. All west-bound vessels will be able to travel some 32 miles at a fair rate of speed from the Gatun lake to the Pedro Miguel basin.

The Atlantic entrance is at sea level and has an average depth of 41 feet of water and an average tide of about 12 inches. On the Pacific the tides range from 22 feet and the channel is dredged to 45 feet of water.

When at full tide the Pacific is 10 feet above the Atlantic, which was one of the reasons why it was impossible to have a sea-level canal. The Chagres River now partly flows into the Gatun lake and when the present barriers are removed and the river permitted to flood the lake bed, its waters will fill the entire canal, the surplus leaving by the Gatun spillways.

The area between Gatun and Gorgona, on account of low ground, will be entirely submerged and will become part of the Gatun lake, and very little excavation is required in comparison with the higher altitudes. A channel has of course been excavated the entire distance, and when the lake district is flooded, the towns of Tabernilla, Bohio, and Lion Hill will be deep under water.

An enterprise of this character is not free from accidents, but in consideration of the conditions under which the work is carried on and the large number of different nationalities employed, the casualties are very small.

Abandoned French Bucket Dredge, Showing Jungle Growth

Every day hundreds of tons of dynamite are used, the traffic on the railroads being exceptionally heavy, land slides, floods, and a hundred and one other causes make the work hazardous. The total working force is about 37,500 men throughout the various divisions, each of whom completes an average of 8 hours per day, with the exception of the Fire and Police Departments who are always on duty. Little improvement could be made in the organization of the force, and each division cooperates together and all runs like machinery.

Much credit is due to the Sanitary Department, under which the Medical Division, consisting of hospitals, dispensaries, etc. are operated, and the Inspectors Division that maintains a system of sanitary regulations, and sees that they are properly enforced. This department employs a large corps of men engaged in the extermination of the mosquito, and thousands of gallons of crude oil are expended for this purpose.

Handling Concrete Buckets, Gatun Locks

The following table of distances via Colon will show the number of miles that will be saved by using the Panama Canal:

From	Via Cape Horn	Via Canal	Distance Saved
Pensacola to Panama	11,421	1,401	10,020
Gulfport to Panama	11,443	1,423	10,020
Mobile to Panama	11,452	1,432	10,020
New Orleans to Panama	11,461	1,441	10,020
Galveston to Panama	11,477	1,457	10,020
Charleston to Panama	11,576	1,556	10,020
Savannah to Panama	11,582	1,562	10,020
New York to Panama	11,977	1,957	10,020
Philadelphia to Panama	11,956	1,936	10,020
Boston to Panama	12,161	2,143	10,020
Norfolk to Panama	11,775	1,755	10,020

A great factor toward keeping up the good health of the employes is the Cristobal ice plant operated by the Subsistence Department, without which it would be almost impossible to carry on the work, especially in a climate as unhealthy as this one. An ice plant is really as much an essential to the success of a mining or railroad venture in these climes as is any part of the equipment.

The great migration to the Pacific coast in 1849, following the discovery of gold in California, acted as a strong incentive to the immediate establishment of an isthmian route to avoid the long and hazardous journey across the western territories.

Thousands of persons made their way across the Isthmus on foot to Panama and from thence up the coast

to the gold fields. Few if any of the old time prospectors seemed to have tarried along the Isthmus, although deep in the heart of the jungles there can be found a few ancient workings, such as shallow vertical shafts. It is thought that most of these are Indian workings but none of them have ever been thoroughly examined, as the parties who ran across them were unable to spend much time in that vicinity.

It was thought that the exploration of the canal would uncover an extensive mineral field, but so far no minerals have been discovered in such quantity as to be of any commercial value. Gold, copper, manganese, and iron are known to exist in small quantities, also large beds of lignite. The country is yet to undergo a thorough geological survey.

On the island of Muerto, near David, on the Pacific coast, is said to be almost a solid mass of coal covered with a clay stratum. In 1851 a geologist by the name of Whiting made a report of this deposit which was published in London, England.

It is thought that the regions out beyond the 10-mile boundaries will be worthy of examination as soon as the canal is completed and the necessary transportation thus provided. Prospecting in such rank undergrowth is frightfully unhealthy and otherwise dangerous, it also being necessary for parties to cut their way through every step they take, and travel is therefore very slow.

As stated in the July issue of Mines and Minerals, the country is nothing but a big volcanic outburst and in many instances is bottomside up. The views accompanying this article, many of which are quite recent, will give a better idea of what is being accomplished here at the present time.

A MEXICAN RIVAL TO THE PANAMA CANAL

The *South American Journal* (London) reports that the new transport route across the Isthmus of Tehuantepec is proving successful beyond all expectation, and that those connected with its administration are now hopeful that by the time the Panama Canal is completed the Mexican line will have secured a preeminence in certain trades which it will be somewhat difficult to overthrow.

"Since the railway was opened 2 years ago, the traffic between the Atlantic and the Pacific coasts has steadily increased, as goods are handled at through rates for transport between the United Kingdom and continent, the United States, Canada, South America, and the Orient.

"To and from Salina Cruz, the Pacific terminal of the line, no fewer than seven steamship companies operate, while from Puerto Mexico, the Atlantic terminal, there are eight steamship lines, including the Hamburg-American, Royal Mail Steam Packet, Leyland, Harrison, and French transatlantics running, and arrangements are being made for others. At both these ports the most up-to-date facilities for handling cargo are now in operation, and so successful have been the results that work is about to be taken in hand for the doubling of the railroad track under an arrangement with the Government of Mexico. Transport from Seattle and San Francisco to Europe is now done in about 6 weeks, which is practically as fast time as that via the United States railways, and then across the North Atlantic by express steamer.

"With a view to saving the expense of passing through the Suez Canal, it is said that one of the leading Japanese steamship companies intends to take advantage of the Tehuantepec route—a course which, while avoiding the long journey around India and through the Mediterranean, will, however, necessitate connections on the Atlantic."

Assuming that the *South American Journal* knows what it is printing, it seems strange that an American railroad 48 miles in length across the Isthmus could not compete with the Mexican railroad across Tehuantepec approximately 175 miles long. May be it is a political utility.

CUBAN MINING DEVELOPMENT

Minister John B. Jackson, under date of June 25, 1910, reports from the American legation at Habana, the following measure designed to stimulate the exploitation of Cuban mineral resources:

The Committee on Agriculture, Industry, and Commerce, of the Cuban House of Representatives, has reported favorably a bill exempting mining claims and properties, under exploitation or not, from all national, provincial, and municipal taxes for a period of 10 years, and extending this exemption for a full 10 years to all mining claims and properties put in operation at any time within the original 10-year period. During a period of 30 years all vessels entering Cuban ports in ballast, coming from any foreign or Cuban port, and which clear with cargo of minerals or other products of national mines, are to be entitled to the remission of one-half of the port and tonnage dues paid on entry, and during a similar period minerals and mineral products are to be exempt from payment of export duties. During a period of 10 years, machinery, apparatus, and railway material imported into Cuba for use in mining and metallurgical industries are to pay no more than the minimum duty prescribed for similar articles when imported for use in the most favored Cuban industry.

In the preamble to this bill reference is made to the fact that Cuba is to a great extent dependent upon her export of sugar and tobacco, and attention is called to the desirability of "pluralizing" her production, especially by facilitating the development of her mineral wealth.

The following table by Francisco I. de Vildosola, Secretary pro tem of Agriculture, Labor, and Commerce, gives an idea of the number and area of the Cuban mines, with concessions in force on December 31, 1907:

Mineral	Pinar del Rio		Habana		Matanzas	
	Number	Area Hectares*	Number	Area Hectares*	Number	Area Hectares*
Asphalt	28	911	29	956	30	1,459
Coal	6	578	8	859		
Copper	23	1,206	5	413	3	97
Gold	(†)	(†)	1	64	1	125
Iron	31	2,140	4	189	3	500
Manganese					1	110
Petroleum	4	274	2	108	3	632
All other minerals	4	76				
Total	96	5,185	49	2,589	41	2,983

Mineral	Santa Clara		Camaguey		Oriente	
	Number	Area Hectares*	Number	Area Hectares*	Number	Area Hectares*
Asphalt	12	189	16	218	7	253
Coal	2	52			9	521
Copper	24	459	27	1,274	223	7,745
Gold	(†)	(†)			20	537
Iron	14	500	51	4,100	271	44,999
Manganese					218	11,364
Petroleum					3	405
All other minerals	39	2,828	3	54	106	6,843
Total	91	4,028	97	5,646	857	72,667

* Hectare = 2.471 acres.
† Included in "All other minerals."

There are other mineral riches absolutely neglected which are no less important than those mentioned, and which will prove great sources of wealth. Thus vast deposits of iron of very good quality remain unexploited, and there are extensive peat beds which at some future day will be utilized as fuel and in the production of nitrate.

The Cuba Railway runs from Matanzas to Santiago and is the backbone of Cuban development. From the backbone ribs extend wherever the developments will warrant. The latest rib is from Marti to Bayamo and Manzanillo.

IRON ORES OF SANTIAGO, CUBA

*By E. B. Wilson**

In Volume 13, page 613, of the Transactions of the American Institute of Mining Engineers, will be found an able paper on the Iron Ore Range of the Santiago District of Cuba.

Location and Extent of the Deposits, Methods of Mining and Transportation

It was written by Prof James P. Kimball, who introduced the comparatively new theory of the formation of ore deposits as the results of epigenesis and the circulation of meteoric waters. The paper was also the first public one on the structural geology of that part of Cuba, and this article is largely an abstract of the Kimball, Souders,† and Lawrence‡ papers.

The Sierra Maestra extends east and west along the southern coast of Oriente Province, Cuba. West of the Bay of Santiago the metallic minerals are copper and possibly others; to the east as far as the Bay of Guantanamo they are iron and manganese oxides.

The south slope of the mountain range for a distance of at least 30 miles is made up of eruptive syenite, covered with a massive flow of epidotic diorite. The serrated crest of this range of mountains within the 30 miles appears at a distance to be uniformly denuded of diorite, as are also the foot-hills of the coast. Along the coast are submerged coral reefs, and in the interior mountain district are emerged coral reefs resting directly on the syenite. The Bay of Santiago is excavated from coralline, having a thickness of 300 feet. The so-called Juragua Hills are the culmination of the foot-hills of the Sierra, and are in four parallel ranges, successively declining in elevation from the north toward the sea, on a general slope. The summits of the first range of the Juragua Hills reach an elevation of 2,000 feet above sea level and are mainly eroded of diorite down to the syenite.

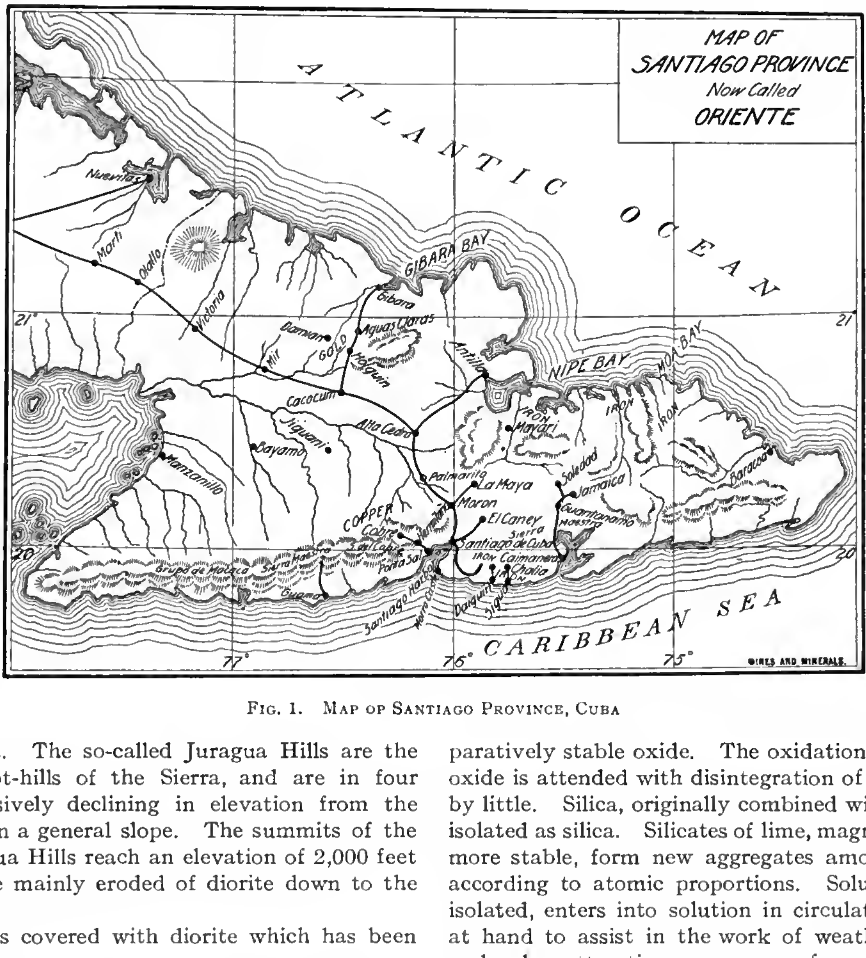


FIG. 1. MAP OF SANTIAGO PROVINCE, CUBA

The second range is covered with diorite which has been eroded into the syenite.

The third and fourth ranges are completely denuded of diorite if they were ever covered with it. These latter hills form the syenite coast margin, conspicuous features of which are the numerous dikes of epidotic diorite.

The first and second ranges present the more favorable conditions both for the original development and for the preservation of iron-ore deposits. The Lola and Berraco Hills to the eastward are an extension of the second range under changed structural conditions due to erosion, and contain undeveloped ore bodies, as do the Sigua hills still further to the east.

The second range of hills contains the best ore bodies, as is exemplified in the East and West mines of the Juragua Iron Co., and this Mr. Kimball terms "the second or lower iron-ore belt," to distinguish it from the iron-ore belt of the first or upper range.

The lower iron-ore belt contains large bodies of hematite, associated ferruginous aggregates—in part of magnetic oxide, which are secondary products from the decomposition of basic eruptive rocks now represented by epidotic diorite which has penetrated and overflowed the syenitic base of this part of the Sierra Maestra.

The iron oxide deposits are the result of the alteration of highly basic, and therefore unstable rock aggregates, and of their resolution into new aggregates, with the aid of surface agencies in subordination to the new conditions met with at and near the surface.

Thus the presence of atmospheric oxygen and of alkaline and earthy bicarbonates from decomposing silicates has rendered unstable the highly ferriferous material from deep-seated sources. These aggregates were basic from excess of protoxide bases and especially protoxide of iron.

Such eruptive material cannot long resist further oxidation at the surface, with the result of more or less complete disintegration of the original aggregate and its recomposition into new compounds.

The eruptive material which gave origin to the iron ore consisted of protosilicates, or silica combined with the protoxide bases iron, lime, and magnesia, and with alumina. Under the new conditions and in circulating waters charged with oxygen, bicarbonates of the alkalies and alkaline earths, the protoxide of iron became further oxidized into ferric oxide, which is a comparatively stable oxide. The oxidation of the ferrous to ferric oxide is attended with disintegration of the eruptive rock little by little. Silica, originally combined with the ferrous oxide, is isolated as silica. Silicates of lime, magnesia and alumina, being more stable, form new aggregates among themselves, in part according to atomic proportions. Soluble material, as fast as isolated, enters into solution in circulating waters, and is thus at hand to assist in the work of weathering. By the law of molecular attraction a process of concentration has gone on simultaneously with weathering.

Homogeneous material, such as ferric oxide, has collected by itself to a degree far greater than the other earthy residues, because in the process of conversion from ferrous to ferric oxide it has been in solution, and so in circulation, and hence becomes finally deposited. This explanation will be completed by the following observation:

The best of the iron-ore deposits are mainly, if not indeed wholly, replacements of coralline limestone.

The eruptive flow, which probably took place in part below the sea level, involved masses of littoral coralline limestone and of coral reef. The final elevation of the Sierra Maestra was a subsequent event. It is these masses of coralline limestone that have been replaced by the concentrated ferric oxide. Thus,

*Compiled for MINES AND MINERALS.
† Mineral Deposits, Santiago, Vol. XXXV, T. A. I. M. E., June, 1910.
‡ Two Cuban Mines, Canadian Mining Institute.

the outline of the iron-ore bodies, as well as their dimensions, corresponds to those of the original limestone masses.

The process of the replacement of limestone by ferric oxide is explained as follows:

Meteoric water circulating within the weathering zone of rock takes up enough carbon dioxide from the atmosphere and other sources, also sulphuric acid from the oxidation of pyrite,

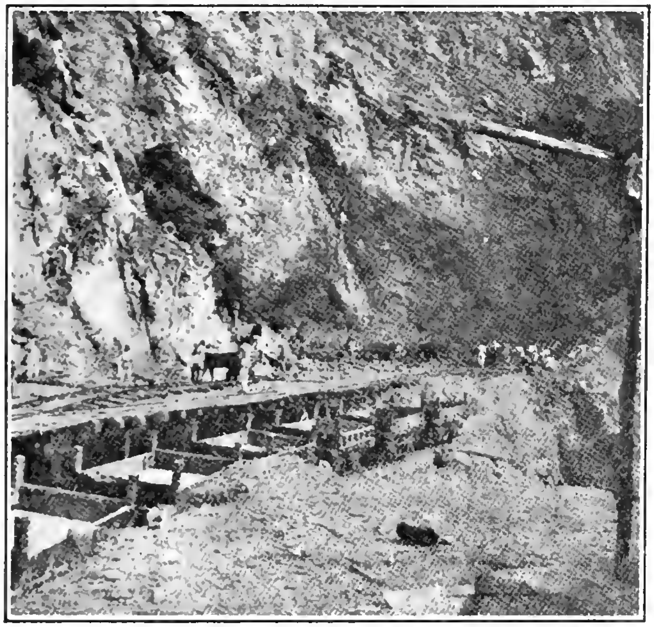


Fig. 2. Ore Face, East Mine, Juragua Iron Co.

to impart to it solvent effects, especially in its action on limestone. Coralline limestone before consolidation yields readily to such solvents. As particles enter into solution their places are taken by any available substance in the process of precipitation from the circulating water.

Lime becomes dissolved in water that has taken up ferrous oxide as a bicarbonate and that is precipitating ferric oxide. Or again, through the mediation of sulphuric acid, ferrous carbonate results from the reaction of ferrous sulphate upon limestone, and by exchanging acids and bases with alkaline carbonates also upon dioritic and syenitic rocks which supply these carbonates including carbonate of magnesia. Or still further, carbon dioxide with ferrous sulphide may follow from the reduction of ferrous sulphate by decaying organic matter.

Proof of the replacement of coralline limestone is afforded by fragments of ore still retaining the structure of coral. Other indications also quoted to establish the proof of this theory are incorporated in the paper, among which it is stated: "The presence of organic life is indicated by the pyritous ore possessing the structure of corallum at the northeast mine, which is near the contact of one of the ore bodies with the syenite."

The Sierra Maestra ores are derived without intermediate stages of development from basic silicated aggregates rich in ferrous oxide, an important portion of which has been preserved as ferric oxide in places below the surface near its source, without ever reaching the channels of drainage.

The ore deposits, so far as known in 1885, extended from Sigua, on the east, to Sevilla, on the west, a distance of some 18 miles; in 1909 they reached to Guam, 40 miles west of Santiago. The ore occurs in all forms of red hematite, including micaceous, amorphous granular, and subcrystalline varieties, the last named having minute crystals of magnetite and martite.

At Camorones, 56 miles from Santiago, near Guantanamo Bay, there is said to be a deposit of 68 per cent. magnetite. In the northern part of Oriente Province there are large deposits of brown iron ores which, however, have not been worked.

Variations in the color of drillings from red to black imply unequal distribution of altered hematite or of unaltered magnetite with water of hydration and manganic oxide.

Proximate analyses of these ores are as follows:

Moisture	.240	.810
Silica	5.000	10.500
Iron	61.000	68.500
Phosphorus	.009	.065
Sulphur	.045	.248

Mr. Kimball next theorizes on the causes which produce a lenticular outline to the ore deposits and says:

"All sections of ore bodies, whether longitudinal, transverse, or oblique, if projected to terminal outlines, result in approximately elliptical figures."

There is a second kind of ore that has been formed in prismatic blocks, the genesis of which he explains as follows:

The concentration of ferric oxide *in situ* from basic traps differs from the process of replacement of limestone from external sources chiefly in point of time required in the two cases. The circulation of chalybeate waters through dense trappean rocks would be so retarded as to induce the peroxidation of the ferrous oxide *in situ*, or at least soon after entering into solution. Its greater insolubility as compared with the protosilicates when liberated from their original bond and especially with quartz under certain conditions of temperature and pressure, ultimately leads to the elimination of all but difficultly soluble silicates. These still remain as earthy impurities of this imperfectly concentrated mixture of ferric and magnetic oxides. Magnetic oxide as a stable or difficultly

Fig. 3. Tramming in Cuba

soluble mineral originally present as such in basic eruptives has likewise been left as a residuum. This occurs in a mixture with ferric oxide. The occurrence of iron oxide in both types of deposits mainly as ferric oxide, and not as ferric hydrate, is the result of gradual dehydration more or less complete.

The first range of iron-ore deposits occurs within the foothills next below the summit of the Sierra Maestra. The ore

bodies here are ferric oxide, more or less magnetic from admixture with magnetic oxide. Nothing has been done to explore any of these ore bodies. The bulk of the specimens from the surface indicate they are inferior to the second-class ore of the Juragua East and West mines.

The first iron mine in Cuba was denounced in 1861, although mining did not commence until 1883. In 1884 the Juragua Iron Co., Ltd., exported 25,295 tons of ore, and in 1908, 366,580 tons.* The mines, which are situated northeast of Siboney, are connected by 24 miles of narrow-gauge railroad with a steel dock in Santiago harbor. The railroad runs southeast to

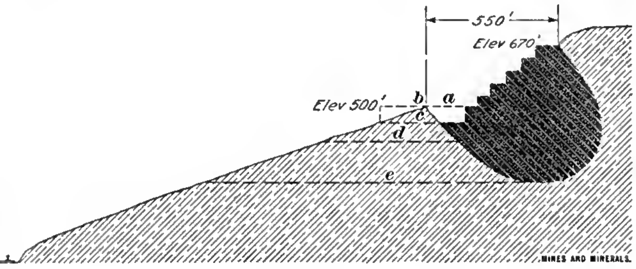


Fig. 4

Siboney, then skirts the shore to Fort Aguadores, thence a northwesterly course to the docks in Santiago harbor. The ore is mined in open cuts by benches 40 to 50 feet high, though at the East Mine of the company, Fig. 2, the face is 200 feet high. The ore in this case is loaded in cars, trammed, as shown in Fig. 3, to the loading plant and dumped into the cars that take it to the dock.

The Juragua Iron Co. owns or leases the Sevilla, Juranasito, and Firmeza deposits, besides the East, West, Columbia and Colon mines.

While the ore is shot down from the benches so as to fall by gravity to the bench beneath, it will be noted from the cross-section, Fig. 4,† that when the step *a* is excavated below the brow of the incline *b*, the ore must be hoisted or *b* tunneled as at *c*. The objection to tunneling is that with each excavation the tunnels become longer, as shown at *d* and *e*, and more expensive to drive. The method adopted in working these side-hill ore deposits is best shown by Figs. 5 and 6, which are taken from photographs loaned by Benjamin Lawrence, to illustrate this article. The ore is trammed from the floor of the steps to bins on the incline. From the bins the ore goes by gravity to the railroad bins at the foot of the incline. This mining is done by powder and steam shovels, the tramming by mules, and is apparently easy and economical mining on the lower steps shown. At the top of the hill the steps above the brow of the incline are worked the same as those on the side hill and the ore is sent by gravity to the inclines. Below the brow a much different state of affairs is found to exist as shown, in the relief cut, Fig. 6, which without doubt depicts open-cut mining more intelligently and forcefully than it could be described in words. Where the ore has been excavated below the brow of the deposit the slope up which the material is hoisted is shown. On this slope is shown an ore bin into which the ore from the bench above, but still below the brow of the deposit, is dumped prior to being loaded in a skip and hoisted up the slope. On the bench next above, mule haulage takes the ore to the ore bin on the incline. On the next bench above, a steam shovel is shown loading cars which are trammed around the hill to the incline, and so on work progresses to the top benches from which ore is lowered to the level of the incline.

The Sigua Iron Co. mine, about 36 miles east of Santiago, was opened in 1892, and connected with Playa, the sea port, by rail. According to the United States Geological Survey reports the company shipped 20,438 tons in two years and closed down. Harrison Souder‡ says "the ore is reported to be 'red specular hematite' containing 58.40 per cent. of metallic iron; 13.36 per cent. of silica, and .02 per cent phosphorus."

The Spanish American Iron Ore Co. made a shipment of 74,991 tons of iron ore in 1895. It operates the Lola, Magdalena, Providencia, San Antonio, Berraco, and Fausto ore deposits, and is said to have acquired the Sigua Iron Co.'s property, in the Sierra Maestra iron district. In addition to this it has more recently purchased and developed an extensive ore body at Mayari, on the north coast, with a port 12 miles away, at Nipi Bay, connected by railroad. This body of iron ore is also in Oriente Province, almost directly north of the Sierra Maestra deposits. Figs. 5 and 6 are illustrations of the Sierra Maestra mines, and the description need not be repeated. From the bins at the foot of the incline the ore is dumped into 10-ton hopper-bottomed cars and hauled over a broad-gauge railroad 5 miles to Daiquiri, where it is dumped, crushed, and stocked. By erecting several trestles, Fig. 7, to act as storage bins, a large quantity of ore is on hand to be loaded in steamers, thus avoiding delay; further, the regular shipments from the mines may be continued in case storms prevent steamships from being loaded. Steamships are able to carry from 3,000 to 5,000 tons of ore, and to ship 500,000 tons of iron ore, as the Spanish-

Fig. 5. Working a Side-Hill Deposit

* Mineral Resources of U. S., 1908.

† After Harrison Souder, Vol. XXXV, Trans. A. I. M. E., page 319.

‡ Trans. A. I. M. E., Vol. XXXV, page 314, Mineral Deposits of Santiago, Cuba.

American company did in 1907, requires from 100 to 150 steamer loads. The ore is carried from the trestle bins on an endless belt, Fig. 8, moving on a cantilever extending out to deep water, where it is delivered by gravity to the hold of the steamship, as shown in Fig. 9. More than 50,000,000 tons of iron ore have been taken from the Spanish-American Iron Co.'s mines in Cuba and there is considerable still in sight. Various analyses of the ore are given; that which follows is an average for the year 1902 taken from Mr. Souder's paper: Iron, 62.250 per cent.; silica, 8.050 per cent.; phosphorus, .032 per cent.; sulphur, .080 per cent.; copper, .300 per cent.

The copper is associated with ores from the Magdalena Mine; 1.73 per cent. of chromium and 1.04 per cent. of nickel is found with the ore at Mayari, and manganese is found associated with the brown ores. Impurities such as these are not objectionable where steel is to be made from the iron produced.

In 1900 the Cuban Steel Ore Co. opened mines at Guama, 40 miles west of Santiago, and in 2 years shipped 41,241 tons of ore, after which there are no further records of shipments. The operations were evidently intended to mine and ship ore on a large scale, for extensive docks were constructed, and a broad-gauge railroad built; whatever the cause for the cessation of shipments seems to be known to but few, as no reason has been publicly given for the abandonment of the enterprise.

The best miners on the island are imported by the iron-ore companies from Spain. They are guaranteed work for several years at a specified wage. Mr. Lawrence in his paper on "Two Cuban Mines," furnishes the following information on the Mayari mines, near Nipi Bay, on the north coast:

This iron ore is peculiar inasmuch as it occurs directly on the surface to a depth of some 30 feet, and is said to be the result of a decomposition of the serpentine and the enriching of a zone of iron ore.

A complete analysis of a general sample gave the following results:

	Per Cent.
Iron, after drying to 212° F.	46.030
Silica	5.500
Alumina	10.330
Chromium	1.730
Phosphorus	.015
Hydroscopic water	31.620
Nickel	1.040

The plant is capable of handling about 5,000 tons per day at an estimated cost of $1.01 as follows:

	Cents
Mining	11
Hauling	15
Nodulizing	75
Total	$1.01

Fig. 6. Open-Cut Mining, Sierra Maestra Mine

The ore is apparently a secondary enrichment, and covers the entire plateau for some miles. It is estimated that there are one billion tons of ore in sight.

The handling of this ore is interesting. It is first steam-shoveled into railroad cars of standard gauge of 50 tons capacity. These railroad cars are brought to the top of an incline and lowered 3 miles. Here the ore is dumped by electric crane into a stock pile, and from there taken by a clamshell shovel to the furnaces, where the water is driven off and the ore is nodulized. The ore is heated by coal to a high temperature, water is driven off, and the material is loaded into vessels to be shipped to the United States.

Large deposits of iron ore exist in the vicinity of Moa Bay, back from Punta Gorda, on the north coast, but still in Santiago Province. The owners furnished the following analysis: Iron, 58.070 per cent.; phosphorus, .011 per cent.; chromium, 1.740 per cent.; sulphur, .150 per cent.; and claimed 22,000,000 tons. They also stated that by building 150 feet of dock they would obtain 40 feet of water. Moa Bay is between a reef on the ocean side and the main land.

The acquisition by the Bethlehem Steel Co. of the United States of an important iron-ore deposit located near Santiago, Cuba, is a feature in the development of the mineral resources of the Republic. The ore beds have been measured up by engineers as embracing 75,000,000 tons, a pecularity of the deposit consisting in the fact that it contains 2 per cent. nickel and 1 per cent. chromium. The tract covers an area of 875 acres and lies about 12 miles east of Santiago. It is regarded by experts as the most important discovery of iron ore made within 20 years. Other large iron deposits of recent discovery are to be

worked at Felton, Cuba. These mines are in the Mayari Mountains and are the property of the Spanish-American Iron Co. A large deposit of iron ore was also discovered in Sagua de Tanamo during the last half of the year.

FIG. 7. ORE STORAGE BINS AT DAIQUIRI

During the fiscal year 147 mines, covering an area of 21,880 hectares, were surveyed in the Republic, 97 of which were in the eastern part of the island.

PEBBLE INDUSTRY IN NEWFOUNDLAND

Consul James S. Benedict, of St. John's, states that the Newfoundland government has given a concession for handling the pebbles on its shores to a company which is shipping them to the United States.

"The government agrees to grant to the Atlantic Pebble Co. for 3 years from May 1, 1910, the exclusive right to export pebbles and beach stones from the colony, and also for 25 years from the same date the exclusive right to enter upon any crown lands situated inland within 100 yards of high-water mark along the shore of Conception Bay, between Cape St. Francis and Splint Point (near Bay de Verde), to search for pebbles and beach stones, but their sole right of export is restricted to Conception Bay. The company's present operations are at Long Pond, near Manuels, 15 miles from St. John's.

FIG. 9. ARRANGEMENT FOR LOADING ORE INTO STEAMERS

"After 3 years, exploitation of pebbles for the rest of the island is open to everybody, and only in Conception Bay has the company any exclusive right. During the 3 years all machinery which cannot be made in the colony, and all sacks and sacking imported by the company for use in its operations, will be admitted free of duty.

"Only a certain kind of quartz pebble is required, and they vary in circumference from 3 to 9 inches and are divided into three grades. They are packed in sacks weighing 168 pounds each when filled. The first cargo of 1,650 tons, valued at $13,200, was shipped to Philadelphia on July 14."

Flints and flintstone were imported into the United States in the fiscal year 1909 to the value of $230,000, almost all from France, Denmark, and Belgium. Only $60 worth came from Newfoundland. This flintstone has been mostly used for grinding cement, being placed in rotary cylinders with partly crushed cement rock. An excellent report on flint pebbles appeared in Daily Consular and Trade Reports for December 20, 1909, and in the monthly edition of February, 1910.

Quartz pebbles find use in the paint industry as well as the cement industry. The tube mills attached to cyanide mills

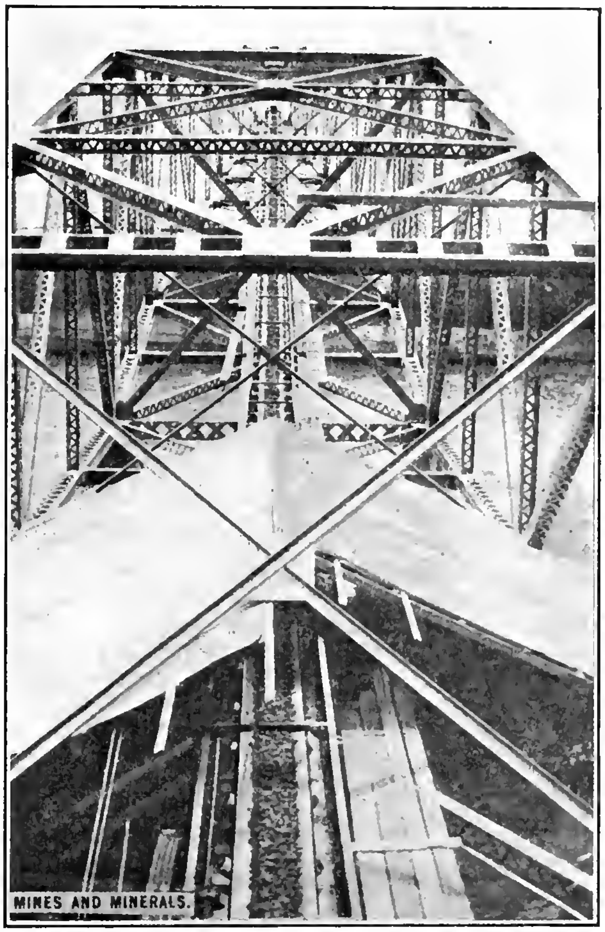


FIG. 8. CANTILEVER AND ENDLESS BELT FOR LOADING ORE

for the recovery of gold and silver, use large quantities of these pebbles, and the Danish stones being considered the best for this purpose are shipped all over the world. Each terminal morraine in the eastern United States contains suitable quartz pebbles for cement grinding, and nearly every cyanide mill can obtain suitable stones from streams in its vicinity. In many cases the quartz ore from the mine is used in large pieces to comminute the ore that has been crushed by other machinery. The ore in such cases is not rounded, although it soon becomes so in the tube mill, which is nothing more than a large tumbling barrel. Along the shores of New England from New York City to New Brunswick there are places where large quantities of excellent quartz pebbles can be found. In Northeastern Pennsylvania and other states where the ancient glacial rivers terminated, similar material embedded in sand will be found. Where particularly good cement work is desirable crushed quartz is the best material for the purpose.

ORE MINING NOTES

When Judge Dietrich in the federal court at Moscow, Idaho, ruled that mining companies operating in the Cœur d'Alene district have the right to dump tailing into streams, a decision of prime importance became a matter of record in Northern Idaho. Unless reversed by the Court of Appeals the question is settled for all time, and thus removes one of the most vexing situations which the mining companies have had to face. These cases have been pending 6 years. The suit upon which the decision was rendered was instituted by Elmer Doty, a rancher on the Cœur d'Alene River. There were 65 cases, but all of them involved the same principle, the damages claimed amounting to $1,223,000. Doty alleged that by dumping the tailing in the river the mining companies caused the river to rise and overflow the lands along the river. The case on trial was against the Bunker Hill Mining, Milling, and Smelting Co., and testimony was heard in nine of the cases. The mining companies did not introduce any evidence. Judge Dietrich said there was not enough evidence to warrant giving the cases to the jury, but instructed that a verdict be returned giving the nine plaintiffs the sum of $1 in all. The other cases against the Bunker Hill and Sullivan Co. were thrown out of court. The cases against the Federal and Larson & Greenough companies were continued until the fall term.

Mining operations are active in the St. Joe country on the north side of the Cœur d'Alenes. The Booster group, which consists of 7 or 8 claims, located north of town, had a crew of men on the property until the fires stopped them. They have a high-grade copper proposition. The Red Squirrel, located 3 miles west of Avery, has a large low-grade gold property. Nine claims have been located along the strike of the ledge, and they also worked all summer until the fires came in. Twenty-seven miles above Avery, on the river, several of the claims of the old St. Joe Quartz Mining Co. have been relocated and are being worked by Blake Miller. A strike of gold was made there last spring and as a result the mineral showing has been well staked by the Millers and J. A. Haggerty. The White Spruce group is one of the best prospects in the district. They have exposed 4 feet of milling galena and are working now about 8 or 10 men. Three parties of the geological survey are working there, principally concerned with the examination of the Northern Pacific Railway Co. land grant, to see that no mineral is included in the railroad's lien land selections.

Shortly before leaving White Horse, W. D. Greenough, of the Atlas Mining Co., White Horse, Yukon Territory, entertained Frank Oliver, Minister of the Interior for Canada, who spent a day looking over the mine and plant. He gave him a great deal of information regarding mining operations in the Klondike district, and stated that the output of gold from the creeks around Dawson would in all probability be twice as large this year as last, the product being almost exclusively obtained from dredging operations. "Coal mining in the Yukon is still in its speculative stage," said Mr. Greenough, "nothing but prospecting work having been done as yet on any of the properties."

Paul H. Fitzgerald, of Nome, Alaska, writes: "This town is in the transition state, passing from a placer to a dredging proposition. Nearly all the rich ground is worked out. The high price of fuel $18 to $20 per ton for a poor quality of soft coal and a like amount for hauling it back to the Tundra, 15 or 20 miles, prohibits the working of an immense area of low-grade ground. Wherever the ground is not frozen the dredges handle this low-grade placer profitably. Eight new dredges were brought in this spring, and expect to be in working order before the season closes, which will make 14 or 15 dredges in the Nome district. These dredges each handle from 2,000 to 3,000 cubic yards of placer dirt in 24 hours."

The most important placer find in recent years in Alaska is at Iditorod near the head of navigation on the Iditorod River, a tributary of the Yukon River. It is probable that a good camp will be established there as soon as the country is opened, say in about 1 year. The reasons are that Iditorod is in a mineralized country, 40 miles from the Innoko country and just across the Kuskyguin divide.

Of the products imported from the United States into the Philippines, 37 per cent. of the total were derived from the minerals in the United States. Of the articles exported to the United States 100 per cent. were derived from agricultural products. No country can go forward until it is able to make use of its mineral products, and the Philippines under the development of mining engineers are making a beginning.

D. W. Shanks, General Manager of the Rio Plata Mining Co., in the Arteaga district of Chihuahua, Mexico, says: "We have run about 1,200 feet of tunnels in the mine since the first of January, and the vein of silver ore continues unbroken. We find that we can treat profitably ore that has 17 ounces of silver to the ton. Altogether, there is not any other mine being operated under similar conditions in Mexico where the cost of operation is so low as at the Rio Plata. A new tube mill has recently been installed at the cost of about $5,000. There is always some scarcity of labor in the planting and garnering seasons, when part of the workers leave the mine to devote themselves to their agricultural pursuits. This crippled us to a slight extent recently, but the men have begun to come back now, and we have a full force. Despite our great distance from the railroad, and the consequent necessity of using mules exclusively, the transportation of the bullion and concentrates across the mountains continues to go smoothly. The mine is 85 miles from the nearest railroad station, Sanchez, on the Kansas City, Mexico & Oriental Railway (The Stillwell Line), but the grading for 10 more miles of railroad in our direction is now completed. Eventually, the railroad will run within 20 miles of the mine."

The New Zealand Government is making a systematic geological survey of that country. Gold dredging has been rather quiet lately with no reason assigned. A large dredge has been made in England to deepen the harbor of New Plymouth which is sand and conglomerate bottom. In front of the buckets there is a revolving digger which loosens the stuff and enables the bucket to pick it up readily. Whenever large boulders are met they are worked around and then broken up by dynamite. The Mount Morgan Mine ore deposit has undergone a decided change recently. It now produces copper with some gold associated, and is no longer worked as an open quarry. The former rich deposit of gold nearer the surface was probably a concentration due to weathering and solutions.

Away back, when herds of buffalo grazed along the foothills of the western mountains, two hardy prospectors fell in with a bull bison that seemed to have been separated from his kind and run amuck. One of the prospectors took to the branches of a tree and the other dived into a cave, but immediately came out and the buffalo took after him again. The man made another dive for the hole and again came right out. After he went right in and turned around and came right out again several times, the man in the tree shouted:

"Stay in the cave, you idiot!"

"You don't know nothing about this yere hole," bawled the other. "There's a bear in it!"

Gold placer and lode mining in Alaska in 1909 showed marked progress in production as well as in preparation for larger operations. The production of gold was about $20,463,000, an increase of $1,170,182 (about 6 per cent.) over the output of 1908 and the largest yield since 1906, the year of greatest production.

The placer production was $16,322,000; the lode production, including gold derived from copper ores, was $4,107,363. The figures representing the product of the same kinds of mining in 1908 were $15,888,000 and $3,357,335.

EVOLUTION OF HOISTING

Written for Mines and Minerals, by E. B. W.

(Continued from October)

According to Galloway* the steam engine was suggested as a means for hoisting minerals from mines by Keane Fitzgerald, in 1758. The engine and machinery he advised must have been complex, as it was originally intended to work Doctor Hale's revolving ventilators, but by the addition of horses "it could raise 600 pounds of coal 600 feet in one minute, the horses moving at a walk." In 1763, Joseph Oxley patented a steam hoister, and the first machine was considered the greatest improvement since the invention of the "fire engine" for pumping. A second improved machine by Thomas Delaval, in 1765, is said to have drawn up the coal at the rate of a bucket a minute. Oxley's machine was used at Hartley's colliery, and Mr. Jars suggested that it would be better to use the steam engine to raise water for a water-wheel hoist than to hoist with it, and this system was afterwards adopted in the North of England. Smeaton held similar ideas owing to the difficulty of producing a satisfactory rotary motion by means of gears. James Watt visited the Oxley machine at Hartley colliery and remarked that it moved irregularly and sluggishly. Matthew Washbrough, in 1779, added an improvement to the Oxley engine in the form of a flywheel.

Development of the Steam Engine for Hoisting. Balancing Weight of Ropes

The application of the crank to the steam engine is clouded by several claimants to the first invention, it was patented, however, by James Pickard, of Birmingham, England, in 1780.

In 1782, Watt arranged to produce a vacuum alternately on each side of the engine piston, thus solving the difficulty of applying the steam engine to a continuous rotary motion. There is a record of one of Watt's rotative engines being erected at Walker colliery, Newcastle, in 1784,† which continued to work as late as 1863.

The Trevithick, or high-pressure engine was introduced in 1800, and on account of its simplicity and comparative cheapness was received with favor.

The first steam engine made in the United States was a Watt engine, and it was made and placed in operation at the Schuyler copper mine, near Bellville, the town now called Arlington, New Jersey.‡ It was used for pumping water from the Schuyler Mine. The first steam engines used for hoisting were attached to a walking beam, to which was also attached the pump rod. This kind of engine carried from 2 to 8 pounds of steam above atmospheric pressure. Fig. 16 is a cut of one of the old Watt engines, which were so well constructed that some of them are said to be in use in South Staffordshire at the present time. Loaded down with pump rods and working barrels they were not rapid hoisters, and in addition were geared making them second-motion engines. The combination of pump and hoisting drum was quite popular at one time in the metal mines of the United States, particularly in the iron mines of New York and New Jersey.

The first engines used for hoisting in this country were geared to pole pumps and carried a friction wheel on the main flywheel shaft. The drum, which was on a separate shaft, was arranged with a lever attached to an eccentric so that the pillar block would slide forward and a V-shaped friction on the drum would come in contact with the friction wheel on the main shaft. The engines were of the plain horizontal slide-valve type with flywheel, and geared so that the pump would make from 8 to 10 revolutions per minute. As the engines were not reversible the bucket or slope car was lowered by gravity and the speed of descent regulated by a wooden brake block, or a brake band. Steam could be used expansively in arrangements of this kind and was at the Scott Mine, at Lakeville, New York, under the supervision of Benjamin Moffatt, Sr., who for many years has been at this property.

It is now almost the universal practice, unless as previously stated, in the case of engines coupled to pole pumps, to use two engines coupled together by means of a common crank-shaft, the object being to have them arranged so that should one be on the center the other will be off and thus start the motion. In the old one-cylinder second-motion hoisting engines there was an arrangement whereby the eccentric rod could be lifted from the valve rod and the valves thrown by a lever until the engine got under headway, after which the eccentric was hooked up. Such an arrangement would be out of the question at a large mine; and even at small ones where single-cylinder hoisting engines, such as is shown in Fig. 17, are used, the engineer takes the greatest care to avoid stopping on the center, so that when given a signal to hoist there will be no delay. In all single-engine hoists the empty car, cage, or bucket, is lowered by gravity and as a precaution against accident the drum is supplied with hand brake and the engine has reversing gear.

The single-drum engine with the single engine hooked up

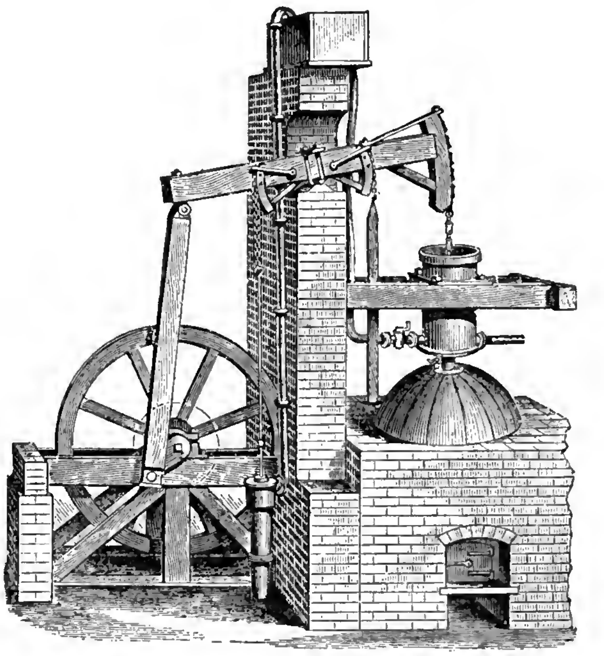


FIG. 14. NEWCOMEN ENGINE, WITH CRANK AND FLYWHEEL, 1780

to the pole pump presented some difficulties which were overcome by the engineer.

In case the engine stopped on its center, the eccentric was unhooked and the valve thrown by a lever until the engine got in motion, when the eccentric was hooked to the valve stem. In case the pump went "in fork", that is the water was pumped from the sump, the pump rod was disconnected from its gearing and the speed of the hoisting engine regulated by the throttle. The engineer, in any case used care not to stop his engine on the center, but just a trifle beyond. Of course in such cases it was impossible to balance the engine and it ran irregularly and at a speed commensurate with the speed of the pump. The skip was always lowered by gravity, and the output limited; however, these engines were the great machines in their day, and in one case a Corliss engine did the work.

Fifty years have made considerable difference in hoisting machines, and the engine shown in Fig. 16, which is a Watt engine, erected in a South Staffordshire colliery, and one of the first to which a crank was applied, made room for the slide-valve puffers as the high-pressure engines were called. In the next

* Annals of Coal Mining, page 277.
† Trans. N. E. Inst. M. E., Vol. 34, page 134.
‡ Trans. A. I. M. E., Vol. 5, page 168.

50 years even greater changes were made, as the output of the mines increased and the shafts deepened. Those mines which used the pole pump engine for hoisting, abandoned that and adopted a separate-geared engine and single drum, using the slide-valve engine to do pumping only. Such arrangements are still in use in some Missouri zinc mines.

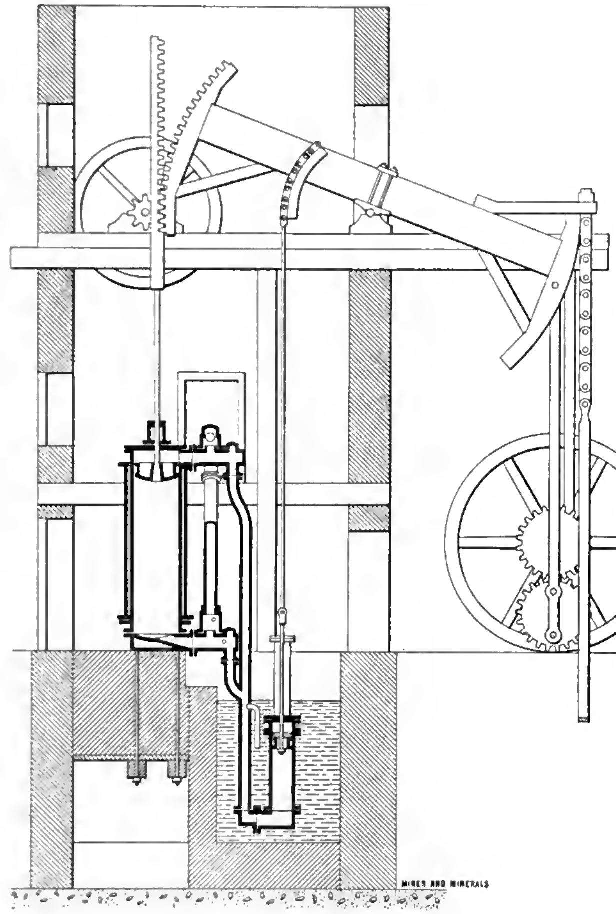


Fig. 15. Watt Double Acting Engine, 1782

The Lidgerwood second-motion hoisting engine, shown in Fig. 19, gradually displaced the slide-valve engine for hoisting, as it could be handled readily and raise a load much quicker, it also had other advantages from the hoisting standpoint that earlier engines did not possess, for instance, it could be run under or over, and hoist in balance, and could be braked by steam if necessary. At coal mines first-motion engines were always favorites, as hoisting must be done quickly to furnish a large output, and later this system was adopted at the deep ore mines Expansion engines have not come into general use at coal mines although they are in use at many ore mines, as well as compound tandem engines.

The requirements of hoisting engines are: that they must be thoroughly under the control of the engineer, that they may be stopped quickly when running at full speed; that they may be moved with certainty through a small fraction of a revolution that landing and caging may be done quickly; that they may attain full speed in two or three revolutions; in addition they must have strength to withstand the severe strains placed upon them. Until recently economy in fuel was not of much consideration at collieries, and compounding has not been practiced on hoisting engines, although in some cases expansion engines with high- and low-pressure cylinders have been coupled to the same shafts. These are not to be confounded with tandem engines, in fact they were two separate engines, one with a high-pressure and the other with a low-pressure cylinder. It was at first difficult to start these engines quickly, but this was rectified by placing an automatic throttle valve between the steam pipe and the low-pressure cylinder, thus giving both cylinders live steam at the start only.

Few mining engineers feel favorably disposed toward hoisting engines using steam expansively, although the conditions in connection with some ore mines make the economy in steam of prime importance, and not only are tandem hoisting engines installed but they are run in balance.

Balancing the load to be hoisted by means of two buckets was invented in ancient times when coggins and waterwheels were in use. However, such devices as flat ropes, cone drums, taper ropes, and balance or tail-rope are of more recent date. At first sight the question of balancing seems simple, nevertheless it has a number of features in connection with it that lend difficulties to its ready establishment, particularly where hoisting must be done from different levels.

Hoisting engines of the present time are driven by compressed air, electricity, or steam, and in their design it is necessary to know the depth of the shaft, the output per hour, and the maximum weight which can be lifted at each turn of the drum to which the engines are coupled. Usually two ropes are coiled on a drum in such a manner that one unwinds as the other winds. These ropes are attached to cages, and in their movement up and down the shaft are guided by sheaves at the top of a head-frame. The cages are so arranged that one is at the top landing of the shaft while the other is at the bottom landing,

Fig. 16. Watt Hoisting Engine

and they reverse their relative positions with each winding operation.

If the diameter of the drum, the weight of the rope, and the weight of mineral be known, a diagram may be drawn representing the resultant moments due to these weights for any position of the cages during hoisting. In Fig. 20, diagram a represents

hoisting from moderate depths when the weight of the mineral exceeds the weight of the winding rope; diagram *b* shows the conditions which prevail when the weight of the coal exactly balances the weight of the rope; and *c* when the weight of the rope overbalances the coal at the end of the hoist. Assume that the load the engine is to hoist is made up as follows:

	Pounds
Mineral	4,000
Car and cage	4,500
2,000 feet of 1½-inch rope	4,000
Total load to be lifted	12,500

It will be seen that the weight of the rope equals the weight of the mineral, and the total load is more than three times the

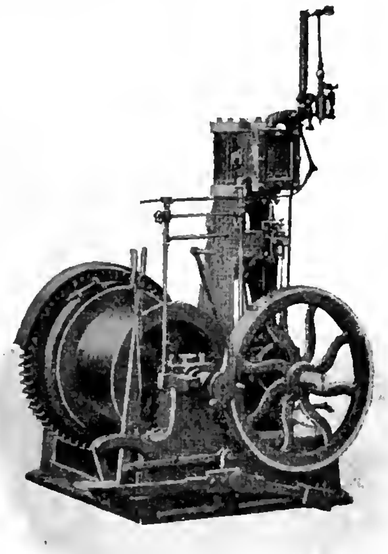
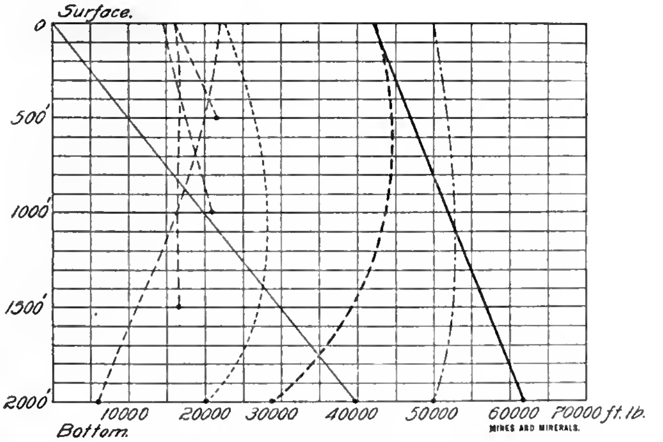


FIG. 17

weight of the mineral, and that if the latter is to be raised without balancing, a great deal of power will be wasted. The load at the bottom is 12,500 pounds, when it reaches the surface the

DIAGRAM OF STATIC MOMENTS FOR THE FOLLOWING CONDITIONS:

Ore	4,000 pounds
Cars, cage, and rope, from head gear to cage	4,500 pounds
Round rope	1½ inch diameter
Flat rope	4½ in. × ⅜ in.
Straight drum	9 feet 10½ inches diameter
Conical drum	8 feet diameter at small end
Conical drum	11 feet 9 inches diameter at large end
Reel center	4 feet diameter



Straight drum, hoisting singly, loaded
Conical drum, hoisting singly, loaded
Flat-rope reel, hoisting singly, loaded
Flat-rope reel, hoisting singly, empty, to change levels
Straight drum, hoisting in balance, loaded, from different levels
Flat-rope reel, hoisting in balance, loaded, from different levels

FIG. 18

weight of the rope is removed and the weight is 8,500 pounds; thus the hoister must be 40.4 per cent. more powerful than would be otherwise needed in order to overcome the weight of

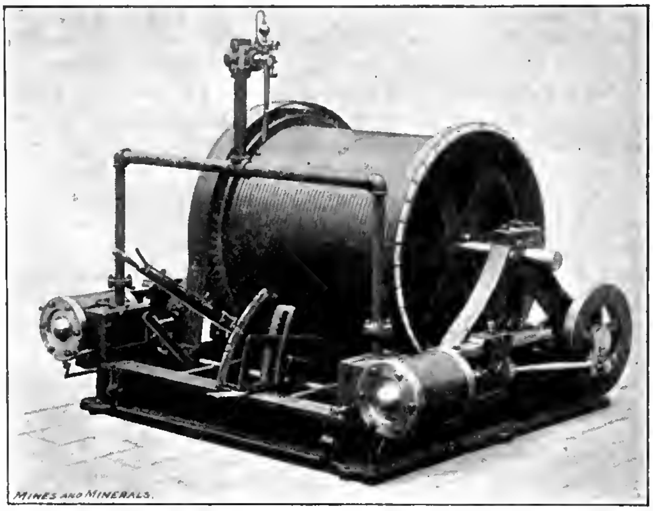


FIG. 19. LIDGERWOOD SECOND-MOTION HOIST

the hoisting rope. Different devices have been employed to overcome this condition, such as using two cages in a shaft; a tail-rope attached to the bottom of each cage, and extending to the bottom of the shaft; a taper rope so arranged that it is thicker and heavier at the top of the shaft than at the bottom; a flat rope wound on a reel and made to coil on itself; the conical drum; and the Koepe, and the Whiting systems of hoisting.

In this country the two systems of balancing most in use are the conical drum and the flat rope and reel. Both are based on the diameter of the drum or reel becoming larger and

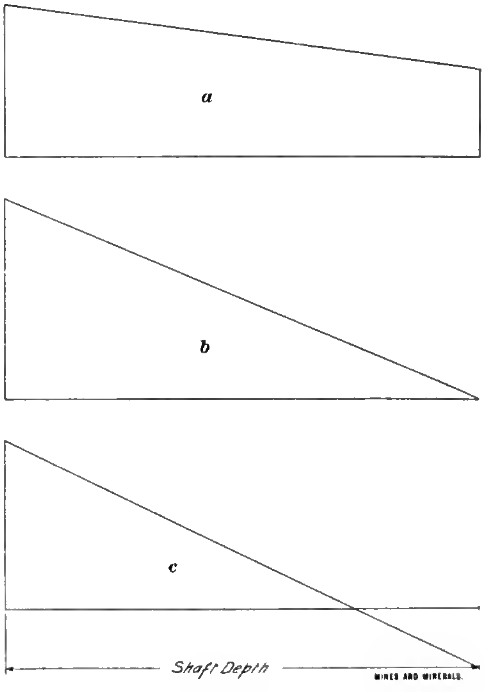


FIG. 20

larger as the rope winds up and thus offering more resistance to the engine.

When hoisting, the actual time during which the cages are moving may be divided into three periods as follows:* The time

* Arthur Whitten Brown, in Vol. 31, Trans. Manchester Geol. and Mining Society.

of acceleration t_1; the time during which the hoisting engine runs at full speed t_2; and the period during which the load is being brought to rest or the deceleration period t_3. During the acceleration period energy is stored in the moving masses and this energy must be returned in useful work or as heat during the decelerating period when the steam is shut off. As shown in Fig. 23 this stored energy *a b c d* may be represented as acting for a certain number of revolutions and having an area equal to the area *e f g h*, or that the stored energy has been used to complete the hoisting after the power has been shut off. By reference to Fig. 20 *b* it will be seen that the decelerating period will have to be longer than in the case of *a*, thus increasing the winding time and decreasing the output of mineral with a given cage loading. With *c*, deceleration is impossible and the stored energy must be disposed of in some other way, for example by a brake.

Whatever the shape of the diagram, the deceleration may be artificially increased by a friction brake, by admitting steam to the cylinder against the piston, and in one type of electric hoisting motor the energy may be returned to the generating system. Whichever method is considered, some waste is necessarily involved, and for economical reasons it is desirable that the hoisting engine should run out, that is, lose its energy, wherever possible.

As already seen in Fig. 20 *c* the engine will not, and in *b* may not, allow of deceleration so that the engine will run out in a certain fixed time. E. T. Sederholm* wrote an article for Mines and Minerals from which an abstract that has considerable bearing on equalization is taken. One method of equalization is by means of the cone drum, and to discuss this it must be understood that the duty an engine will perform depends on the size of the steam cylinders, on the steam pressure, the load to be raised, and on the radius of the hoisting drum. The larger the radius of the drum the greater is the leverage that the load exerts and the larger the cylinder will need to be to overcome that leverage. An engine works most advantageously if its load is constant and correctly proportioned to the size of the engine. It therefore follows that to obtain economy the variation due to the weight of the down-hanging rope must be equalized. With a cone drum, using the figures assumed, the relative weights to be lifted at top and bottom of the shaft are as 8,500 pounds is to 12,500 pounds, and the two corresponding diameters of the drum are in inverse proportion. If 8 feet is the desirable diameter for the load and rope selected, then the diameter of the large end would be $\frac{12,500 \times 8}{8,500} = 11$ feet 9 inches, and the average diameter would be 9 feet $10\frac{1}{2}$ inches. To wind the 2,000 feet would require 64.5 turns of the drum and that its length be 6 feet 9 inches long. Sufficient additional length should be given the drum to hold a couple of turns at the small end and a few extra grooves at the large end to provide for overwinding. The static moment to be overcome will be $\frac{12,500 \times 8}{2} = 50,000$ pounds at either top or bottom of the shaft. This static moment is not, however, constant during the whole lift, for when the rope has reached the middle groove of the drum, sometimes less than half the length of the rope has been wound up, while just one-half of the balancing effect of the radius has come into play. It follows then that the leverage of the rope on the drum will be slightly too great or the drum a trifle smaller at that point than that due to a true cone. The proper shape for the drum would be conoidal but that is out of the question as the shape then would be that of a conoid with a concave generating line. The truncated cone is therefore universally adopted as shown in Fig. 21, which is an engine at the Maxwell shaft of the Lehigh and Wilkes-Barre Coal Co , at Ashley, Pa.

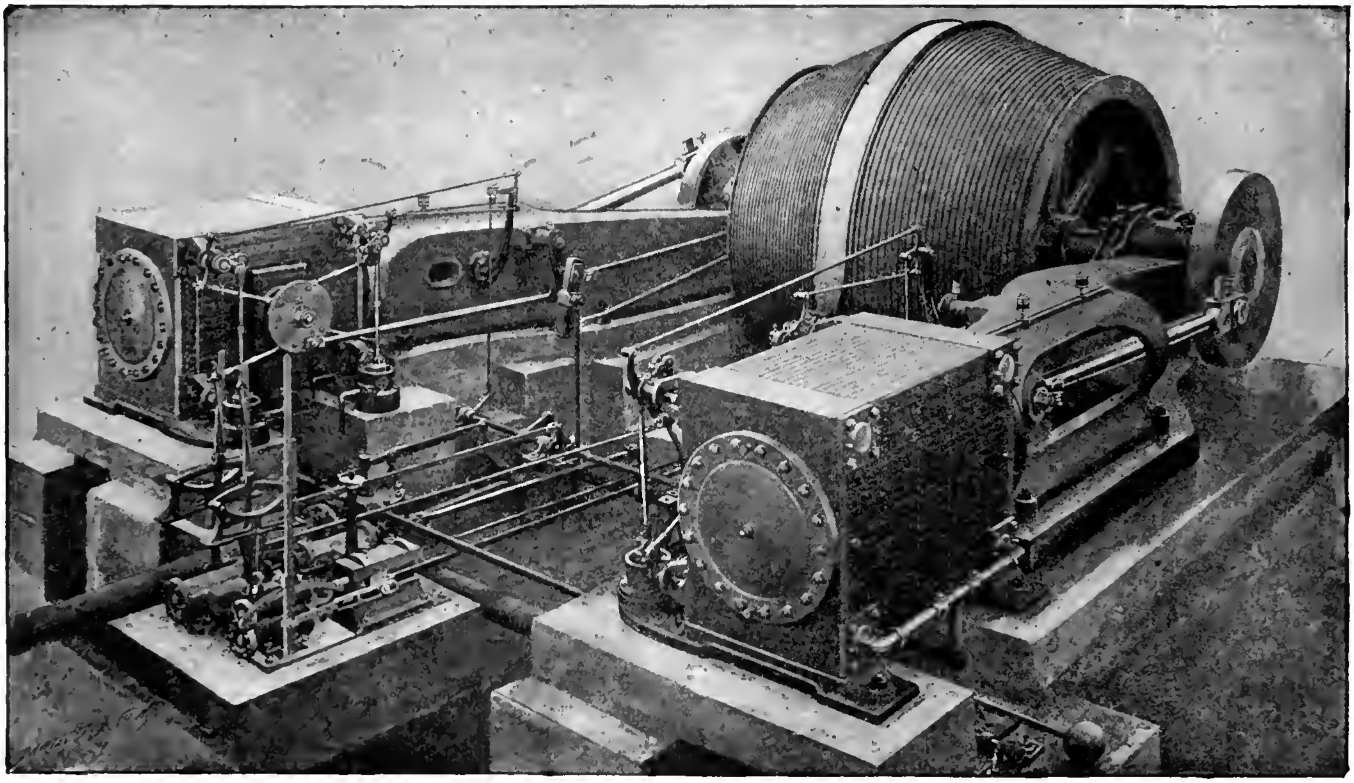


Fig. 21. First-Motion Hoist, Maxwell Shaft, Ashley, Pa.

In figuring out the different turning moments of the load on the drum it will be found that they vary. The diagram, Fig. 18, shows what they would be with the loads and dimensions given for a straight drum, conical drum, and flat-rope reel, hoisting singly, and for a straight drum and flat-rope reel hoisting in balance. From this it is seen that the maximum moment

*"Balanced and Unbalanced Hoists." Mines and Minerals, Vol. 24, page 577.

for the straight drum is at the bottom, while for the cone drum it is near the middle of the shaft. These respective moments are 61,800 foot-pounds and 52,900 foot-pounds, from which it is deduced that the cylinders of the straight-drum hoist will have to be about 13 per cent. larger than those of the conical drum, and with the cylinders of course all other parts of the engine. In the case under consideration the straight-drum engines would have cylinders 22×60 inches, and the conical drum engines cylinders 20×60 inches. The cost would vary approximately in the same ratio but so the economy, for the diagram shows little would be gained in coal consumption by the use of the conical drum. To gain in economy it will be necessary to do something more than equalize the weight of the rope.

Double hoists with conical drums show that when hoisting in balance less effort is required to start the load, but such hoists must always hoist from the same level. Under such conditions it is possible to proportion the drums to obtain a very perfect balance provided the hoist is built to always hoist in balance. Then it is not necessary to use clutch drums.

Perhaps the most common type of balanced hoist is the plain double-drum hoisting engine shown in Fig. 22, with cylindrical drums clutched to the shaft. In this hoist the cages and cars are perfectly balanced, while the rope is not balanced at all. Its proper place is therefore in mines which are not deep and where the effect of the rope is of comparatively small consequence. If the same conditions are assumed as before and drums of the same diameter are used, it will be found that 100 per cent. more power is required to start the load from the bottom than is needed when it reaches the top. This shows that while the straight drum is superior as far as balancing the cages and cars are concerned, it is inferior to the taper drum when it becomes important to balance a long rope. But the taper drum leaves very much to be desired if, as usually is the case, the hoisting is to be carried on from several levels. This has led to the adoption of the so-called tail-rope for balancing the rope itself. It consists of a rope of exactly the same diameter and weight, as the main hoisting rope, attached to the bottom of the cage and carried down to the bottom of the shaft, from which point it returns up to the other cage after passing around guide sheaves as shown in Fig. 24. With this arrangement it does not matter what position in the shaft the cages may occupy there is always exactly the full weight of rope dependent from each drum. One rope balances the other and one cage with its cars balances the other which is an ideal arrangement as far as balancing goes.

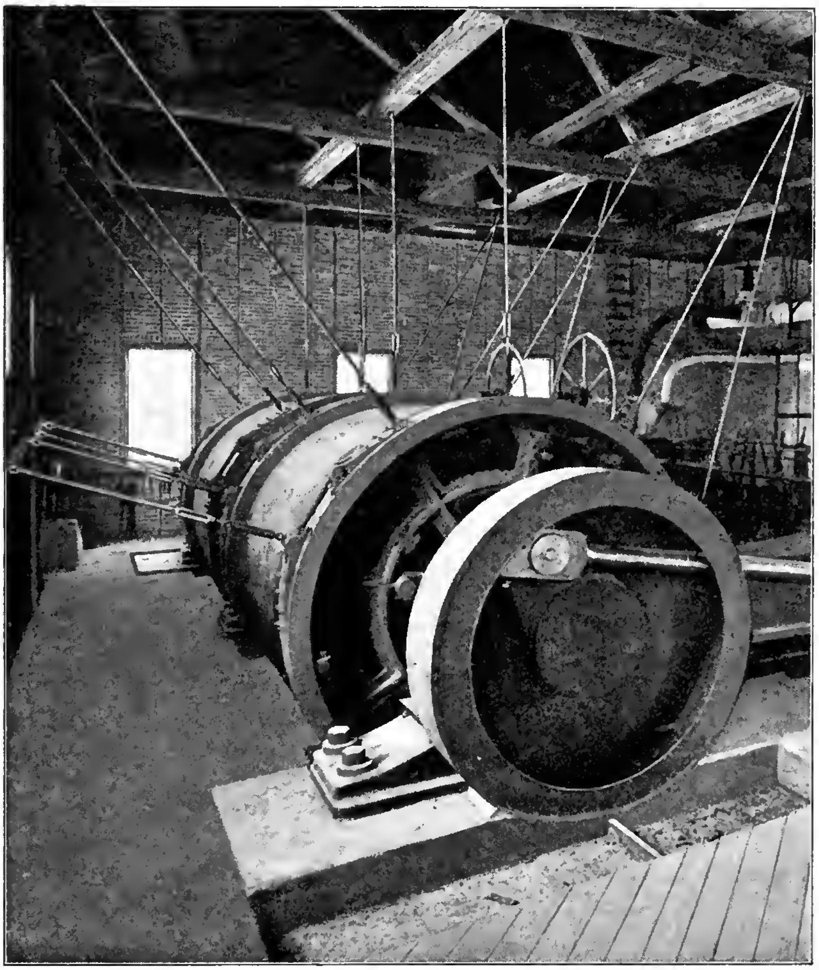


FIG. 22. SAVOY AND SIBLEY HOIST, 2,500 FEET CAPACITY, ELY, MINN.

As already stated and shown in Fig. 19 *a, b, c*, it is advisable to avoid as far as possible the use of means for artificially increasing the deceleration, and to this end attempt is made to arrange the winding moment so that it is sufficiently great at the end of the hoist to produce the required deceleration. If the weight per foot of the balance rope is equal to that of the hoisting rope, then the winding moment due to the load will not vary during a hoisting period and the diagram will become a rectangle. Such a diagram represents the greatest advantage to be gained by the use of a balance rope. If the balance rope is heavier than the winding rope the diagram will be more favorable to running out; and the stress on the winding rope will be increased, also a larger rope for the same factor of safety and perhaps a larger drum to accommodate the rope will be required. There are practical disadvantages that may outweigh the advantages of a balance rope; for instance, where it is necessary to hoist from different levels the timbering in the shaft must be arranged so as not to interfere with the tail-rope when hoisting from higher levels. This objection is not so serious, for all that is necessary is to make the timbers heavy enough to stand without partitions between the two hoisting compartments, and put the guide timbers on the long sides of the shaft.

Assuming* that the balance rope is of the same weight as the winding rope, then in Fig. 23, if the depth of the shaft be s feet, and $s_1\ s_2\ s_3$ feet be the distances described by either cage during acceleration, full speed, and deceleration, respectively, and the times $t_1+t_2+t_3$ be T seconds, the velocity in the shaft at full speed being V feet per minute, then the mean velocity over s_1 and s_3 will be $\frac{V}{2}$ feet per minute; and

$$V = \frac{120s}{t_1+2t_2+t_3} \text{ ft. per min.}$$

If the diameter of the drum is known V, may be replaced by N revolutions per minute and S by N_1 where N_1 is the number of revolutions made by the drum in completing one hoist. Similarly $n_1\ n_2\ n_3$ replace $s_1\ s_2\ s_3$, respectively, as shown in Fig. 25.

For a given speed of N revolutions per minute, the energy stored during acceleration is proportional to the sum of the moments of inertia of all the moving masses referred to, say the drum diameter. If this inertia be expressed as I = pounds

* A. W. Brown's Method.

weight × (diameter of gyration in feet)², then the winding moment M_a in foot-pounds required to produce a speed of N revolutions per minute in t_1 seconds is

$$M_a = \frac{I\,N}{1{,}220\,t_1}\text{ ft.-lb.}$$

Expressing the winding moment due to the load as M foot-pounds, then the time required for the hoisting engine to come to a state of rest is given by

$$t_3\text{ seconds} = \frac{I\,N}{1{,}220\,M}$$

Since for a given output of mineral per hour with a fixed weight raised at each hoist, the running time T seconds is fixed, the speed of winding will depend on the value t_2 and consequently the brake horsepower to be exerted in raising the load. The brake horsepower is found by the following formula:

$$\text{Brake horsepower} = \frac{M\,N}{5{,}280}\text{ during full speed; or}$$

$$\frac{(M+M_a)\,N}{5{,}280}\text{ during acceleration.}$$

Values varying from zero to .8 T may be given t_2 yet the winding engine will run out.

(To be continued)

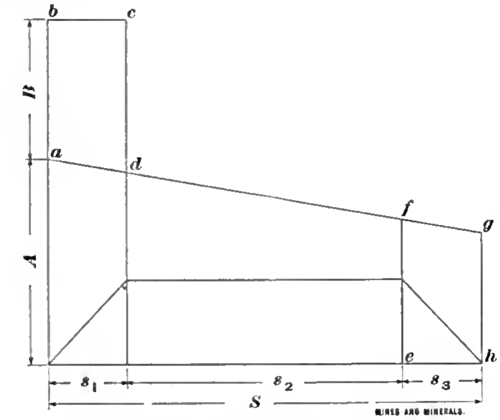


Fig. 23
A, Resultant torques due to load and stress; B, Accelerating torque.

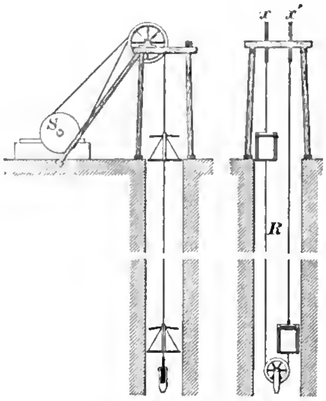


Fig 24

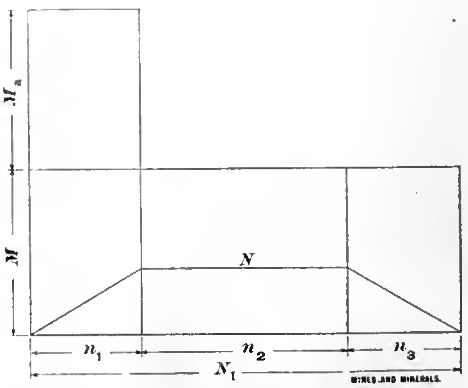


Fig. 25
N, Revolutions of drum per minute; N_1, Revolutions per wind; M, Torque due to load; M_a, Accelerating torque.

New Inventions

PATENTS PERTAINING TO MINING ISSUED AUGUST 30 TO SEPTEMBER 27, 1910, INCLUSIVE

Nos. 968,651, 968,652, and 968,845. Process of extracting metals from their ores, William E. Greenawalt, Denver, Colo.

No. 968,954. Metallurgical furnace, John E. Jones, New York, N. Y.

No. 968,881. Coal-mining apparatus, Andrew Powell, Uniontown, Pa.

No. 968,951. Ore concentrator, Ulysses S. James, Newark, N. J.

No. 969,081. Drill sharpener, Richard A. Schmidt, Bayard Station, N. Mex.

No. 969,048. Slate-picking apparatus, Frederick H. Emery, Scranton, Pa.

No. 969,044. Well-drilling machine, William D. Deschamp and George C. Jones, McAlester, Okla.

No. 969,186. Blasting cartridge, Gershom Moore Peters, Cincinnati, Ohio.

No. 969,151. Fuel cartridge, Thomas D. Bausher, Reading, Pa.

No. 969.812. Miner's lamp, Peter Togleson, Deepwater, Mo.

No. 970,409. Apparatus for impregnating logs, Sigmund Willner, Chicago, Ill.

No. 969,851. Mine prop, John H. Eickershoff, Dusseldorf, Germany.

No. 970,720. Heating arrangement for retort coke ovens Eugene W. King, Syracuse, N. Y.

No. 969,927. Ore roaster, Arthur R. Wilfley, Denver, Colo.

No. 970,325. Process of treating ores, Edwin B. Goodwin, Ward, Colo.

Catalogs Received

In writing for catalogs, please mention Mines and Minerals

General Electric Co., Schenectady, N. Y., Bulletin No. 4749, Alternating Current Switchboard Panels, 69 pages; Bulletin No. 4763, Isolated Plant—Direct Current Combination Generator and Feeder Panels, 6 pages; National Electric Code Standard Red Core; Tricoat; and 30-per-cent. Para, 20 pages.

A. S. Aloe Co., 513 Olive Street, St. Louis, Mo., Bargain Circular No. 48, 10 pages.

Jonathan Bartley Crucible Co., Trenton, N. J., Graphite and Mining in Ceylon, 10 pages.

Colorado Iron Works Co., Denver, Colo., Pamphlet No. 27, The Akins-Rothwell Agitating Tank, 4 pages.

Kolesch & Co., 138 Fulton Street, New York, N. Y., Richter's Precision Instruments, 8 pages.

Ingersoll-Rand Co., 11 Broadway, New York, N. Y., "Sergeant" Rock Drills, 24 pages.

Goldschmidt Thermit Co., 90 West Street, New York, N. Y., Reactions, 26 pages; Thermit Butt-Welded Pipe Joints, 4 pages.

Ohio Brass Co., Mansfield, Ohio, O-B Bulletin of Electric Railway and Mine Haulage Materials, 26 pages.

The Industrial Instrument Co., Foxboro, Mass., Bulletin No. 39, Industrial Dial Type Thermometers, 4 pages.

National Brake and Electric Co., Milwaukee, Wis., Publication No. 391, National Air Compressors, 68 pages.

Vulcan Iron Works, Wilkes-Barre, Pa., Vulcan Direct Acting Hoisting Engines, 114 pages.

Western Electric Co., New York, N. Y., Bulletin No. 9635, Hawthorn Commercial and Folding Type Mazdaliers, 20 pages.

Wright Mfg. Co., Detroit, Mich., Catalog No. 12, Wright Steam Specialties, 44 pages.

The Wellman-Seaver-Morgan Co., Cleveland, Ohio, Hoisting Machinery, Steam and Electric, 48 pages.

Ackroyd & Best, Ltd., Morley, near Leeds, England, Safety Lamps, 34 pages.

American Blower Co., Detroit, Mich., Bulletin No. 283, "Sirocco" Fans and Blowers, 16 pages.

Rockwell Furnace Co., 26 Cortland Street, New York, N. Y., Building M., Tilting Crucible Melting Furnace, 4 pages.

W. S. Rockwell Co., 50 Church Street, New York., N. Y., Rockwell Reverberatory Melting and Refining Furnaces, 4 pages. The Miller Gas Engine, 1910, 23 pages.

Mines and Minerals

Vol. XXXI—No. 5. SCRANTON, PA.—DECEMBER, 1910—DENVER, COLO. Price, 25 Cents

RETORT COKE OVENS IN MEXICO

Written for Mines and Minerals, by E. B. Wilson

About all the coal under development in Mexico is in Coahuila, tributary to the Mexican International Railroad.

Increased Percentage of Coke as Compared With Output of Beehive Ovens

The Sabinas coal field derives its name from the town near which the first coal mine in Mexico was opened in 1884.

The principal coal basins in this field, shown on the map, Fig. 3, are Sabinas, Fuente, Palau, Las Esperanzas, Saltillo, San Blas, and Lampacitos. Before a study of this basin was made by Jose G. Aguilera, Director of the Mexican Geological Survey, the coal measures were considered as belonging to the same system as the Laramie beds in the United States. Senor Aguilera found, however, that the fossils in these measures corresponded to the Montana division of the Upper Cretaceous in the United States. The Sabinas and Fuente basins are separated by what is termed the Peyotes anticlinal. The Las Esperanzas basin is separated from the Sabinas basin by a belt of hills $1\frac{3}{4}$ miles wide. The Saltillito basin is 5 miles south of Barroteran. The San Blas basin is 30 miles north of Obayos, but as this basin is not on the railroad it is not worked at present.

The Lampacitos basin is 1 mile north of Baluarte, on the Mexican International Railway. The town of Lampacitos and a general view of the coke oven plant is shown in Fig. 1. There is at this place a 6-foot bed of coking coal belonging to the Cia. Carbonifera Agujita, y Anexas, S. A. Until recently the Mexican coal, which in structure strongly resembles the semi-bituminous coking coal of the Pocahontas field, West Virginia, has been coked in beehive ovens, and in places subjected to dressing before being sent to the ovens. Owing to the low coke yields of this coal in the beehive oven, due to the fact that it is necessary to burn off a considerable amount of fixed carbon in order to generate enough heat to carry on the coking process, experiments were made in 1908 with a battery of three non-by-product retort ovens.

The first plant of retort ovens, built at Lampacitos, proved their ability to make good coke and a block of Koppers retort ovens were started, as shown in Figs. 2 and 3, the latter being a view of the foundation up to the oven yard.

At Lampacitos there are 36 Koppers retort ovens whose inside dimensions are 33 feet $1\frac{1}{2}$ inches long, 6 feet $10\frac{1}{2}$ inches high, 22 inches wide at the discharge end, and 19 inches wide at the pusher end. An analysis of the Lampacitos coal which is crushed, but not washed, previous to coking, is given as volatile matter, 17.5 per cent.; fixed carbon, 67.0 per cent.; ash, 15.5 per cent.

The theoretical yield of coke from this coal should be 82.5 per cent., or 1,650 pounds of coke for every ton of coal. In beehive-oven practice the coal produced about 60 per cent. coke, or 1,200 pounds per ton of coal. In other words, it required 1.66 tons of coal to make 1 ton of coke. It was found

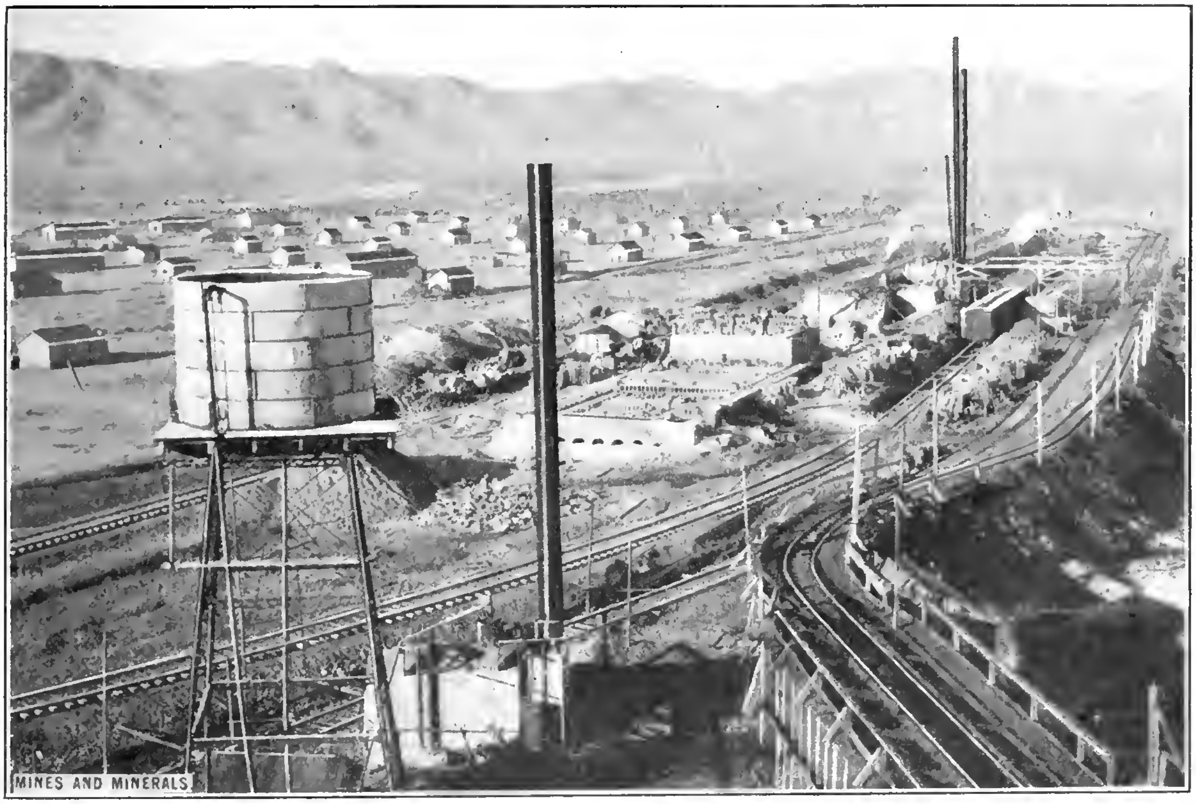


Fig. 1. Lampacitos, Mexico, Coke Plant

from the experimental retort oven plant that the coke yield amounted to 80 per cent. of the coal charged, or 1,600 pounds, thus economizing to the extent of 400 pounds of coal for each ton of coke made.

Mexican coal operators seemed to have become satisfied that if they could produce 5 tons coke where they produced four from the same quantity of coal there was conservation to be practiced, and lost little time in adopting retort ovens. The

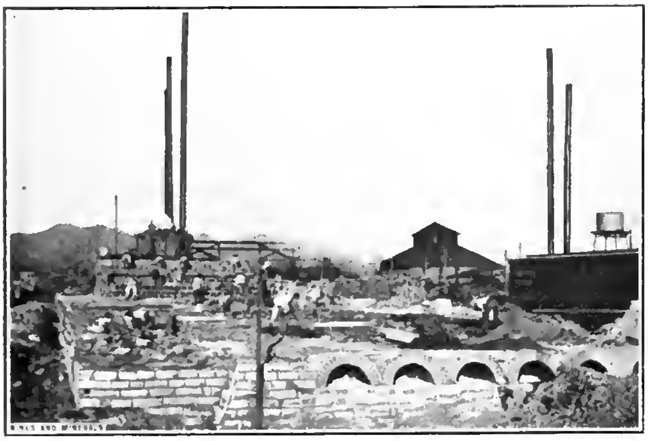


Fig. 2. Retort Coke Ovens Under Construction at Lampacitos

Mexican Coal & Coke Co., whose ovens are shown in the course of construction in Fig. 4, were next to adopt retort ovens, and have at present a battery of 50 of the same size as those at the Lampacitos plant, although differing slightly in flue construction. An analysis of the Las Esparanza coal is given as follows: Volatile matter, 21.1 per cent.; fixed carbon, 67.4 per cent.; ash, 11.5 per cent.; moisture. This is practically the same coal as that at Lampacitos, but is washed previous to coking. Theoretically, 1.27 tons of this coal should yield 1 ton of coke. However, in beehive ovens the yield is 60 per cent., thus making a difference between theory and practice of 820 pounds of coal per ton of coke. Owing to the ash in the coal remaining constant while the hydrocarbons are expelled in coking, it follows that since it requires 1 66 tons of coal to make 1 ton of coke in the beehive

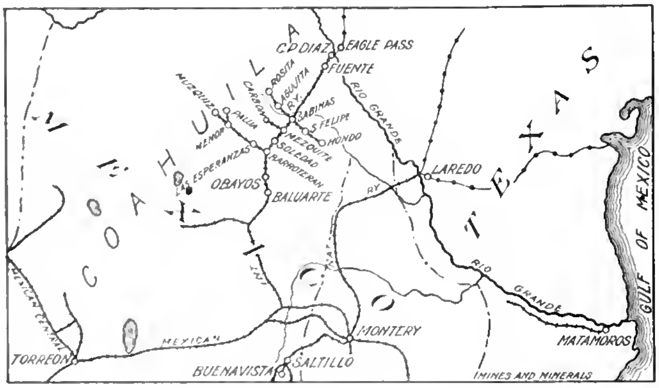


Fig. 3. Map of Sabinas Coal Field

ovens, the ash will increase from 11.5 per cent. in the coal to 19.09 per cent. in the coke. When it is understood that by the use of retort ovens the yield of coke from this coal is increased one-fifth and the ash is reduced from 20 to 15 per cent. in the coke, their value from an economic and metallurgical standpoint is evident.

The illustrations accompanying this article are of the first non-by-product retort ovens constructed on this side of the Atlantic, known as "Koppers waste heat ovens"; and are more descriptive in showing the progress and mode of construction than probably any hitherto published.

From the illustrations, the construction of beehive ovens seems simple compared with retort ovens. When, however, the crown of the beehive oven has been properly constructed there has been as much intelligence involved as anything in the line of experience required in retort oven building.

It will be noticed that in the Lampacitos ovens there are a series of flues under the retorts, and in Fig. 6 there is shown a large flue under the oven yard for carrying off the waste heat.

The flues under the ovens perform two services: (1) They raise the oven to the desired height above the loading track with less material. (2) They keep the foundation cool. The air entering at the outer end of these flues travels to a division wall in the middle of the battery where it passes through openings in the arch of the flues into checker work under the oven floor. It then works its way to the outer ends of the battery and is drawn off by a short stack. The flue under the pusher machine track collects the waste heat and leads it to the boiler plant.

The gases volatilized from the coal in these ovens are not treated for by-products, but are burned to heat the ovens. At Las Esperanzas the hot gases resulting from the combustion of the gas in the heating flues are used under three 150-horsepower boilers, and three more are being installed, making 900 horsepower of steam generated by the waste heat from 50 ovens. History states that retort ovens were first constructed for the purpose of increasing the quantity of coke from a given quantity of coal, and at the same time to accelerate the coking process. According to this, the saving of hydrocarbon and ammonium by-products distilled from the coal was an after thought which overshadowed the fact that in many cases it was not by-products but increased coke yield the operator desired.

In the beehive ovens the volatile matter in the coal is volatilized by radiated heat from the oven walls, and, by burn-

Fig. 4. Retort Ovens at Las Esperanzas

ing inside the oven, reheats the walls. The temperature of beehive ovens reaches at times 2,800° F., while that of retort ovens will vary from 2,100° F. to 2,200° F., depending on the temperature to which the air is preheated before mixing with the gas. It is probable that there is a greater loss of heat due to radiation in beehive ovens than in retort ovens, as more surface is exposed, consequently the higher temperature of the former is directly due to the combustion of the volatile matter in contact with the coal and firebrick. The

thickness of the walls of retort ovens is important because all heat for coking must pass by conductivity through the firebrick, and naturally the thinner the walls the more heat will be transmitted in a given time. The thickness of the retort walls in Koppers, vertical-flue, non-by-product coke ovens is 4½ inches, but strength is added to the ovens through the numerous flue partitions which are from 5 to 6 inches thick. When beehive ovens are watered, drawn, and left standing, it is a question of time until the bricks lose so much heat by radiation that the charge will not fire. To make good solid coke the charge must catch fire quickly. This is accomplished by keeping the oven hot and charging soon after drawing the coke. The retort ovens can be kept hot by closing the oven doors soon after the coke is pushed and by burning gas in the combustion flues, both of which add to the dispatch in coking. In Fig. 7 is shown the pusher side of the retort ovens under construction, and also the top of the vertical flues so constructed that cool air may enter for the combustion of the gas which takes place in the vertical flues. Another view of the same ovens further along in construction is shown in Fig. 5. In this illustration it is seen that the retort chambers are open at both ends and are separated by what are apparently brick walls but which in reality are partitioned inside so as to form combustion flues.

Retort chambers in general are made up to 34 feet in length; 6.5 to 7 feet in height, and from 18 to 24 inches in width; the latter width, however, is exceptional and requires about as much time to make coke as a beehive oven. Between the ovens are flues, placed either horizontally or vertically—in these particular ovens vertically—so that each flue may be supplied with a gas jet for heating the walls.*

In the Koppers non-by-product oven the gases given off from the coal pass directly into the vertical heating flues between the ovens, entering these at their upper ends and meeting cold air admitted through the dampered openings on top of the ovens. Where, in by-product ovens, the combustion takes place in these vertical flues from the bottom up, in the non-

FIG. 5. PUSHER SIDE OF OVENS AT LAMPACITOS

by-product ovens the combustion takes place from the top down, and the waste gases all enter the large flue leading to the boilers. In Fig. 8 the construction of the Koppers oven at Lampacitos is shown at that stage where two buckstaves are placed at each combustion chamber to prevent the heat expanding the walls to such an extent as to produce cracks and leaks. Each pair of buckstaves are yoked by tie-rods to the pair on the opposite side of the oven, and each buckstave is supplied with a gate hook to receive a gate bar which holds the oven door upright. It will be noted that from every alternate pair of buckstaves steel brackets are fastened whose object is to carry one of the rails on which the traveling crane moves. The oven doors are supplied with eyes at their upper ends to receive the hook suspended from the crane by a chain.

It is necessary to be able to remove and replace the heavy

FIG. 6. RETORT OVENS AT LAMPACITOS

doors quickly, and the traveling crane shown in Fig. 9 answers this requirement.

The Las Esperanzas ovens are charged with 8 tons of crushed coal, sometimes carrying as much as 12 to 15 per cent. moisture, owing to the coal being previously washed. After charging, the doors are put in place and kept tightly luted with clay, so that there will be no leakage of air into the ovens. The draft on the ovens is from 4 to 5 millimeters of water gauge.

To remove the coke from the ovens, traveling cranes, one on each side, remove the doors, as shown in Figs. 9 and 10, and the pusher shown in Fig. 9 rams the coke out of the oven, as shown in Fig. 10. The coke, which is uniform in texture, is broken by hand into lumps about 4 inches in diameter, and is then forked into cars. Mexican coke is consumed entirely by metallurgical furnaces, and although the supply does not meet the demand, the Mexican furnacemen insist upon the waste, resulting from breaking coke into lumps which will burn freely, being charged to coke production.

In Fig. 9 the various operations connected with coking on the pusher side of the ovens are shown; for instance, the crane is holding the door from the oven being discharged, while the larry is shown on top, charging an oven from which the coke has been removed.

TABLE 1

Coal	Moisture at 212° F. Per Cent.	Volatile Matter Per Cent.	Fixed Carbon Per Cent.	Ash Per Cent.	Sulphur Per Cent.	Yield in	
						Beehive Oven Per Cent.	By-product Oven Per Cent.
Connellsville	1.86	30.12	59.61	8.41	.78	64	72
Pocahontas	1.01	18.81	72.71	5.91	.79	62	85
Coahuila	1.60	15.00	67.64	12.01	.86	60	80

In Fig. 10 the door is shown suspended from the crane; the coke is on the oven yard just after it has been pushed from the oven; and the man in the rear of the hot coke is examining the interior of an oven to ascertain its condition.

Coal which is coked in beehive ovens loses some fixed carbon through combustion during the coking process and

* See Vol. XXXI, MINES AND MINERALS, page 185.

through combustion after the coke has formed. Apparently this loss follows in the inverse ratio to the percentage of volatile matter in the coal, other things such as ash being equal.

Taking as examples the Connellsville, Pocahontas, and Coahuila coking coals, and comparing the yields in coke in beehive and by-product ovens, this and other interesting conditions are observed. Table 1 gives the analyses of the three

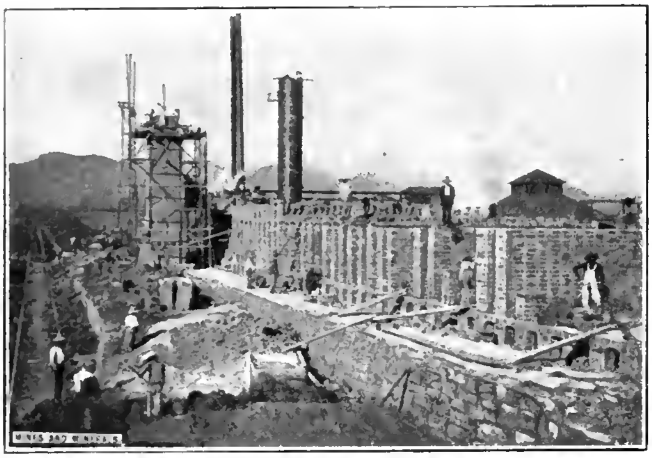


Fig. 7. Constructing Retort Ovens at Lampacitos

coals and their actual yield in coke per ton of coal in beehive and retort coke ovens.

Theoretically, Connellsville coal should furnish 68.38 per cent. coke; in beehive ovens it yield 64 per cent., thus it takes 1.562 tons of coal to make 1 ton of coke. When the same coal is coked in retort ovens it yields 72 per cent. of coke, thus conserving 340 pounds of coal for each ton of coke made, when compared with the product from beehive ovens. Pocahontas coal cokes in beehive ovens at the expense of some of its fixed carbon, and only under exceptional circumstances yields 62 per cent. coke. In retort ovens it has yielded 85 per cent. of coke, thus conserving 740 pounds of coal for every ton of coke made. At Joliet, Ill., where there are 280 Koppers regenerative by-product ovens, 80 per cent. Pocahontas coal is mixed with 20 per cent. Ronco coal, forming a mixture containing 22 per cent. volatile matter. The coke yield is 84 per cent. and the gas yield 10,000 cubic feet per ton of coal. Less than 50 per cent. of this gas is used to heat the ovens. The increased yield of coke in retort ovens with Coahuila coal has been stated previously in this article.

That coal coked in retort ovens should yield more than the theoretical quantity of coke is due probably to the decomposition of the volatile hydrocarbons, which adds materially to the fixed carbon, but the increase seems to be greater as the percentage of volatile matter decreases, as will be noted in Table 1, giving the analyses of the three coals. This is diametrically the reverse of beehive coking.

The object in coking coal is to supply fuel for blast furnaces that is high in fixed carbon and low in volatile matter, sulphur, and phosphorus. The analysis of a good furnace coke for Bessemer pig iron is about as follows: Fixed carbon, 89.55 per cent.; ash, 8.285 per cent.; volatile matter, .5 per cent.; moisture, .85 per cent.; sulphur, .8 per cent.; phosphorus, .015 per cent.

Volatile matter in coal, unless it be anthracite, prevents its use in smelting, for in addition to the large quantity of unconsumed gases created and the danger from explosions, the coal produces an agglomerate, if of the coking variety, and scaffolds if of the non-coking variety. Anthracite coal was long used in the East for the manufacture of pig iron and possibly may be mixed with coke and used at the present time, particularly where the cost of coke is high. Coke should not contain more than 1.22 per cent. in volatile matter nor more than 12 per cent. ash. The latter of course reduces the fixed carbon in the coke just 20 pounds per ton for each 1 per cent., besides it demands extra flux and heat to convert it into slag and entails extra expense for its handling and disposal. Moisture greatly lowers the heat of the furnace, and so annuls the heat of the fixed carbon to a certain equivalent extent.

Phosphorus in coal goes into the coke and will appear in a larger percentage in a ton of beehive coke than in a ton of retort-oven coke. The effect of phosphorus on iron is to make it brittle when the iron is cold, and it should not exceed .03 per cent. in coke used in blast furnaces, unless basic iron is desired. Sulphur should not exceed 1 per cent. in coke. Should a coking coal contain 2 per cent. in sulphur 1 per cent. will be oxidized to SO_2, and pass off as gas; the remainder, however, will be retained in the coke; but it requires more than 1 ton of coal to make a ton of coke, and the increase in sulphur will condemn the coke for pig-iron manufacture, although it may be employed for copper and lead smelting.

Although this article is on non-by-product retort ovens, as a whole it would be incomplete without mentioning the 60 combination retort ovens now being constructed by Mr. Piette for the Sabinas Coal Co., at Rosita, Coahuila. These ovens have horizontal flues instead of vertical and would therefore belong to the more expensive Semet-Solvay type, but so arranged that they can be changed from waste-heat ovens to by-product ovens, while the Koppers ovens, as constructed in Mexico, can be used only for waste gas. The 60 ovens of the Sabinas Coal Co. will utilize the surplus gas under six French boilers of the Babcock & Wilcox type, with superheaters, and capable of producing steam to the amount of 1,500 horsepower. At Rosita a steam turbine and central electric plant will be installed to distribute power to the various mines of the Sabinas company. The 50 ovens of the Mexican Coal and Coke Co., at Las Esperanzas,

Fig. 8. Constructing Ovens at Lampacitos

furnish steam for a 1,000-horsepower "Curtis" turbine, which furnishes electric power for the company's mines, performing all the work at the mines with the exception of hoisting.

The writer is indebted to Mr. R. D. Martin, of Agujita, for the illustrations appearing with this article; Mr. M. C. Scheble, general manager of the Compania Carbonifera Agujita y Anexas; Mr. Edwin Ludlow, general manager, Mexican Coal and Coke Co.; and Mr. W. E. Hartman, Joliet, Ill., for assistance rendered in this article.

THE STARKVILLE, COLO., EXPLOSION

Written for Mines and Minerals, by Our Special Correspondent

Method of Mining and Ventilation and Conditions Existing in the Mine Before and After the Explosion

The Starkville Mine of the Colorado Fuel and Iron Co., is situated at the town of the same name on a branch of the Atchison, Topeka & Santa Fe Railroad Co., some 4 miles south of Trinidad, Las Animas County, Colo. The population of some 1,500 is entirely dependent upon the mine for its livelihood.

The plant consists of a wooden tipple of about 1,500 tons daily maximum capacity, a 1,200-ton storage slack bin, coal-washing plant, 190 beehive ovens in blocks, and the usual repair shops, stables, etc., none of which was injured by the explosion. The tipple is some three-quarters of a mile from the mine, the coal being hauled around the hillside by a steam locomotive. The daily output, prior to the accident, was about 1,200 tons, over one-half of which was used to charge the ovens, the remainder being consumed by the Colorado Fuel and Iron Co. for various purposes.

The seam worked is known as the Starkville and is a member of the Laramie formation of the Cretaceous. Its composition, is, essentially, 35 per cent. volatile matter, 5 to 6 per cent. combined moisture and 6 to 8 per cent. of ash, with sulphur well under 1 per cent. When crushed and washed it makes a satisfactory coke.

The seam varies from 4 to 7 feet in thickness, averaging about 6 feet, and is clean except for a variable bone of from 1 to 6 inches some 2 feet above the floor. The roof, to which the coal adheres, consists of from 6 inches to 15 feet of slate to the sand rock, which, however, in numerous instances is immediately upon the coal. The draw slate is left up in both rooms and entries, and falls when wet owing to the probable admixture of a small percentage of lime. The floor is a hard slate. Seventeen feet below the Starkville is found another 6-foot seam, originally worked, but now abandoned by reason of two large and persistent slate partings. Fifty feet above is a smaller seam, not opened. The intervening strata consist of the usual shales and sandstones.

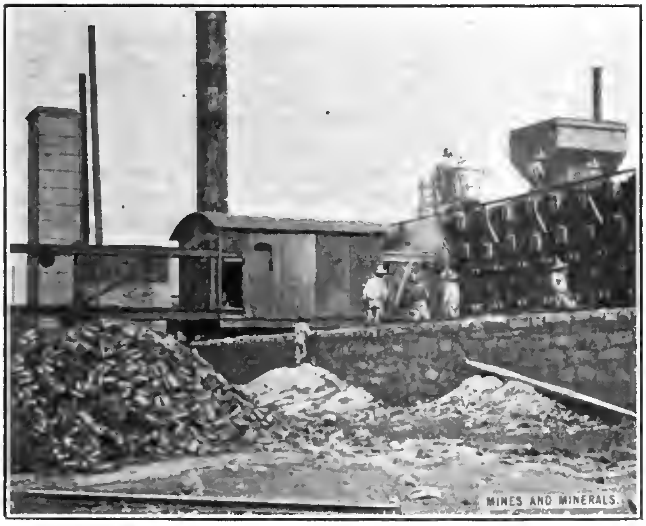


Fig. 9. Pusher Operating at Lampacitos Ovens

The covering above the coal averages a trifle more than 600 feet in thickness, running up to over 1,000 in places.

No powder is or was used in the Starkville Mine and the seam is essentially non-gaseous. The coal has a most pronounced fracture, breaking off in plates like the leaves of a book, 2 inches to 6 inches in thickness, and is shoveled out, even a pick not being generally necessary. This tendency to flake or break off is probably due to the pressure of the superincumbent rock. The mine is dry, but not unusually so, and the floors, but not the ribs or the roof of the entries were sprinkled daily by a watering car. Gas in small quantities has been found at rare intervals at the extremities of the new workings, but open lights were used and were permissible.

There are two openings about 300 feet apart at Starkville, the old and the new, although these form practically one mine.

The general plan of the workings is shown in Fig. 1. The original opening was made some 28 years ago when the mine was the property of the Atchison, Topeka & Santa Fe Railroad Co., and was for 1,100 feet driven single entry, but subsequently worked double, with a 10-foot air-shaft on the hillside, which was abandoned years ago.

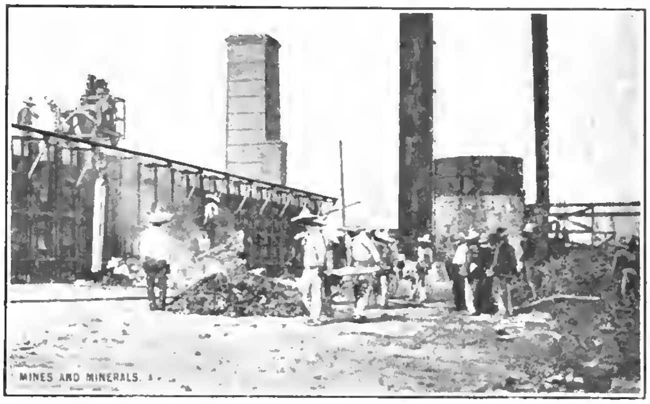


Fig. 10. Coke Charge Just Pushed From Ovens at Lampacitos

Forty-seven hundred feet in, are the third and fourth south entries, 2,400 feet long, driven over to entries M1 and M2 of the new mine. All the coal from these cross-entries to daylight is worked out and the pillars drawn. At the time of the explosion the new mine was being equipped with electric haulage and a 2-inch water line with hose connections every 75 feet for the purpose of watering the entries. Thirty-nine hundred feet further in is another pair of cross-entries, known as the seventh and eighth south, connecting the two mines as before. The coal between the fourth and seventh south entry is worked out.

There were three trolley partings or gathering places in the mine, one 8,500 feet in on the main straight road, another on the tenth south, 2,000 feet beyond, and the third on the new entry, M2, near its intersection with the seventh south, 9,400 feet from the drift mouth. The gathering was done by mules, and the hauling to daylight by Jeffrey motors, operated under 550 volts pressure, this being the oldest haulage plant in the western states. The course of the coal from the old mine was from the inner parting 8,500 feet in a direct line.

From the new mine the coal was hauled to the third south cross-entry, thence along this to the old mine and along the old heading to daylight, a total distance of 12,000 feet.

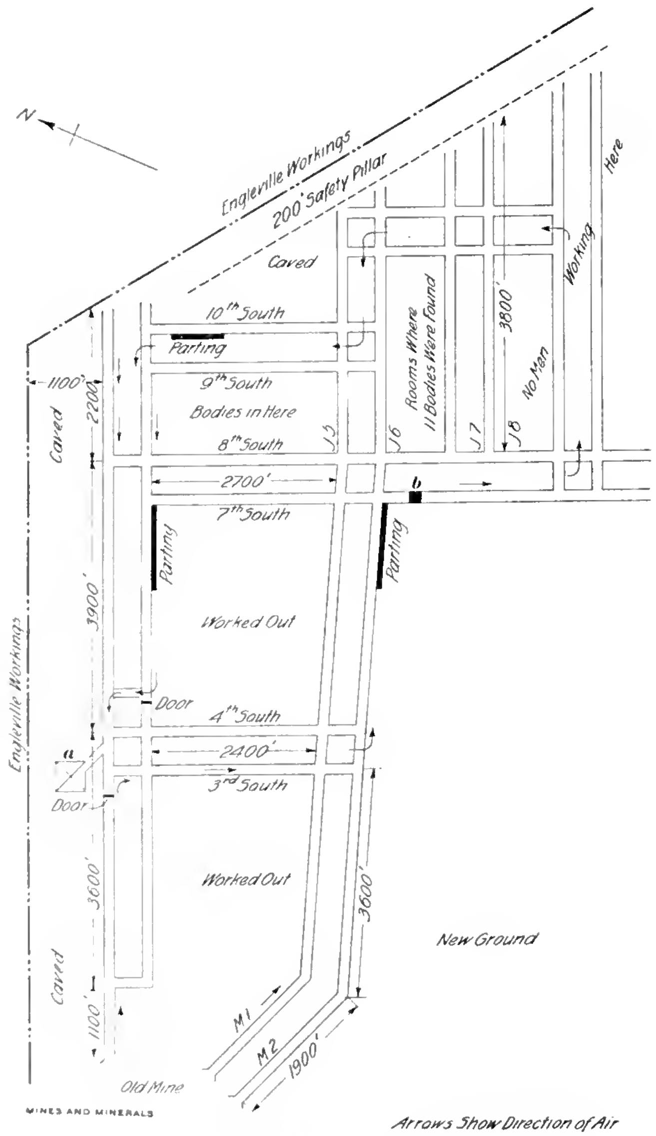
The 1,350 acres comprising the old mine was practically worked out and a new mine was being equipped. This mine is to draw its coal from the untouched area reached by an extension of the seventh and eighth cross-entries to the right, and from which extension butt entries on the original course of 78° E are driven. These new entries are known as J7 and J8, etc., and represent the only new work in the mine.

The old main heading extends 2,200 feet beyond the eighth south to the Engleville workings, the boundary between the mines extending diagonally across the field, it being 3,800 feet along the M1 and M2 or new headings to this other mine. The Engleville Mine is older than the Starkville by several years, but is still in operation, its workings extending parallel to those of Starkville on the left some 1,200 feet therefrom for a distance of over 2 miles. These workings were at one time connected, but owing to a fire some years ago, pillar drawing, etc., entrance

cannot now be had along this line between the two. As a rule, at the boundary at the end of both the old main and new main haulage roads a pillar of 200 feet has been left to protect the Engleville main heading, which here bends to follow the property line.

However, at the extreme end of the old main haulage, the two mines came together, but the openings have been bratticed off.

Ventilation.—On an average 200 men and 40 mules were employed under ground and were supplied with 60,000 cubic feet of air per minute. The ventilating system was extremely simple, only some 7 or 8 brattices and doors and no splits being used in the mine, the air making a complete circuit of over



PLAN OF MINE STARKVILLE, COLO.

6 miles from daylight to the fan. This fan was built at the foot of a shaft *a* just inside of the point where the fourth south entry turns off from the main haulage road and 4,700 feet from daylight. The air entering the mine on the old main heading, being stopped by double doors at the third and fourth south entries, was drawn down the same over to the new workings, where it met the incoming air from the new openings. The joint current proceeded along the new entry to the seventh south where it passed through a 7-foot Stine fan *b* having a measured capacity of 45,000 cubic feet a minute. Here diverting to the right, it passed up the new working places, J7 and J8, etc., turned to the left down the tenth south to the old main heading and back along it to the foot of the air-shaft *a* mentioned above. Here the air passed through a $10' \times 3\frac{1}{2}'$ fan built by the company and driven by an electric motor from the trolley current. This fan has a measured capacity of 60,000 cubic feet of air per minute at 1.5 inches water gauge and exhausted into the air-shaft 95 feet deep with a cross-section of 6 ft.$\times$8 ft.

Explosion.—At 10:07 P. M. (determined by a stopped watch found upon one of the bodies) on Saturday, October 8, 1910, the mine "blew up" and the 55 men, under the supervision of Luke Upperdine, night foreman, as well as 7 mules, were killed.

The immediate effect of the explosion was to totally destroy both fans and to partly block the mouth of the old main entry, but doing no harm either to the adjoining Engleville mine or to the new entries M1 and M2 from their intersection with the third and fourth south, toward daylight.

The condition of affairs at the focus of the explosion was found to be as follows: Six hundred feet in on the old main heading, which had caved to a height of 10 to 12 feet above the timbers, was found the electric motor with the power shut off and brakes applied. It should be noted that the motor was not jammed into or covered by the fall. Attached to the motor were 10 loaded cars, and 66 feet back were 21 more cars representing the remainder of a regular trip. The body of the motorman was found two car lengths back from his locomotive, and that of the trip rider, four car lengths forward from the rear car. In the rear section, the second car from daylight was off the track and "into the rib," and the trolley wires were down. It is reasonable to suppose that the accident happened in this manner: The motorman recognizing that his trip had split, shut off the power, applied his brakes and started back to investigate the cause of the trouble. The trip rider also started forward. Probably the jumping of the car mentioned above was the cause of the trip splitting. In any case this car knocked down several posts, and large quantities of dry dust were suddenly thrown into the air-current. The motorman had time to walk two car lengths after stopping his machine and the trip rider four car lengths before the trolley wires came sufficiently close together to "short-circuit" and thus produce the flame required to ignite the dust.

One section of the flame started toward daylight along the top of the dust-covered lagging on the entry timbers and scorched the trolley poles 250 feet from the drift mouth from 10 feet above the ground upwards. Such fallen timbers as could be observed in this short distance lay with the tops toward daylight.

Another sheet of flame started inward, but meeting the wet section just beyond the third and fourth south entries, was stopped for lack of fuel and turned to the right down the above entries over to the new workings. Following along the new entries to the seventh and eighth south, it here passed through the smaller fan and completely blew out the blades and threw the motor to a distance of 30 feet. It there turned to the right and then to the left, up J7 and J8 and their parallel entries with branches to the left along the seventh, eighth, ninth, and tenth south entries. That along the seventh and eighth south stopped for want of fuel as soon as the wet section on the old main entry described before, was reached.

The variation in intensity of the current was plainly marked. In places where dust was plentiful and the mine dry, the velocity and force of the explosion scoured the ribs clean. In other places the lack of dust or the presence of more moisture lessened the velocity and permitted the deposition of soot and coke, these conditions alternating. At the focus of the explosion, the timbers had fallen in opposite directions, but further in the heads of several sets were found pointing out while those of the following sets pointed in. Presumably this variation was due to some of the timbers being felled by the explosion and others by the recoil.

Of the bodies recovered, about 17 showed clearly the effect of burns. These were the motorman and trip rider near the old drift mouth, 11 men in the rooms driven from J7 entry near the extremity of the workings, three on the trolley parting on

the tenth south entry, and one in the workings nearby. Almost all the others were killed by afterdamp, most of them having time to seize clothing and dinner buckets and walk or run for from 100 to 1,000 feet from their working places toward daylight or toward the Engleville workings, where they probably hoped to escape through the brattices. A driver was found fully 2,000 feet from his mules, and several groups of bodies were encountered, as if the men had time to get together and talk over means of escape. In all cases, however, they were met by volumes of afterdamp and probably all were dead in 10 minutes at the most after the explosion.

As soon after the explosion as possible, word was telephoned to the local offices of the Colorado Fuel and Iron Co., at Trinidad, and by 10:45 the rescue car was on the ground.

At first, efforts were made to penetrate through the new openings but this was impossible even with oxygen helmets on account of the volume of afterdamp. It was then decided to endeavor to restore the ventilating current. To this end, a 7-foot Stine fan was placed at the mouth of the M1 entry and the M2 entry was bratticed off to prevent short-circuiting the air. This fan making 300 revolutions per minute with a 25-horse-power motor was installed by 3 A. M. Monday, and forced 50,000 cubic feet a minute into the mine. On Sunday, pending the erection of the temporary fan, entry through the old Engleville workings at the end of the Starkville main heading was effected, but the rescuers were driven back by the volume of afterdamp drawn into the workings by the fan in the other mine. In a few hours it was possible to enter as far as the third and fourth south cross-entries, which in turn were bratticed off. Work was then pushed ahead along the main entry and the necessary brattices built on seven and eight south entries to force the air up J7 and J8. When this was accomplished, only some seven brattices being necessary, the air was following its usual course throughout the mine.

The first bodies, badly decomposed, were found in the rooms off J7 and were the 11 mentioned before as being among those killed outright. These were found by the morning shift between midnight and 7 A. M. on Tuesday. After this the work of removing the dead was comparatively rapid. The last bodies, those of the motorman and helper near the old drift mouth and the last to be reached, were brought out on the following Sunday, 8 days after the explosion.

Forty-two of the 56 dead were brought out by the day rescue shift in which were B. P. Manly, mine inspector for the Colorado Fuel and Iron Co., Geo. B. Parker, in charge of the rescue station of the Carbon Coal and Coke Co., at Cokedale, who volunteered his services and remained throughout the entire week, and Thomas Tweeddale, mining engineer, who has just assumed charge of the rescue car belonging to the government service. James Wilson, superintendent of the Starkville Mine, left a sick bed to take charge of one of the rescue parties, and John T. Thomas, division superintendent, David Griffiths, superintendent at Fremont, and J. S. Thompson, also division superintendent, were on the ground and constantly in the mine. John D. Jones, State Coal Mine Inspector for Colorado, arrived Sunday afternoon and took charge on behalf of the state.

The rescue car with its four oxygen helmets and its four pulmoters were of great service. The Carbon Coal and Coke Co., Cokedale, also sent three oxygen helmets, and two others were brought by automobile from Dawson, N. Mex., 60 miles distant, sent by the Stag Cañon Fuel Co. of that place. F. C. Miller, chemist for the Colorado Fuel and Iron Co. continued throughout the rescue work to take samples of the air for analysis from the top of the air-shaft and through the brattices from the adjoining Engleville Mine. This was done to determine if a mine fire had broken out or not. Mr. Miller found as high as 1.64 per cent. of CO_2 and .5 per cent. of CO. As an item of interest, this mine is but a short distance from Sopris, Primero, and Tercio, all of which have been visited by disastrous explosions in recent years.

ELECTRIC AIR COMPRESSOR

Written for Mines and Minerals, by James A. Seager

In Great Britain the rival claims of electricity and compressed air have been for some time a matter of considerable interest and it is therefore interesting to note a recent equipment which has, in the case of the Newton pit of the Clifton & Kearsley Coal Co., yielded a satisfactory solution of the efficient application of electrical power to those appliances which are most economically worked by compressed air in collieries. In Fig. 1 is shown a large three-phase induction motor driving a Tilghman air compressor through the flexible coupling, the two machines being mounted on a combination base plate. The compressor is of the three-crank type and is driven by a motor of 350 horsepower running from a 2,000-volt, three-phase, 50-period electric circuit. This set is capable of continuously compressing 2,000 cubic feet of air per minute to an outlet pressure of 75 pounds per square inch.

The special feature of the motor is the brush shifting and short-circuiting device, the brush gear being of the box type. This gear is actuated by the small handle on the end of the slip ring cover. On pressing the handle toward the machine the slip rings are short circuited by a plate with three switch

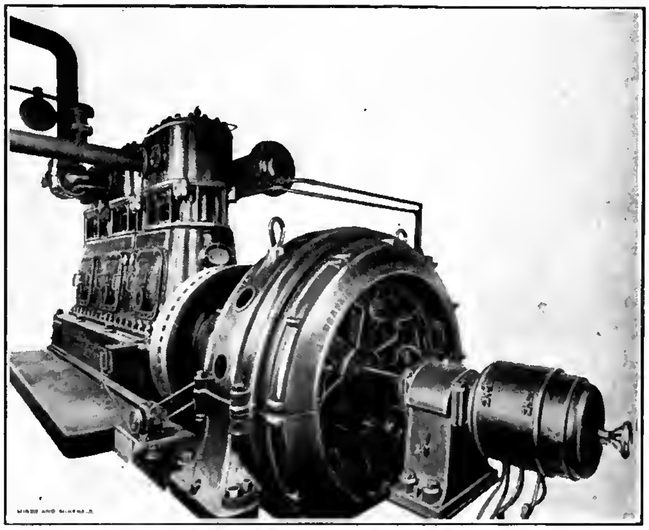


FIG. 1. ELECTRIC MOTOR DRIVING AIR COMPRESSOR

jaws making contact with knife blades connected to each slip ring. On further pressing the handle the brushes are moved on to stationary supports placed between the slip rings. This device does away with the comparatively heavy heat loss and the rotor leads to the starter, increases the efficiency of the machine, prevents wear of brushes, and keeps the slip rings cool, as they have no current to carry.

The starting panel has a three-pole oil-break switch with no volt and overload release trips fitted in two phases, and is in a sheet-iron case with hinged doors interlocked with the oil switch so that they can be opened only when this switch is off position. A liquid-type starter is used provided with a hand wheel and screw gear for slow starting, and controller fingers make contact with copper rods in the short-circuit position. The safety of the appliance, even if used by untechnical assistants, is increased by interlocking the cover of the starter so that it cannot be raised unless the blades are out of the liquid. As an example of recent British work in air-compression plant, this equipment is not without interest to American engineers.

The introduction of the Pneumelectric coal cutter and the Ingersoll-Rand electric-air rock drills into mines in the United States is a step forward which again shows the advisability of combining the two powers air and electricity wherever the use of one is unsatisfactory. Electricity alone has not proven satisfactory for rock drills.

CHEMISTRY OF COAL-DUST EXPLOSIONS

Written for Mines and Minerals

Resnlts of Studies of Dust Explosions at the Experiment Station at Liévin, France.

The chemical aspect of coal-dust explosions has been carefully studied at the Experimental Station at Liévin, France, and the discussion of the results obtained in this direction forms one of the most important parts of a report on the production of dust explosions recently issued by the Director of the Station. The experiments were conducted in the main test gallery, then 65 meters long (one meter being equivalent to .914 yard), and in order to collect the gaseous products of combustion a glass flask, exhausted of air, and connected by means of a sealed glass tube to the interior of the gallery, was caused to fill itself immediately the explosion reached that point. This was done by the flame firing a detonator attached to the tube which was shattered, thus allowing the gases to rush into the flask. To ascertain the composition of the gases entering in this way, the analyses had to be corrected for the amount of air previously existing in the flask owing to incomplete exhaustion. In the accompanying Table 1 the actual analyses are set forth in columns 3, 4, and 5, while the corrected figures are given in columns 6, 7, and 8. The figures in the first column refer to the number of the experiment, samples of the gases not being taken in every one of the tests, but only now and again.

TABLE 1

Number	Dust Charge in Grams Per Cubic Meter of Gallery	Analyses of the Gases Collected in the Flask			Corrected Composition of the Gaseous Products of Combustion			$\frac{CO}{CO_2+CO}$ (Volumetric Proportion)	Relation Between the Weight of Hydrogen Burnt and the Weight of Carbon Burnt
(1)	(2)	CO_2 (3)	O (4)	CO (5)	CO_2 (6)	O (7)	CO (8)	(9)	(10)
		Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	Per Cent.	
43	112	7.50	11.00	.50	9.00	8.40	.60	6.0	.1043
53	225	10.25	5.00	3.00	12.10	1.20	3.60	23.0	.1196
79	337	10.25	5.75	1.50	10.50	4.60	1.50	13.0	.1391
56	337	9.75	5.75	2.00	11.40	2.10	2.30	17.0	.1463
39	450	0.75	4.25	4.50	11.60	.30	5.40	32.0	.1210
139	450	12.50	3.25	1.50	13.05	1.45	1.60	10.7	.1242
36	450	11.00	3.00	4.00	9.95	1.90	4.10	28.5	.1610
58	450	10.25	5.00	2.25	12.00	1.20	2.60	18.0	.1410
77	450	12.00	2.50	1.50	12.00	1.25	1.50	11.1	.1677
57	450	9.25	5.75	2.00	11.10	1.50	2.40	17.7	.1715
62	450	7.75	7.00	3.25	11.00		4.60	29.5	.1565
63	450	7.50	7.00	3.25	9.90	1.65	4.30	30.2	.1700
108	450	10.00	2.75	3.75	10.60	.48	4.00	27.2	.1772
35	450	8.00	6.50	1.50	9.20	2.90	1.70	15.7	.2394
150	450	5.50	2.00	6.50	5.30	.95	6.30	54.1	.3275
151	450	4.25	2.50	6.50	4.20	1.00	6.40	60.4	.3972
75	610	10.50	5.00	1.25	10.60	3.80	1.30	10.6	.1623
74	610	10.25	2.75	4.50	10.50	1.50	4.60	30.5	.1454
59	900	7.00	6.75	4.25	8.40	3.00	5.10	37.7	.1682

An examination of column 7 shows how little free oxygen remained after the passage of the flame; the combustion must therefore have been almost complete. In the test No. 43 the residual oxygen attained 8.4 per cent., but it is seen that this experiment was conducted with the lowest dust charge; namely, 112 grams per cubic meter, a gram being equivalent to .0353 ounce. Theoretically, this amount of dust is sufficient to absorb the whole of the free oxygen of the air; therefore, the analysis shows that, as 40 per cent. of the air remained unchanged, a portion only of the 112 grams of dust took part in the explosion.

In column 9 is expressed the relation between the volume of carbon monoxide and the total volume of the two oxides of carbon formed in the combustion—that is $\frac{CO}{CO_2+CO}$. Leaving the results of Nos. 150 and 151 out of account for the present, it is seen that this ratio, though very variable, is low, and that it appears rather larger for the heavier dust charges than for the lighter ones. The fact revealed here could hardly have been anticipated; it is that although an excess of dust may be present at a high temperature, the amount of carbon dioxide formed in the flame greatly predominates over the amount of carbon monoxide. It must, however, be noted that the gases were taken from immediately behind the breast of the flame, and before the red-hot carbon had a chance to react on the carbon dioxide and reduce it to the monoxide. That the latter reduction actually takes place, and that the reaction takes a short but appreciable time to complete, was demonstrated by tests Nos. 150 and 151, wherein the gaseous products were obtained from distances, respectively, 10 and 5 meters behind the breast of the flame. A glance at the analyses of the gases taken in these two tests will show that there was an excess of the monoxide over the dioxide of carbon at these distances behind the flame. It is therefore apparent that chemical activity continued behind the flame, resulting in the rapid increase in the percentage of carbon monoxide.

How very dangerous is the afterdamp of a dust explosion may be gathered from these last analyses. Not only had the oxygen almost entirely disappeared, but carbon monoxide was produced in such high proportions that if the products had been diluted with a large quantity of fresh air they would still have been dangerous to breathe even for a short time.

In the last column of Table 1 is stated the ratio by weight, w, of the hydrogen to the carbon consumed in the combustion. The figures in this column are noteworthy in that they indicate the importance of the part played in the ignition by the volatile constituents of the dust. Before the bearing of these relations can be discussed it is necessary to explain the manner in which they were calculated.

The quantity of carbon burnt is deducible straightway from the amounts of carbon dioxide and monoxide produced. The hydrogen, however, cannot be computed quite so easily, since it forms watery vapor in burning, which condenses, and hence does not make an appearance in the results of the analyses. The weight of hydrogen has therefore to be obtained indirectly by calculating the amount of oxygen which has disappeared as water. Now the analyses render account of all the oxygen, free and combined, except that forming water; while the nitrogen formerly existing mixed with oxygen, as the air of the gallery, is obtainable by difference. Thus, by using the well-known relation between the volumes of nitrogen and oxygen in normal air, we are able to compute the total original amount of the latter gas, and then, by difference, the amount which, after the combustion, exists in combination with hydrogen as water.

The following gives further particulars of the calculation:

Let 100 volumes of the gas in the flask contain a volumes of CO_2, b volumes of oxygen, c volumes of CO, and d volumes of hydrogen and hydrocarbons, the remainder being nitrogen. Among these various gases will be mixed r volumes of air, being the quantity initially present in the flask by reason of imperfect exhaustion. The latter quantity, r, is easily found from the initial pressure in the flask. Making the necessary correction for r, the total volume of oxygen entering the flask, both in the free and combined states, is:

$$O = a + b + \frac{c}{2} - .21r;$$

and in the same way the quantity of nitrogen entering is

$$N = 100 - (a + b + c + d) - .79r.$$

Of the nitrogen so calculated a small quantity, n, will have been derived from the coal dust; the remainder $(N-n)$ existed originally in the air of the gallery, and accompanying it there must have been $\frac{21}{79}(N-n)$ volumes of oxygen. If k volumes of oxygen were also derived from the coal dust, then, by difference,

$$\frac{21}{79}(N-n) + k - O$$

measures the amount of oxygen now in union with hydrogen in the form of water. Therefore, by volume,

$$\frac{\text{Watery vapor}}{CO_2+CO}=2\left(\frac{\frac{21}{79}(N-n)+k-O}{a+c}\right)$$

and by weight,

$$\frac{\text{Hydrogen consumed}}{\text{Carbon consumed}}=w=\frac{2}{12}.2\left(\frac{\frac{21}{79}(N-n)+k-O}{a+c}\right)$$

Substituting for N and O their values as obtained above, we may write:

$$w=\frac{8.75-.421(a+b)-.255c}{a+c}-\frac{d}{12(a+c)}+\frac{4k-n}{a+c}$$

This expression includes a principal term—the first one, and two corrections—the last two terms, of which the latter are of little importance. It has been found that d has a value never exceeding unity, and that the smallest value of $(a+c)$ is 12; therefore the term $\frac{d}{12(a+c)}$ is at the most equal to $\frac{1}{144}$; i. e., only 5 per cent. of the mean value of w. To evaluate the term $\frac{4k-n}{a+c}$, it is necessary to consider the weight of dust producing the quantities k and n. Let us suppose that 450 grams per cubic meter of air had been put into effective suspension. This amount will be equivalent to 45 grams per 100 liters of air, and will very nearly be equivalent to 45 grams per 100 liters of the gas in the flask. Now 45 grams of the dust used in these experiments contained 8.5 per cent. by weight of oxygen and nitrogen, that is to say, 3.8 grams of these gases; and these when liberated would occupy 3 liters. Since the residual particles after an explosion still contain a large part of the volatile constituents of the dust, only a portion of the 3 liters of gas will actually be set free. Inasmuch as those quantities of oxygen and nitrogen existing in the proportion in which those gases are found in normal air will be eliminated in the difference $(4k-n)$, the latter term can only represent a fraction of a liter. It is therefore apparent that the last term of the above expression is of even less importance than the middle one. Hence, in calculating the figures in column 10 of Table 1 these last two terms were neglected.

The general high value of w is striking and important. In the dust used for the experiments there were only 4 parts by weight of hydrogen to 75 of carbon. If the carbon had completely burned, w would have been about .05, while as a matter of fact, it always attained at least double that figure.

If the volatile matter were taken to consist entirely of hydrogen—certainly an incorrect assumption—the above results would show that the weight of the volatiles taking part in the combustion would be derived from at least twice the weight of the dust wholly burned; the proportion will, as a fact, exceed this, for besides hydrogen, the volatiles contain carbon in the form of hydrocarbons, and that element goes to swell the denominator of the ratio w.

The mean value of w appears to be increased when heavier dust charges were used, and this might be explained from the fact that the more numerous the particles of dust in a cloud, the greater the total surface from which volatiles may be distilled.

Now it has already been stated that the gases entering the flask in all the other experiments contained only an insignificant amount of unburnt volatiles; however, on estimating as marsh gas the unburnt volatiles obtained in tests Nos. 150 and 151, they were found to be present in the proportions of 3.8 per cent. and 5.2 per cent. by volume, respectively. It therefore follows that for these tests the last two terms of the formula for w derived above are no longer negligible, and must be taken into account. After the necessary corrections had been applied, it was found that the values of w for the two experiments in question were, respectively, .301 and .352. Although these figures are less than those given in column 10, they are nevertheless considerably larger than their equivalents in the other tests where the gases were taken from close behind the flame. The presence of unconsumed volatile matter, and the high value of w obtaining when the gases were taken some distance behind the front of the flame, show that the distillation of the dust continued in that region after the flame had passed. Since there is no more, or nearly no more, free oxygen available to unite with the volatiles so produced, there must remain a surplus of the latter. The increment in w shows, however, that some of the hydrogen of the volatiles liberated after the passage of the flame had united with oxygen, while the proportion of carbon monoxide has also increased; it is therefore probable that, in the region behind the flame, reactions such as the following take place:

$$H_2+CO_2=H_2O+CO.$$

Other observations were made on the phenomena of combustion. The first of these was in reference to the amount of smoke produced by a dust explosion. By raising a shutter, a connection was made after every experiment between the gallery and a fan drift, and the smoke then blown out of the open end of the gallery by the fan, when its color and other characteristics could be noted. When the dust charge was at least 450 grams per cubic meter of air, the smoke was always thick and black. With lesser charges it was a clear grey, and sometimes almost white.

Dense black smoke consists of an excess of fine dust particles, together with watery vapor which has cooled and condensed, and soot resulting from the combustion of hydrocarbons in an atmosphere almost depleted of oxygen. Grey or white smoke indicates a combustion of the dust which is complete or nearly so, only ash remaining, together with watery vapor in process of condensation. In the last case there existed a sufficiency of oxygen to burn the hydrocarbons without the formation of soot.

In a number of experiments the solid residues remaining in the gallery were systematically collected. They were obtained from the floor and from a dozen transverse shelves each 20 centimeters wide, placed every 2 meters between points in the gallery, respectively, 18 and 40 meters from the origin of the explosion, it being in this portion of the gallery that the bulk of the solid products were deposited. As a rule, little of the residues was found near the source of the explosion and often none at all in the last few meters of the gallery. This latter fact might be explained through the reentry of air occurring immediately after an explosion, which would sweep back into the interior any solid matter lying near the mouth of the gallery.

Table 2 gives the weight and composition of the solid residues collected, the weights being stated per cubic meter of the zone from which the products were taken. From what has just been said, it will be apparent that had the products been collected from the full length, and compared with the full volume of the gallery, the results would have been lower than those given below. On the other hand, they would have been higher

TABLE 2

Number of Experiment	Weight of Dust Per Cubic Meter of Gallery	Weight of Solid Residues Per Cubic Meter of Gallery	Analyses of Solid Residues			Percentage of Volatile Matter (Moisture Deducted) in the Solid Products After the Ash Has Been Eliminated	
			Volatiles Including Moisture Per Cent.	Moisture Per Cent.	Ash Per Cent.	In Detail	Mean
	Grams	Grams					
43	112	40	21.15	1.69	22.52	25.0	25.00
53	225	123	21.57	2.42	14.52	22.4	22.40
56	337	210	25.08	1.99	12.45	26.3	26.25
79	337	220	25.09	2.19	12.94	26.2	26.25
36	450	240	25.70	2.20	13.95	27.2	25.30
39	450	227	22.86	2.20	14.54	24.1	25.30
58	450	335	23.37	2.21	13.12	24.6	25.30
75	610	530	26.49	2.19	12.70	27.8	27.80
54	900	670	27.09	1.87	10.52	28.2	27.50
59	900	680	25.98	2.02	10.39	26.8	27.50
Composition of the coal dust used in the experiments..........			28.00	1.00	9.00	29.6	29.6

had the matter from the walls also been included, and also if it had been possible to have added those solid particles ejected

Table 3

	Volatile Matter Per Cent.	Ash Per Cent.
Coke in grains, or agglomerated	15.3	14.04
	14.4	14.23
	13.4	11.08
	12.3	19.20
	11.4	27.20
Coke in thin flakes from the surface of dust	24.6	11.71
	24.0	11.35
Soot mixed with dust	13.8	19.20
Residual coal dust	28.1	10.75
	26.3	14.05
	26.2	10.11
	22.4	13.10
	18.9	14.70

from the gallery by the force of the explosion and not drawn in again by the back suction of air.

As would be expected, the weight of residual solid matter increases with the dust charged into the gallery; it is almost proportional to the excess of the charge over about 100 grams per cubic meter, which corresponds to the complete combustion of the carbon of the coal dust by the oxygen of the air.

The analyses show that the products were poorer in volatiles and richer in ash than the original dust from which they were derived. A reference to the last column of the table also shows that, even after the ash constituent has been eliminated from the products and the original dust, the former is still the poorer in volatiles, thus proving that the reduction in volatiles revealed by the analyses does not result merely from the increase in ash, but that there must have been a partial distillation of hydrogen and hydrocarbons from that part of the dust which was not completely burned. This result is in concordance with the conclusions which have already been drawn from the gas analyses. The solid products were appreciably more humid than the original dust, owing to the condensation of some of the large quantity of watery vapor produced in the explosion. Water was also found condensed on cold portions of the walls of the gallery.

A further examination of Table 2 shows that there was generally a more pronounced reduction in the percentage of volatile matter when the dust charge was small. In other words, although the amount of volatile matter per cubic meter is greater with the heavier charges, it does not increase proportionately with the charge, but at a lesser rate. This can be explained partly from the laws of variation of temperature during combustion, and partly from the fact that, with the heavier charges, the dust is not perfectly lifted; thus there must be a portion of the dust in contact with the floor, which, although included as "solid residue," has not been thoroughly subjected to the effects of a high temperature. Hence, in reality, Table 2 registers the analyses of a mixture of the actual solid products of combustion of the dust lifted, with dust lying immediately against the floor, the composition of which would hardly be altered.

Examination showed that the solid residues usually consisted of three parts; namely, dust whose physical aspect remained unchanged, coke either in a finely divided condition or as grains and soot. These three elements, which were difficult to separate properly, were distributed in unequal proportions along the length of the gallery. Circumstances in that part of the gallery not allowing of the agglomeration of large pieces, the coke forming in grains the size of a pea or of a pin's head, within a few meters of the origin of the explosion, was carried by the rush of air either toward the middle of the gallery or sometimes—especially for the heavier dust charges—nearer the open end. The soot, very fine and greasy to the touch, was deposited on top of the coal dust and coke. It required some time to settle, and was not seen at all when the fan was started straightway after an explosion.

In certain parts of the gallery a firm thin skin of coked matter was occasionally noticed on the surface of the unconsumed dust, but the conditions under which the skin was produced were not clearly determined.

Table 3 gives the analyses of the various solid products of combustion, separated fairly well one from another:

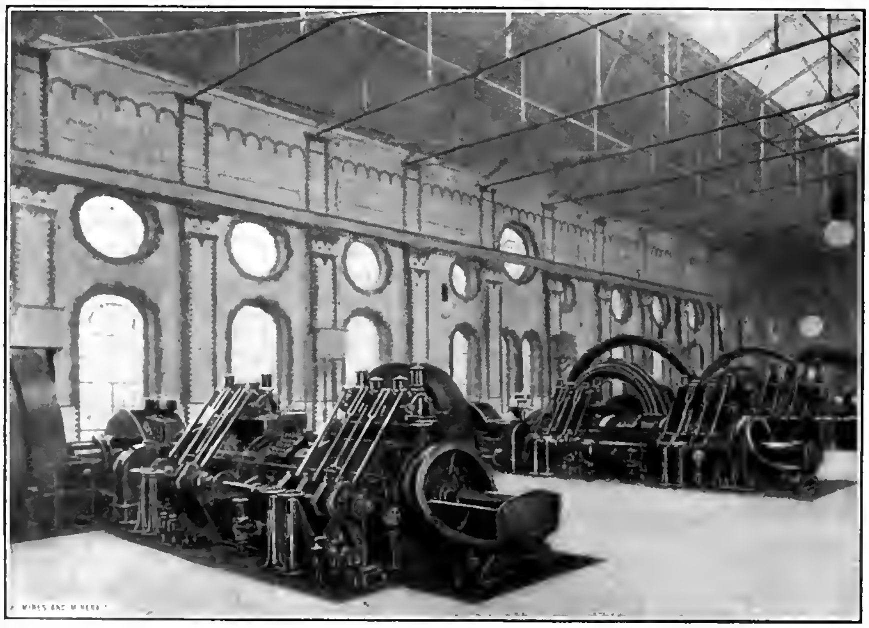


Fig. 1. German Gas Engines Using Coke-Oven Gas

COKE-OVEN GAS

The interior of the power house at Alsdorf, Germany, of the Eschweiler Bergwerksverein is shown in Fig. 1. This gas-engine power station, near Aix-La-Chapelle, is said to be the largest plant of gas engines driven by coke-oven gas in the world and consists of nine tandem double-acting units of the Nurnberg type having a total capacity of 14,600 brake horsepower.

All of these gas engines drive three-phase alternators which operate day and night in parallel, the electric current being transmitted throughout the mines for driving all the machinery, A part of the power is transformed to a pressure of 35,000 volts, the current being conducted to a considerable distance for operating various mining machinery.

Gas from bituminous coal carries from 1 to 5 per cent. of tar and other products that would gum and choke the valves of gas engines if admitted directly to the cylinders. It is customary therefore to extract the tar oils, and ammonia sulphate by previously passing the gas through suitable apparatus before it is admitted to the gas-engine cylinders. The operation decreases the heat units in the gas by lessening the hydrocarbons, nevertheless there are sufficient remaining. In some places it is customary to make producer gas from coke breeze for gas engines. This material which contains a mere trace of solid hydrocarbon and usually none at all, produces an inferior quality of gas so far as heat units are concerned, but still there are sufficient for gas-engine power.

HANDLING MINE SUPPLIES

By H. H. Fitch *

It is our understanding of the object of our institute, that it is intended, among other things, to encourage education in practical and scientific mining, and to promote study and research into mining problems of every nature.

Advantages of and Methods for Systematic Handling. Disbursement and Accounting

The great mass of subjects selected by, or assigned to, your speakers, bear on the all-important question of the actual mining of the coal, and its treatment and handling after it has left the hands of the miner.

To get just the proper procedure injected into our mining practice, it is extremely easy to lose sight of sundry matters of material importance, because they wear the commonplace aspect of ordinary business transactions, and are not garbed in the attractive phraseology of a scientific problem.

It is for this reason we decided to give you the benefit of some experience along lines quite apt to be neglected by the rank and file of coal-mining men.

The tendency of the present day is toward greater system in all lines of business procedure, and no business can long continue in successful operation without it. Where can we find a better field for system, than in the vital one bearing on supplies and machinery repairs, for use at operations equipped with costly electric plants, air plants, and steam plants, and in fact a machine of some description for nearly every operation formerly performed by man or mule, when we consider that any one of these machines is liable to require at a moment's notice repairs ranging from a set screw up to some part weighing a ton or two, which, while not particularly high in intrinsic value, is capable of tieing up your whole operation if not at hand when needed, and so kept as to be available without delay?

We believe that the well-equipped and properly managed supply department is essential to the coal plant in order to minister to the demands of the sick and disabled machinery that is being pushed to the limit of its endurance to keep the output up, and if not properly fed on repair rations, drawn from the bounty of the supply house, will soon lie down to die by the wayside of commercial hustle, utterly unable longer to perform the duties laid upon it.

At first glance, the uninitiated are apt to pass over the supply account as one of minor importance, to be handled by any old dub of a clerk who possesses sufficient ability to write legibly and add a row of figures.

To those possessing that idea, let us quote from our supply department, which looks after the needs of five medium-sized plants.

Our annual inventory shows that we carry a stock containing approximately 20,000 items of material. The value of this stock averages about $25,000, and from it we make an annual disbursement of $100,000 worth of supplies, of all kinds. This means that the stock is moving all the time, and if you will stop to think of the details involved in properly storing, disbursing, and accounting for a stock of this size, and the care needed to keep it up to such a standard of efficiency as to enable it to be of actual service to your operation, you will realize that it is a man's job, and deserving of serious consideration.

The account is not an abnormal one, but merely such a one as must be carried by any operation embracing two or more plants, if the management makes any pretense of keeping its equipment up to successful working efficiency. As a rule, most developments are away from the centers of supply, and in cases it takes days and even weeks to procure repairs or supplies of the simplest nature. This puts us at the mercy of Mr. Breakdown, whenever he visits us, or of some local merchant who is enterprising enough to keep what we want and let us pay him 50 per cent. on his investment, when we buy his material. The only successful method of getting away from these alternatives is to maintain a supply depot of your own, managed by a man big enough for the job, and with enough experience in mining work to give him an intelligent conception of what is needed, and sufficient interest in the work in hand to take pride in keeping his department up to the mark set by the needs of his operations.

For the sake of comparison, glance for a moment at the fellow who does not consider it necessary to use any system in caring for his repair supplies. In his scramble to get his production up, he has brushed this question aside as one not worthy of attention.

On the day we make his acquaintance, Mr. No-System, the man in charge of our plant, is a little bit ruffled, for just when everything was going along finely, one of his haulage locomotive armatures comes to grief.

At first glance this does not appear very serious, for we have a set of coils somewhere (?) about the place, and we have the man who can wind the armature.

We will set him to work at once and with any kind of luck we will be ready to haul coal again tomorrow. We simply must have that motor, says our friend, for if we do not, it will mean a loss of a hundred cars in our production, and since we are already short of coal we cannot charge our ovens.

In the meantime the search for the needed coils goes hurriedly on, but of no avail. They cannot be located high nor low. It finally dawns on our electrician that he must have used that set of coils on the last armature, and somebody, we can't tell who, has forgotten to send us a new set.

We are then treated to an example of desperate hustle on the part of our friends who do not believe in system, in their effort to locate a set of armature coils to fit that motor. The long-distance telephone is brought into use in a vain endeavor to borrow a set from some one within a radius of 100 miles or so, but no one is found who has a set that can be spared. As a course of last resort the manufacturer is called upon, and since they are 2 or 3 days away from our plant by the fastest express service, the only thing left for us to do is to wait.

The result of this is plenty of trouble and annoyance back at the plant. The locomotive is snugly lying in the shop, keeping perfectly cool, and taking a welcome rest. The men down on the heading are howling for cars that they cannot get. The coke foreman is yelling for coal to keep from blowing out his ovens. The sales department is loudly calling for the coke that cannot be made. In fact, if we were inclined to be poetical, we could make you a nice little parody on the old nursery rhyme, which tells about the horse shoe nail that first caused the loss of a shoe, then a horse, a man, a battle, and finally a kingdom; but in this case we would end up by telling about the contract that was lost all on account of an armature coil.

If persisted in, how many occurrences of this kind would it take to put you down and out.

But, says our friend, No-system, this is not going to happen again. We are going to correct this condition at once. A few dollars worth of copper wire is not going to shut me down another time. We will order enough stuff to run this plant in safety in the future. Forthwith, the orders begin to go in from that mine, and they do not stop until everything has been bought that can be anticipated as liable to be needed for months and weeks to come.

Not satisfied with getting plenty of stuff for himself, he proceeds to be neighborly, and citing his own case as an example of the danger of not being prepared, he has the fellows at the company's other mines joining the scramble to get ready for trouble. This causes them to duplicate most of the material bought for the other plant. With an operation embracing three of four plants, everything being charged direct to cost of

* Read before the Coal Mining Institute of America, June, 1910.

production, what is going to happen to the cost when the bills are paid.

Of course this heavy demand for material is noticed at the general office, but is passed on the assurance of the mine heads that the stuff is needed.

The operator is usually a pretty keen sort of fellow, and when he embarked in the coal-mining business he did so with his eyes open, and after careful estimates of the cost of operation. He knows what he expects to realize from his investment, after all reasonable allowance has been made for waste, bad management, etc.

When the bills for this lot of material reach him, he is shocked to see his prospective profits dissipated, and when the yearly statement shows a deficit against the operating department, he asks pointed but rather unpleasant questions.

While pondering over the matter let us return to our plants and see where his profits really go. Our mine people are still unconvinced of the fact that loose material absolutely refuses to stay around a coal plant, so when the newly purchased lot arrives, it is hauled up to the mine and turned over to the care of Mr. Any Old Chap. He scatters it around nicely, so it will be handy when needed. This proves to be an excellent method of getting rid of it quickly, for we see all of the boys dropping around and helping themselves to whatever they need, and whenever they need it. The result is that it is only a short time until we are in our same old state of unpreparedness, and Mr. Break Down catches us in short order and repeats his former treatment without the least compunction.

In the meantime our friend the operator has been assured that the trouble would not be allowed to occur again, and he puts the first experience away in his memory for future reference. When the second fall dawn comes to his notice, he is about convinced that his methods need fixing about as badly as his machinery.

A general conference takes place, and results in much talk, and schemes for curing our ills are many. All of the talk and all of the schemes, however, finally narrow down to the one which they think is the best. It is this:

Instead of letting our material lie around loose, we will establish local supply depots at each mine. We will charge all invoices received to a general material account, instead of direct to the operation. The mine clerk never has much (?) to do except to keep the time, issue checks, make his pay roll, rent the houses and sundry other little duties, so we will in addition make him supply clerk, just to give him something to do to kill time.

All material will be issued only on written order from the mine head, and charged out to the cost of coal as it is used. This will insure a uniform monthly cost of production, and we will not lose sight of any material.

Looks like a plausible scheme, does it not? Let us follow it out in practice and see how it works.

We must remember that our man who is to carry the order book, is none other than Mr. No-System, who got us into trouble before. He looks on anything of this nature as mere red tape. True he looks at the matter according to his best light, but very likely he has no conception of anything pertaining to accounting. He knows he uses the stuff at the company's mine for the purpose of getting out the company's production. Why any one should care to know just where it was used or for what particular item in the production is not at all clear to him. However, for the first week he manages to fill out orders for what he needs from the supply room.

The second week it is different; he leaves that pesky order book home and when he finds his machinist waiting for him to get an order for some stuff needed in a hurry, he cannot wait to send for that book, so he telephones the clerk to let the stuff go out and promises to fill out an order in the evening

That breaks the ice, and from that time on it is as easy to get material out of that supply room as it was when it was lying around loose. The clerk cannot be censured, for he is simply carrying out instructions from his superior officer, and any other course would be classed as insubordination.

The orders, which are all the clerk gets from which to make up his list of material used, are neglected more and more as time goes on. After a time the machinist, the electrician, and the pump man have, when the clerk was busy, been allowed to go in and help themselves, and we find all kinds of material disappearing from the supply room without record being made of it, and as far as any one in authority knows, it is still on hand.

This method of doing business results in a nice monthly cost while it is going on, but wait. Down in the auditor's office, all of the invoices have been religiously charged up to material account, as agreed in the beginning. After a time the inventory day rolls around. On that day we must square accounts for the year and start new. The auditor comes along with his assistants, and they proceed to give our supply rooms a careful check. They list and price everything they find, and cannot be persuaded to list a single article which we are unable to show them. They go back to headquarters and make a comparison with what we have and what their books say we ought to have. The chances are that every supply room will show a shortage of from $1,000 to $2,000 worth of material.

What are we going to do with this shortage? It must be taken care of, or else our yearly statement will look like a joke. There is only one fellow who is ever able to take care of a thing of this nature. He is our old friend Mr. Coal Cost, and it has a bad effect on him. We see this particular cost bulging far beyond what we had falsely been led to believe would be his size, from the estimate, based on the monthly cost, and our reform in handling supplies has placed us in a worse position than ever with our management.

Having thus briefly glanced at what actually happens to the man who does not believe in system, and to the fellow who tried to remedy his troubles and "got in wrong," let us search for a plan that will cure the troubles our experience has taught us are to be overcome.

Some of the points in our first trial are all right and we will use them where they will fit.

We must first do away with our numerous scattered local houses. To do this we select a location for a central supply house, at some point easiest of access from our principal present development, or our most important future one.

We must plan our building in such a manner as to permit easy handling, and systematic storing of the goods to be taken care of, for we are going to do a pretty extensive business, of a much varied character, in our new home.

The house itself must be fireproof and of sufficient size to take care of our growth for a number of years to come. It should be located on the railroad main line, as near as possible to the depot, to avoid hauling when shipping to other mines.

We must have a raised platform of ample room, where the local freight may load and unload supplies, and where heavy articles, such as locomotive ends, trucks, armatures, etc., may be handled in and out of the cars without the use of a crane.

If room permits, a short siding should be provided at the house where carloads may be placed for unloading.

Our unloading platforms must be covered with a shed roof to protect any perishable matter being handled in wet weather.

The working floor of our house must be on a level with the raised platform, so that material may be trucked to and from the trains without the use of an elevator or an incline.

The inside of our building must have the walls solidly built, with bins or compartments for storing goods, and must have a balcony extending completely around the room, about midway from the floor to the top, so that the material stored in the upper bins may be used without using ladders.

The bins must all be provided with label holders, in which can be placed cards, naming the article in each bin. This is necessary, for we are going to have some queer looking stuff to

take care of, and we must have it in such shape as to be found by the uninitiated when occasion demands.

The center of the room will be used for the storing of extra motors, pulleys, armatures, and such other material of a bulky nature as cannot be placed in the bins.

In one end of the room we need a rack for the storing of the different sizes of pipe and iron which we need to carry.

With our building in readiness, we need to select a man for the position of supply agent, whose duty it shall be to give his exclusive attention to the care of the supply business, for our operation.

We must have a man of sufficient business ability and experience to manage the details of a business of these proportions. He must be a man able to keep his book records in such a manner as to enable him to tell at all times just how his account is running.

And above all he must have sufficient acquaintance with mining machinery in general to give him a working knowledge of the different component parts of the various machines in use.

After we get the proper man, he is sent out to the local houses, and at each of them a careful inventory is made of what is on hand, and it is shipped to the central house.

When it is all assembled at that point it is carefully sorted and classified and placed in the bins. In this operation it is important to begin the observance of system in its strictest sense. A section of the bins must be allotted to the storing of pipe fittings, another to trolley-line material, still another to electric-light supplies, and others to the parts of each separate machine in use, etc.

The total of the inventory from the local houses is used as a basis from which to start the new stock account and retire the former material account from the auditor's records. Whatever shortage appears in the old account is charged to the operation and cleaned up.

The items of the inventory are then entered in a permanent stock record. This form carries three general divisions, the first showing a complete record of all goods received, with its final cost at the supply house divided into a unit price at which the material will be sold to the mines, the second showing a record of disbursement, and the third a periodical balance of the account. On account of the continual expansion of the stock on hand by the addition of new items to be carried, this record, as well as all others used in our supply work, is bound in loose-leaf form.

The matter of a proper system of indexing this record is a serious study, for the reason that it embraces so large a number of varied items, many of which bear several names, or could be described under so many different letters of the alphabet.

We take as the basis of our index the regular alphabet divisions, and then subdivide our letters as needed to get our material properly classified.

For instance, take the items of bolts. We place their record under the letter B, then classify them as carriage, machine, track, and miscellaneous, giving each class a separate place in the record. Then each class is again divided according to their size, ½-inch bolts of various lengths, in one group, ¾ inch in another, and so on down the list.

In a similar manner motors of the various types would be grouped under the letter M, subdivided according to the class of motor, all motors of the same type in which the parts are interchangeable being placed in one group, and all repair parts for that type motor are listed under this division, regardless of their alphabetical name.

Locomotives would be found under the letter L, with their parts subdivided in accordance with the manufacturer's catalog heading, such as resistance parts, brush-holder parts, controller parts, etc.

This method enables you to turn without unnecessary leafing, to the item wanted, no matter under what name it is called for.

The rule as it now stands is that all requisitions must bear the approval of the general superintendent, or other officer in charge, before they are good for credit at the supply department. With this method the man in authority, who must account to the management for the cost of production, has a chance to be in touch with every item to be used in making the production. If any item on the order appears unnecessary, or unreasonable, he is in a position to veto it or request additional information as to its necessity before passing it, and the supply agent being no longer under the control of the local heads, is in a better position to refuse to issue material except on the proper order.

In order that your plants may not be tied up for want of repair parts in the case of a break down, the supply agent is made the judge as to the immediate need of any material requested, and authorized to issue it in cases of extremity on a telephone or verbal request after satisfying himself that the proper requisition has been made and sent to headquarters. A record is kept of these emergency requests, and same are reported at the end of each month, and the mine foreman who appears to be abusing the privilege is called upon for an explanation, and unless a satisfactory one is given he is liable to have an unpleasant interview with the man who makes the orders down at headquarters.

In order that the prepaid freight from the supply house to the mines may be kept as low as possible, each mine endeavors to cover each week's requirements on one order. This, however, is optional with the mine heads, as they may make a requisition at any time when they find it necessary. On receipt of the order at the supply house, it is immediately filled from stock, if the items are carried, and if not the goods are ordered shipped direct to the mine from the manufacturer on requisition by the supply agent.

After filing the requisition, the items are billed against the mine receiving them, the same as if purchased from a regular mercantile concern.

The form of invoice is an adoption of the usual form of invoice, amended to suit our needs by the use of special ruling.

Each invoice is entered in the middle division of the stock record, as it is made, showing deductions from stock on hand.

A charge of 5 per cent. of the value of all goods passing through the supply house is made, and this is distributed throughout the items on the invoice, making a proportional distribution of the cost of maintaining the supply department, based on the amount of stuff used at each mine. Only sufficient commissions are charged in any one month to cover the actual running expenses of the department, and when that item has been cared for, the charging of future commissions is discontinued for the month.

At the end of each month the supply agent makes an itemized report showing all material disbursed to each mine. The totals of this report are distributed to the several headings of the monthly coal cost, and this distribution shows at a glance the actual amount of material used in each phase of the month's production, and is further run out to a cost per ton basis by the auditor.

By the use of the above system of reports and distributions, under the control of one man, a consistent basis of comparison is possible between the different operations, along certain lines of cost. Under the old plan, each mine clerk had a different idea as to where certain material should be charged, and a consistent comparison of material cost was not possible.

This covers in a brief manner all that is of interest pertaining to the disbursing of supplies passing through the house. A separate account is kept of all bulky material, such as steel rails, pit props, ties, sand, brick, etc., which is delivered direct to the mines in car-load lots. On receipt of a car load of material it is tallied by the mine people, and a receipt for it is forwarded to the supply agent. By him it is attached to the original invoice and becomes a part of its approval record.

At the end of each month, before closing his accounts for

the month, the supply agent makes a personal visit to each mine yard, and in company with the foreman makes a careful check of all yard stock remaining on hand unused. This is deducted from the amount showing against the mine on the supply record, and the difference between the two shows the amount to be billed against the cost for the current month, this being done in the same manner as if it had actually passed through the supply house.

The details of our system follow: In the first place, as the stock is depleted from time to time through the drains of the service, the supply agent replenishes it by means of requisitions made on the purchasing agent of the company.

Experience teaches the supply agent what lines of material are in greatest demand, and he regulates his orders accordingly. His aim should be to keep the amount of money tied up as small as consistent with ability to care for his dependent operations.

The order on the purchasing agent first goes to the general superintendent for approval, thus again giving the man responsible to the management an opportunity to keep in touch with the details of money expended for material.

No one but the supply agent having authority to make requisition on the purchasing agent, it is not a difficult matter for the general officer to keep run of the material being purchased, while under the old scheme, with requisitions coming from every point on his operation for the same lines of stuff, it was practically impossible to keep any consistent line on it.

In due course, the goods having been purchased, the invoice for them is received, in duplicate, by the purchasing department, where it is checked as to price and extensions, and passed to the auditor.

The auditor lists the invoices and sends the original for approval to the mine, if the material has been shipped direct, or to the supply department if the goods are for stock. The original copy of the auditor's list goes to the supply agent, showing to what point each invoice has been sent. The mine foreman approves the invoice when it is to cover goods received direct from the shipper, and passes it along to the supply agent for record, and to be included in the total of the material used for the month, all invoices passing through his hands before going to the general office for final approval and payment.

All invoices received at the supply department are entered in a permanent invoice record.

On their entry in this record they are given a record number, by which means they are easily identified in future handling.

After being checked against the material which they cover, they are carried to the permanent stock record, and the price of the goods, plus the transportation charges, and any other item entering into their cost, in the supply house bins, is carried to a total, and reduced to a unit price, at which price they are sold to the mine.

After passing through the supply department records, the invoices are approved by the supply agent, and sent to the general office.

At the time they leave the supply office they are listed and a copy of this list sent to the auditor.

These lists show a total at the end of the month of what items have been purchased on account of stock, and the auditor balances the total amount he is charging to stock for the month against the total of the lists received from the supply agent, and as these lists show what the supply agent says he is charging to stock, it will readily be seen that the two accounts are bound to agree on the debit side of the ledger.

When the supply agent bills out his material after filling his requisitions, he makes an entry of the total of each of his invoices for each mine in a record provided for that purpose.

This record is ruled to fit the form of invoice, and its total is the amount that is to be credited to the stock account and charged against each mine. At the end of the month the auditor lists all of the approved supply department invoices he has received, showing the amount he is to credit to the stock account for the month. This is entered under auditor's credit on the supply record and shows to a penny how the two departments agree on the credits the stock account is to receive.

As a further means of agreeing on the standing of the stock account, the supply agent makes a monthly summary of all amounts to and from the account, and shows the balance to be carried to the next month's account. Any discrepancy in the amounts one way or the other is taken up immediately and such adjustment made as is necessary to line up the account and thence we see that it only requires careful watching on the part of the supply man to keep his goods on hand up to the amount called for by his records, to be at all times in check with the auditor.

The way we have adopted in our house of keeping the stock up to the records, is to watch the matter and when we find a line of material getting low we make a check of what is left and compare it with our records and balance it up.

By this means we are able to catch up any little shortage that may occur and retire it before it is able to join a lot of other little shortages and make a big one.

We find that our account will invariably run ahead, for the reason that in figuring unit prices they seldom divide out to the exact penny, and we take the next higher full cent for our unit. In an account the size of ours, this item amounts to considerable in the course of a year.

We have thus shown in a general way what a simple matter it is to avoid shortage in our supply account when we are willing to organize for it, and go after it in a businesslike manner.

The benefits of the organized supply department do not stop, however, at the mere improvement in handling, disbursing, and accounting.

They are of a much wider scope. For instance, when we organized our department we found our boys pretty badly spolied by the habits they had acquired under Mr. No-System. Supplies had been allowed to lie around indiscriminately, and they were easy to get. Being easy to get they were not appreciated. It was easier to put in a new valve than to bother sending away for a new stem for the old one, and so with all other classes of repairs. Economy in their use did not enter into the practice in vogue.

As a lesson to the boys, we made it a practice to watch for such little worn material, which had been discarded, and it was gathered up from time to time and sent to the supply house. There we went over it carefully, and renewed all worn parts, and placed it in practically new condition, at a very small percentage of its original cost. It was then sent back to service at the price of a new article, and the resultant profit allowed to accumulate to the credit of the stock account.

The first 18 months gave us a profit from this source amounting to \$1,658.32. At the end of that period it was credited to coal cost for the year, and that gentleman thus got back some of the losses he had suffered under the old plan.

Another important opportunity for saving offered the supply agent, is the plan adopting standards for every line as far as conditions will allow.

After having decided on a standard article, it becomes possible at the central house to lump the total requirements along that line, and on the basis thus arrived at to secure contracts direct with a manufacturer of the goods wanted, at prices much more favorable than those paid to jobbers or middle men.

To show the saving possible along this line, allow us to quote briefly a few items from our own account.

First we will glance at the item of brake shoes. When we were buying our shoes, a set at a time, from each of the concerns who built the machines, it cost us on an average, counting transportation, etc., \$1.30 per shoe, or \$5.20 per set of four, for our locomotive shoes. Our car shoes and larry shoes were correspondingly expensive.

A standard locomotive shoe, and a standard for cars and

larries was adpoted, our yearly requirements estimated, and bids asked for a contract to cover them.

In due time one was placed with a prominent brake-shoe foundry at a price which brought our locomotive shoes down to 60 cents each and all others on the same proportion, resulting in a saving on our first year's requirements of $256.

Take again the item of brass and other composition bushings for bearings, etc., used in such abundance about a coal plant, where the conditions are so adverse to long life in linings.

Nearly every machine on the place has from two to a dozen which need frequent renewals. The machines themselves are made in widely scattered localities, and if your buy your linings from the concerns who manufacture the machines you not only have to shop all over the country for what you need, and pay correspondingly heavy freight bills, but you also pay the machine builder a midleman's commission on what he sells you, for very few of them maintain their own factory for making this kind of material, but buy by contract from some foundry or machinist.

A contract placed direct with a brass foundry, covering our total requirements, netted us a saving of 30 per cent. on previous prices. This meant to us a saving of from $400 to $500 yearly.

We have a couple of machines using considerable detachable link chain for driving its conveyers. The chain put out by the builders of the machine was of the ordinary soft metal used in such chains, easily broken, and quickly worn out under the adverse condition in which it was working. After a careful study of these conditions, they were laid before a manufacturer of the type of chain needed and suggestions requested as to the adoption of a type of chain for this work which would lead to greater economy. They sent us a sample chain for which they charged us 28 cents per foot. At this price, a complete chain for the drive costs us $11.20. The new chain lasted over 1 year with 64 cents worth of repairs, making a total cost for the drive for the year $11.84.

The old chain cost us 30 cents per foot, and we used up 20 feet per week. This means that we spent $312 a year for 1,040 feet of the old conveyer chain to keep our drive going. Add to this the time lost by the machine runners in renewing the chain as it broke, which caused the day-workers to run into overtime, and you will get an idea of what our reform in this line meant to us.

We want to convince you without tiring you, and we will refrain from any mention of something like 20 or 30 other lines on which we were able to show similar savings after installing the use of system in our supply account.

By no means the least of the arguments in its favor is the relief from the necessity of carrying so many duplicate parts of the same kind at each mine, which, while not so important in some items, it amounted to considerable in others.

To illustrate, let us compare a few of the leading items we found that we were carrying unnecessarily when we abandoned our local house.

Description	Found on Hand	Necessary to Carry	Price	Excess Investment
Locomotive end frames.......	6	2	$ 51.82	$ 207.28
Locomotive field coils.........	5 set	1	196.20	784.80
Locomotive armature coils....	3 set	1	25.00	100.00
Locomotive main drive gears..	9 ten ton	2	35.52	248.64
Locomotive main drive gears..	5 five ton	1	23.55	94.20
Locomotive controller cylinder	6	1	29.84	89.52
Locomotive armature housings	4	1	25.66	128 30
Locomotive finger boards.....	11	1	4.59	45.90
Mining machine cables........	5	1	45.27	181.08
Mining machine cutter chains..	2	12	13.50	136.50
Total overinvestment.........				$2,016.22
Interest on overinvestment for 1 year at 6 per cent.........				$ 120.97

If we can find an overinvestment of this size in ten items of our stock account, do you not believe we could show you a pretty nice saving if we took your time long enough to go over the entire list?

Some one may object that the cost of maintaining an establishment such as described would be prohibitive to any but large operations.

Allow us a moment to take care of that objection. In our own department, which has now been in existence for $2\frac{1}{2}$ years, we have taken care of the needs of five plants. All of the bookkeeping work, the report work, the handling of the stock in and out of the house, and keeping it in a neat and orderly arrangement while in the house, and in fact every detail that pertains to a business of the size described has been handled by one man, and in addition to doing the work described, he has found time to do much special work, entirely foreign to the supply work, as occasion demanded. At least two additional plants could be taken care of without adding more help. For the years 1908, 1909, and 4 months of 1910, we found it necessary to charge commissions amounting to $4,074.58, this charge covering the entire expense of our supply organization during that time.

During the same period it cost us in prepaid transportation to haul the goods from the supply house to the mines $599.

These two items are the only additional ones entering into the cost of central house as compared to the local ones.

Let us see if we cannot offset them by using only the few examples we have cited in our article, without touching at all on the great body of the account.

When we checked up and closed out our local houses we found an average shortage of $800 for each of the four houses in use at that time. That made a total shortage of $3,200, which covered goods paid for in cash and unaccounted for.

As far as practical business methods are concerned this item constitutes a loss, and if you cut off a shortage you save a loss, therefore we claim credit against the cost of our central department for the amount of the shortage in the previous year, which we cut off entirely.

In addition to stopping the shortage, the account showed a gain of $1,658.32, made possible by having a man to look after material and see that it was not thrown away while yet fit for service. In other words, we saved from the scrap heap the amount mentioned.

Lining the items up in a statement form for the 28 months above mentioned, we find it looks something like this:

Shortage abolished ...	$3,200.00	Cost of supply Dept..	$4,074.58
Gain by salvage......	1,658.32	Cost of prepaid transportation..........	599.00
Saving on brake shoes .	600.00		
Saving on brake brass .	934.00		
Saving on conv. chain..	728.00		
	$7,120.32		$4,673.58

This shows a balance in favor of the central house of $2,446.74, on the few items mentioned. To this could be added the interest saved on duplicate investment under the old style, the amount lost through shut downs caused by lack of material needed when we do not have it where it can be located promptly, and many other items, if we cared to take the time to go into further detail.

The results obtained in our own case are available to any operation of like extent and character, or to any combination of independent operators who care to combine and run a joint supply department and reap the benefits to be derived therefrom.

We believe our system to have been boiled down to the smallest possible amount of so-called red tape consistent with proper record of the transactions involved, still making our records plain to any one who has an interest in examining them.

Briefly summed up then our method has the following advantages, viz.: from the operator's standpoint, it involves less capital invested in supplies than under any other system, and insures a saving in their purchase, and that they will be properly cared for and accounted for after being purchased.

From the standpoint of the general operating official, it keeps him at all times in touch with the expenditures of the company's money for material, and gives him consistent control of what is placed in use at the mines. For his additional information along that line we report to him every 10 days the total amount of material charged against the production at each mine, and from being in touch with the daily production he can readily figure how liberal he can be in approving requisitions and still keep his cost within bounds.

From the auditor's point of view it does away with the annoyances which accompany the loss of invoices, delayed reports, improper charge out, and a dozen other details that apply to his work when he is trying to keep straight on a big account when it is handled by several men in as many different ways before it reaches him.

The mine foreman likes it because he has found it a pretty nice thing to let some other fellow do the worrying about getting his repairs to him when he breaks down. All he has to do now is to go to the telephone and make known his needs for the occasion, and it is up to the supply agent to do the rest.

What further argument do you need to convince you that it is possible to save money and trouble by adopting a rigid system of handling your supplies, if you do not already have it, and after its adoption following it up with a strict enforcement of the lines laid down.

NO PROGRESS IN ALASKAN COAL MINING

The condition in the coal fields in 1909 presented a contrast to that in the gold-placer districts. Not only was there no industrial advance but in some regions there was decided retrogression. The long delay in the issuance of patents to coal lands and the popular clamor against all Alaskan coal claimants has discouraged claimants and investors and it should be hoped that it may soon be possible to devise some reasonable or satisfactory means of exploiting Alaska's wealth of coal, which is so much needed for development of Alaska mines.

THE TAYLOR CONCRETE BREAKER

Written for Mines and Minerals, by E. B. W.

Coal-mine operators have watched with interest the construction and progress of the new Taylor coal breaker of the Delaware, Lackawanna & Western Railroad. This is the first all reinforced-concrete breaker constructed in the anthracite fields, although the Pine Hill breaker, at Minersville, Pa., was built in 1906, of reinforced concrete from the foundations to the main breaker floor, including the coal pockets, slate pockets, and shaker and jig supports.

The First All Reinforced Concrete Breaker in the Anthracite Region

Under favorable conditions the average wooden anthracite breaker has a life of about 20 years, but mostly conditions are unfavórable to such longevity, and this, coupled with the rapidly advancing price of timber suitable for such structures, has caused engineers to consider the more durable iron and reinforced concrete as building materials. Wherever anthracite is prepared wet the timbers are alternately dry and wet, conditions which are unfavorable for keeping timber sound indefinitely as in other buildings, therefore sooner or later rot sets in and the structure is weakened. Where structural steel has been adopted in wet breakers it has been found to corrode badly, especially where mine water is used in cleansing and preparing the coal. Before the Pine Hill breaker was constructed the engineers made a series of tests to ascertain what effect mine water had on concrete. Their experiments, which proved very little, have been confirmed by the structure, which has been completed over 4 years and shows no signs of disintegration or other weakness.

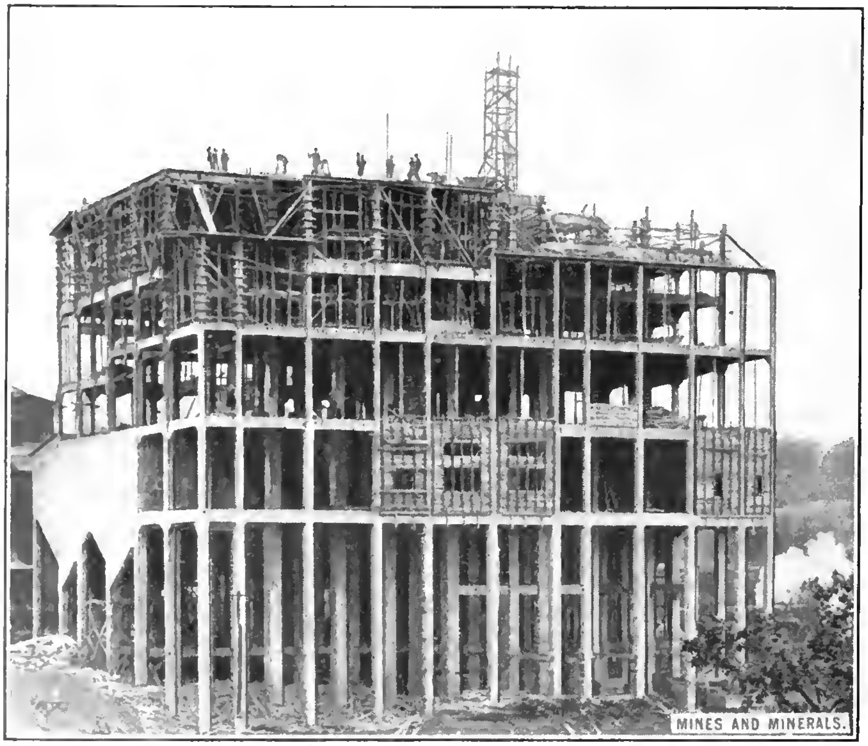


FIG. 1. TAYLOR CONCRETE BREAKER IN COURSE OF CONSTRUCTION

The Taylor breaker, as a model for future reinforced concrete breakers, has developed the difficulties of construction to be overcome in future structures of this kind to such an extent that duplicate breakers could be built in much less time and at reduced cost.

The rear of the breaker in the course of construction is shown in Fig. 1. While the bents from the rear to the front of the breaker are evenly spaced, those from side to side are spaced to accommodate the machinery and support the weight of the machinery and coal. The first two rows of posts in the rear of the breaker are about 2 feet square and up to the second or pocket floor have a length of 65 feet. Directly under and between the pockets where the most weight will come the 2-foot-square posts are supplemented with 36-inch-square posts. As the posts are carried upwards this size is decreased until at the top floors they are but 12 inches square.

In Fig. 2 the breaker is shown practically finished with the top forms still in place, and as it looks today from the outside, could be taken for an office building. It will be noted that the architect has furnished windows for daylight; also each post is supplied with tubes for wiring for electric lights, thus making this breaker an exceptional one for light and comfort of the employes. In the construction of the posts and beams, corrugated rods $1\frac{1}{2}$ inches in diameter were placed at each corner, then bound with hoop bands and wire so as to form a rectangular cage. They were reinforced by smaller corrugated iron rods from $\frac{3}{8}$-inch diameter up to $1\frac{1}{2}$-inch, as the occasion demanded. Inside this cage pipes for the electric wiring were placed and then wooden forms were constructed around these skeleton post frames. Concrete was poured in a little at a time from the top

of the forms and tamped with iron rods about the pipes and rods. In this way the posts were built up to the height of a floor, where the iron corrugated rods for the beams were tied with those of the posts. The beams were made the same as the posts. Forms were built around the skeleton beams and then concrete tamped about the rods and pipes. A description of the construction of these posts and beams from the foundation to the top of the breaker, a height of 149 feet, would in no case adequately explain the delays, petty annoyances from changes necessary in forms and the difficulties of handling material. However, the builders have learned by the experience in what manner they will be in future avoided.

It will be noted that the coal pockets are large and provided with two chutes so that two cars can be loaded from the same pocket at the same time. Each chute is provided with a gate and lip screen. Under the lip screens there will be a scraper line which will carry the fine material to an elevator bucket and back to that part of the breaker where it may be utilized. There are 14 chutes for each track, or 28 for both tracks. To the rear of Fig. 2 is seen the old Taylor breaker, constructed of wood. This structure, which is built over the shaft about 200 feet from the new breaker, must be wrecked before the new breaker is put in commission; then above the shaft will be erected a steel head-frame, connected with the top of the new breaker by a steel incline about 218 feet long carrying a scraper line. Self-dumping cages will be hoisted 50 feet above the shaft collar, then discharged into a boot feeding the scraper line. The coal is to be discharged from the scraper line at the top of the breaker, riddled over bars, and from the bars sent by gravity to the various crushing and sizing machines, finally reaching the coal pockets. About two-thirds of the breaker will prepare coal dry, the other third will be for washed coal.

Another new feature in connection with this breaker is the installation of individual electric motors for driving the individual machines used in the preparation of coal, and it was owing to this feature that so much care was used to place pipes for wires in the posts.

The D., L. & W. Co. have in recent years electrified their breakers and mines, using power generated at conveniently situated electric plants. The power for the Taylor breaker will be supplied by the Hampton electric plant situated about 2½ miles distant, in Keyser Valley.

It may be of interest to know that while the Taylor breaker is concrete, 500,000 feet of lumber will be needed inside for machinery bed-plates and other fittings. However, very little if any of this lumber is large sized and there are no sticks such as breaker framing demands.

We are indebted to S. M. Ives, foreman for D., L. & W. Co. for the information contained in this article.

ENGLISH SLAG PAVING BLOCKS

Scoria bricks, or slag paving blocks, have been manufactured in the Middlesbrough district, England, for many years and considerable quantities have been shipped to the United States since 1894. The bricks are manufactured from molten slag from the blast furnaces. The slag is drawn from the furnace into slag cars lined with firebrick and hauled to the brick plant. There the slag is poured into iron molds made with a hinged bottom and mounted on the circumference of a circular iron framework. This revolves and allows the molds to be filled separately. As the bricks solidify they are removed and placed in a beehive soaking oven, where the residual heat anneals the whole of the brick.

PERSONALS

John Gross, of the firm of Draper & Gross, mining and metallurgical engineers, 746 Equitable Building, Denver, Colo., has gone to Sonora, Mexico, on professional business.

On November 9, Ellis Soper, of Detroit, delivered a lecture on "Rotary Kilns," before the A. S. M. E. in New York City.

Frederick A. Delano, President of the Wabash Railroad Co., addressed the students of the University of Illinois on October 25. His subject was "The Railway as a Profession."

A. Bement, of Chicago, addressed the students of the University of Illinois, October 22, his topic being "The Practical Uses of Coal Analysis."

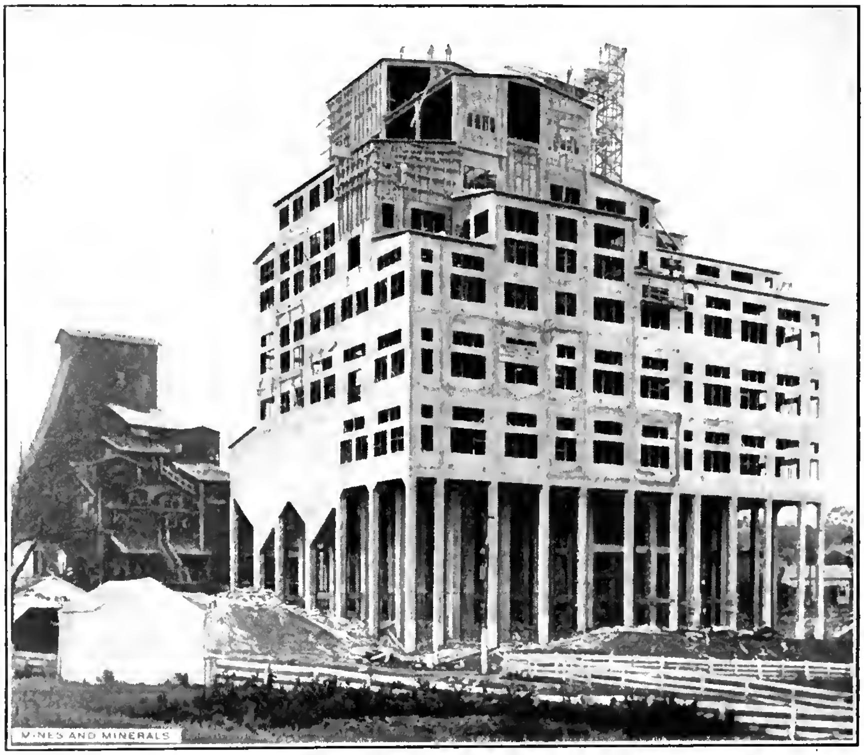


Fig. 2. The Old and the New Breaker at Taylor, Pa.

George F. Duck, E. M., Western Editor of Mines and Minerals, has returned to his office in Denver, after an extensive tour through the West and Southwest in the interest of the western readers of Mines and Minerals.

Kenneth Seaver has been appointed chief engineer of the Harbison-Walker Refractories Co., of Pittsburg, Pa.

Prof. C. T. Knipp, of the Physics Department of the University of Illinois, is in Europe on a year's leave of absence. He will spend the greater part of the year in Cambridge studying under Prof. J. J. Thompson.

Prof. H. H. Stoek, of the mining department of the University of Illinois, has been visiting various cities in the East in connection with his work.

William C. Cuntz, for 18 years with the Pennsylvania Steel Co., has accepted the management of The Goldschmidt Thermit Co., with offices in New York City.

At the November monthly meeting of the A. S. M. E., at their club rooms in New York, Charles W. Baker, editor of *Engineering News*, gave an illustrated lecture on the Panama Canal.

MINE FIRES

*By Thomas K. Adams**

While the death rate resulting from mine fires in this country, in the years past, has not been high compared with those resulting from other causes, yet it is sufficiently high to warrant us in devoting some time to the proper consideration of such a subject.

Lessons from Disastrous Fires in the Past—Precautions—Dealing with Mine Fires

We are cognizant of the fact of the direful effects both in the loss of life of the workingmen and the destruction of valuable property, large expense, and general annoyance to the mine owner.

The causes and effects of mine fires, and their preventatives, are all reasonably well known to every intelligent, thoughtful, and practical mining man. Such disasters ought to be classed as being preventable.

In looking over the accounts of some of the mine fires which have happened in this country, which have startled the general public more than others, I was forcibly struck with three of them (Avondale, Hill Farm, and Cherry), especially in the general aspect at least of the similarity of their cause and effect, and of the cycle of years between each. The Avondale Mine was a single-shaft opening. The structural material used in the shaft lining, partitions, derrick, and breaker, was composed of wood. The fire originated at the bottom of the shaft, caused by the carelessness of the furnace man in lighting the furnace fire, thereby setting fire to the wooden partition, etc. This fire occurred in the month of September, 1869, and in it 109 lives were lost. As you remember, no adequate means were at hand with which to extinguish the fire.

Hill Farm Mine fire was the direct result of a very thoughtless act on the part of the mine superintendent in directing the work of cutting into a drill hole adjacent to the main slope, which was 520 feet deep, with explosive gas and water in it, and doing this work on the day shift when men were at work beyond the location of the bore hole toward the bottom of the slope; the said hole being penetrated by the miner's pick, released its contents, consisting of water and explosive gas, and the boy Hays, standing nearby, seeing the immediate danger, he, with manly presence of mind, thought of the men at the bottom of the slope, and while he was on his thoughtful mission in passing the location of the fateful bore hole, with his open light, to warn the miners below, he ignited the gas, the flame therefrom set fire to the brattice cloth on the opposite side of the slope; the flame from the brattice cloth in turn set fire to the wooden overcast, the men beyond the fire were trapped and their means of escape cut off. This mine fire happened in April, 1890, 21 years after the Avondale disaster. Thirty-one persons were lost, and now, with 19 years intervening, the public was almost paralyzed by the effects of the Cherry Mine fire.

The Cherry Mine disaster, the effects and details of which you are all more or less familiar with, originated at the No. 2 seam landing of the escapement shaft and was caused by the ignition of hay from the flame of a crude, improvised, unprotected illuminating contrivance. The flame from the hay was communicated to the overabundance of wood supporting material at the landing, and adding thereto the inadequate means available to successfully deal with a fire of such magnitude, with the ill-judged actions of the inexperienced men at the bottom, the trap was complete and the men caught therein, so we have now to record the greatest and most disastrous mine fire in the history of the coal-mining industry of this country, so far as the loss of life is concerned. Two hundred and sixty-eight lives were lost in the Cherry Mine disaster.

After the Avondale disaster it became very plain even to our law makers that two openings, separated and apart, were necessary to every seam of coal to afford a reasonable measure of protection to the workmen in the mine in the event of mine fires at shafts or slopes.

The Hill Farm Mine disaster made again evident to our legislators the additional necessity of providing better means of exit to and from mines for the miners, and the elimination, under certain conditions, of the use of combustible material in the erection of overcasts and permanent stoppings, and providing additional passages for traveling ways, etc.

Now that the Cherry Mine disaster has passed into record, with its baneful effects, keenly felt by all sane men, will the lesson to be learned therefrom be remembered and heeded by us?

The coal-mining operations are visited by other mine fires, from other causes than those mentioned, and which are destructive in their effects, such as fires at mule stables, at oil shanties, electric and gasoline pump houses, from blasts firing explosive gas and fine coal at the face of the workings of the mines, fires at trap doors, fires on intake and return air-courses, probably from carelessly constructed and unprotected electric cables or wires, etc., and gob fires caused by reason of leaving fine coal and other combustible matter in them.

It has always appeared to me that the causes of mine fires were so apparent to the thoughtful and intelligent mining man that their occurrence and their ill effects were unnecessary and an unnecessary burden to be borne by the coal-mining industry of this country. The prevention of mine fires lies in the removal of the causes, which are well known, and the knowledge of means and methods to be employed for their elimination, being within the range and scope of the ability of the ordinary mine official, the wonder is that they do happen. To secure freedom from mine fires I believe lies almost entirely within the intelligently directed administrative powers of the mine management, and in my opinion if the mine officials are careful, alert, and capable, immunity from them can be secured.

When the people engaged in the coal-mining industry realize that the mining of coal is surrounded by many hazards, when they settle down to a more sober reflection and learn to reason out the fact that certain causes will produce certain effects, that life is more than meat, more safe and sane methods in operating the coal mines will prevail in this country.

Preventive Measures.—There should be employed at every mine a competent, thoughtful, careful, and ever-vigilant mine foreman to have exclusive control of the employes and the workings of the mine; and a part of his duties should be to see that all employes are instructed and trained in the use of all fire-protective appliances, to keep the mines free and clean, as far as practicable, of all refuse combustible matter, and that other protective regulations, through the maintenance of rigid discipline among all employes in the mine, are observed, and that the details of the operating system relating thereto be strictly attended to.

Every coal mine should consist of two separate openings and one of them should be used exclusively for an escapement, and these two openings should be separated by at least 500 feet, if practicable. The escapement shaft, if over 100 feet in depth, should be equipped with safe and efficient hoisting apparatus. The structure at the hoisting shaft should be built of steel, and the engine and power houses should be built of concrete, brick or masonry; the shaft linings to be of concrete, and the shaft bottoms, if needing supports for the roof, should be of steel I beams, concrete, or brickwork; doors between main shaft and the escapement shaft should be so located as to be easily accessible to the workmen from all parts of the mine by convenient traveling ways, other than those which lead directly to the bottom of the hoisting shaft; mule stables, if not entirely prohibited in the mines, should be built of incombustible material and illuminated with protected incandescent electric lights; all oil, electric and gasoline pump houses should be kept free from combustible material, and be built of concrete, brickwork, or

* Paper read at Summer Meeting of the Mining Institute of America, 1910.

masonry. When the main workings of a mine have advanced 5,000 feet in length and the remaining extent of the property and the other conditions warrant it, an auxiliary escapement opening should be provided and equipped with efficient and necessary machinery; a water system under sufficient pressure, and men of the different languages instructed and trained in its use, should be installed at all important mines, and the said system should be carried into the interior of them, and the whole equipment tested frequently to prove its efficiency, and all parts and connections kept in first-class condition and ready for use at all times; all electric cables or wires, etc., should be well supported and insulated, and not allowed to come in contact with combustible material. In the blasting of coal, permissible explosives should be used; mines should be patrolled by experienced and reliable miners after shots have been fired; all combustible material should as far as practicable be removed from gobs or abandoned parts of the mines and then kept as well ventilated as possible; a telephone system should be provided at important mines so that communication can be had between persons outside, and at all important stations inside of them; refuge chambers, efficiently constructed and equipped and conveniently located should be provided in all large and dangerous mines.

Mines should be provided with a powerful reversible fan, and it should be placed on a separate shaft, cased in steel, and fitted with relief doors opposite the direct line of tunnel or shaft, and so situated as to be out of the direct line of the possible dynamic forces which may result from explosions.

Dealing With Mine Fires.—In dealing with mine fires time is an important factor. A mine fire must be quickly and accurately located and dealt with promptly. The first thing to do after a fire has been discovered is to warn the men in the interior of the mine and have them removed as expeditiously as possible. Never forget this safety measure, no matter how trifling the fire may appear at its inception, especially if the fire originates at a dangerous location in or about the mine. Prepare to fight the fire with the means at hand, and at once. The air-currents must be kept under perfect control by the parties dealing with the fire. The main point to keep in view in the control of the air-currents is to so conduct them as not to carry the smoke to the workmen or to carry the explosive gas to the fire, if there be any accumulation of it at hand. The smoke must be carried directly to the return air-course. Whether or not the fan has to be reversed, or the air-current reversed or short-circuited, will depend on the location of the fire and the surrounding conditions. This matter will be determined by the persons having control of the work. The men dealing with the fire must fight it from the direction of the intake air-current, keeping the current always under control by means of temporary brattice work. The workmen must use safety lamps while doing the work. The fire must be fought persistently by means of hose and water and extinguishers and the fight must never be given up while hope lasts, or until it is certain that the fire has passed the stage of control.

The utmost care should be exercised by those engaged in such work to protect themselves from any dangerous gases which may be produced while dealing with the fire. If the well-directed efforts put forth to extinguish the fire have failed, it should be sealed off promptly so as to exclude the air, at the same time observing the necessary precautions when building the dams to provide them with the escapes so that accurate readings can be had from time to time of the condition of the enclosed atmosphere and of the temperature and nature of the gases in the fire zone. After a reasonable time has been allowed, if the exclusion of the air has not accomplished its purpose, the burning area must be filled with water and allowed to remain flooded until there is no doubt of the fire having been extinguished.

In making preliminary inspections to determine the exact location of a mine fire, we would advise the persons while doing so to use the rescue apparatus or oxygen helmet, and when going to make tests to ascertain the conditions surrounding the fire that they should use such apparatus by all means.

To successfully deal with mine fires the human factor engaged counts for more than anything else. The ideal man to cope successfully with mine fires must have decision of character, good judgment, and general intelligence; be physically strong and not afraid to use his strength, courage, and great tenacity and persistency; and above all, when conditions are well considered, decisions made, and plans and methods agreed upon, do not allow anything emanating from irresponsible parties to intervene in the carrying out to a finality the plans agreed upon.

COST OF SINKING A MEXICAN MINE SHAFT

The following costs of sinking a mine shaft through andesite, at the Esperanza Mine at El Oro, Mexico, are given by Mr. W. E. Hindry.* The shaft was a three-compartment vertical shaft, having two 5'×5' hoisting compartments and a 5'×7' pump and ladderway. The timbering was 10 in.×10 in. with 2-inch lagging; sills 5 feet center to center, and 6 posts per set. The total depth of the shaft was 679 feet, of which 101 feet were sunk by windlass and hand work, and 578 feet by steam hoist and machine drills. The work was done in 1899 and the prices of materials and wages were as follows:

Materials	Prices
Timber per M ft. B. M.	$13.58
Wood per cord	3.15
Coal per ton	7.27
Powder, 60 per cent., per pound	.14$\frac{3}{4}$
Fuse per foot	.0055
Caps, each	.0058
Candles, each	.0194
Labor	
Superintendent, per 24 hours	$4.850
Shaft men, foreign, per 8-hour shift	3.220
Shaft men, native, per 8-hour shift	.528
Top men, per 8-hour shift	.422
Fireman, per 8-hour shift	.485
Hoistmen, per 8-hour shift	.970
Blacksmiths, per 8-hour shift	1.455

The cost of excavation was as follows:

Labor	Per Linear Foot
Superintendence	$2.529
Shaftmen, foreign	3.510
Shaftmen, native	7.043
Top men	.578
Blacksmiths	.718
Firemen	.317
Hoistmen	.894
Miscellaneous	.936
Total	$16.525
Materials	
Timber	$ 3.961
Wood, fuel	3.781
Coal	.179
Powder	2.853
Fuse	.014
Caps	.294
Candles	.223
Oils, grease, etc.	.025
Miscellaneous	.051
Total	$11.381
Grand total	$27.906

The above costs are converted from Mexican money assuming the peso to have a value of 48$\frac{1}{2}$ cents.

**Mining and Scientific Press*, April 9, 1910.

MINE ACCIDENT PREVENTION

*By J. J. Rutledge**

Mine-rescue work is done most efficiently by men who have been carefully trained in such work, who are in good physical condition, who are equipped with the proper apparatus, who are close to the mine at the time the explosion happens, and are familiar with the mine workings. To be of any value the rescue apparatus must be kept at the mine and be always ready for instant use. Some of the employes, both outside and inside, should have received proper training in its use and should be in such physical condition that they can make immediate use of it in rescuing any men who may be left alive in the mine after the explosion. A volunteer corps composed of young, cool-headed, brave, active, and vigorous men can be of great value, if they are brought immediately to the scene of the explosion. If this corps is large enough, some members can be assigned to the auxiliary work that must be done before the men in the mine, living or dead, can be removed—such as putting up brattice, hanging brattice cloth, running telephone lines, extinguishing fires, repairing damaged machinery, and rendering first aid to those injured, while the remainder are busy with the actual rescue work.

Preparation for Rescue Work. Importance of Uniform and Efficient Laws

Those mining men who have somewhat critically observed the operation of the rescue apparatus now used by various rescue stations should remember that this is not the only apparatus of the knapsack type that has been used in rescue work, but there are about five or six other patterns, some using liquid air instead of oxygen. It is perfectly practicable to use these other patterns instead of the type at the rescue station, but it should be remembered that some of the patterns have a more limited range of action.

Prevention of Accidents.—In recent years the medical profession has made great advances in combating disease, but perhaps its greatest achievements have been in what is called "preventative medicine," that is, in endeavoring to prevent disease by doing away with unsanitary and unhealthful conditions, thus removing the necessity of expending more energy in trying to overcome disease after it has made its appearance. This is why some of our Alabama mining camps are so carefully looked after by the physicians in the employ of the various mining companies. In like manner, coal mining should have its methods of "preventative medicine" for its mine accidents, besides and in addition to its method of mine-rescue work after an accident has happened. It is of the preventative phase of the subject that the writer proposes to speak.

Laws.—It will be difficult to enforce safety provisions in coal mines, and to decrease the loss of life, so long as the inspectors, superintendents, mine foremen, and managers are not working under a good mining law. The laws governing mining should be, as far as local conditions permit, uniform over the various coal-producing states. Coal miners are constantly passing from one state to another, and any bad practices that originate in one state, are soon widely disseminated; hence there is great need of uniform laws to meet general conditions. It is especially important that the penalties for violation of the laws should be uniform in all the bituminous coal fields.

SUGGESTED PROVISIONS OF UNIFORM LAWS

Mine Plans and Methods of Working.—The law should stipulate that previous to the commencement of mining operations the operating company must submit to the State Mining Board, or Chief Mine Inspector, a detailed plan showing the surface plant it proposes to erect, the manner in which it plans to reach the coal bed, together with the number and location of the different shafts, slopes, or drifts; the proposed method of working the coal; and the proposed plan of ventilating the mine. It should be distinctly understood that these plans, if approved, are as a whole to be rigorously followed and only departed from, if at all, in minor details. The state authorities should then carefully scrutinize the plans submitted, approve those which in their judgment are in accordance with the best mining practice, and reject those which are not. The law should grant the public officials responsible for the safety of the miners the authority, under the police powers of the state, to approve only such plans as are, in the opinion of the best mining men, most conducive to safe and economical work. The mining company would, by such a procedure receive the best of advice as to the proper choice of methods to be used, and would be assured that every possible means providing for the safety of the employes had been adopted. Many of the conditions under which coal mining is carried on are not conducive to safe or economical work, and these conditions are not always appreciated by a mining company in haste to begin operations. The approval of the plans by an impartial official or officials having authority to approve or reject will be a guarantee that the best possible plans have been adopted.

The writer believes that it is nearly always possible to limit an explosion to the immediate vicinity of its origin, if a suitable method of opening out the mine has been adopted at the outset. Dividing a mine, if it is to be worked by pillar and room, into districts or panels, each of which shall be self-contained, ventilated by a separate split of air, and separated from the remaining panels by a solid, unbroken rib of coal, is, in the writer's opinion, a most effective method of preventing the spread of an explosion.

Ventilation.—One of the most important features of any plan for safe coal mining is the arrangement, adjustment, and control of the appliances for ventilation. The fan should be so installed that it cannot be injured by the force of any explosion in the mine. To this end it should not be placed directly over the mouth of the shaft, slope, or drift, but should be placed at least 50 feet away from the mine opening and communicate with this by a tunnel which makes a sharp angle with the shaft, slope, or drift; there should be balanced explosion doors on the fan, and two independent means of operating it.

Air-courses should be large and free of all obstructions; a large volume of air moving through ample courses at a velocity sufficient to speedily remove all dangerous gases, is, in the writer's estimation, much safer than the same volume of air passing at a great velocity through narrow courses. There is much more liability to dangerous concussion in the latter than in the former. The opening and closing of doors will not as seriously affect the slow-moving current as it will the fast-moving one. The air supply should be split as much as possible, consistent with the proper removal of the dangerous gases, and each district or panel should have its own split. There should be no doors on main entries or slopes. At the mouth of main cross-entries, through which coal passes in considerable quantity, there should be double doors with a space between them sufficiently large to contain the longest trip hauled from the cross-entries. All doors should be automatically self-closing, and at all important doors there should be attendants.

In the writer's opinion, overcasts and stoppings should be made strong enough to withstand only the force of the ordinary ventilating pressure, in order to insure their collapse under the sudden violent pressure exerted by an explosion. In addition, the writer believes there should be explosion doors built into every overcast or undercast. The writer is convinced that by such precautions the danger of widespread explosions would be materially lessened.

In case doors are permitted on main haulageways their operation should be regulated by statute. In some coal fields it is an offense, punishable by fine, to allow an important door upon which the safe ventilation of the mine depends, to remain open.

* Read before the Alabama Coal Operators' Association.

Miners.—Every miner should be shown the risks in using explosives, and the safe methods of handling them. He should, if at all possible, be given ocular demonstration of the fact that coal dust is explosive. This can be done by means of laboratory apparatus. He should be compelled to properly timber his working place, and should be held strictly accountable for the condition of it. To insure the safety of the miner in his working place and in his going to and from the mouth of the mine to the same, suitable rules should be enforced by the mine foreman or manager, violation of these rules to be punished by fine and imprisonment. Those in charge of coal mines and those responsible for the safety of the mine employes should have their actions backed by the power of the law.

The miner should be made to understand and to fully realize that if he places and drills his holes in a workmanlike manner he will not only produce the maximum amount of merchantable lump coal, with consequent financial benefits to himself, but will also be working under safe conditions; but if on the other hand, he does not exercise judgment in the placing and drilling of his holes, the chances are about nine to one that he will not only endanger his own life, but the lives of all others in the mine.

The writer, however, does not place all the blame for mine accidents on the miner by any means. A great deal of the trouble is due to poor discipline in the mine and to lax supervision by mine foremen and managers. Too often the miner continues following dangerous practices, taking chances day after day, because he is permitted to do so. Were he punished when detected in dangerous practices, and were his punishment required by law, he would soon cease to endanger the lives of himself and others. An inspection of the mines in those states which have and enforce strict regulations on the use of impure oil, size of powder cans, and shooting off the solid, will soon convince any one that the average miner has a proper respect for the law when strictly and impartially enforced. As far as the miner is concerned the remedy for the greater mine accidents, though much can be done by education, lies in the establishment of strict mine discipline.

Deputies.—In every large mine there should be for every 50 miners a practical competent miner to act as a deputy foreman. This man should be an assistant to the mine manager or foreman and would report directly to him. He would examine all working places at least once every working shift, see that the places are properly timbered, and that all holes were properly drilled and fired. The miners should preferably be concentrated in a limited area in order that his supervision may be as efficient as possible.

Shot Firers.—In mines generating explosive gas, or in which fine coal dust is found, all shots should be fired either by special shot firers, or from the outside by electrical appliances, after all other employes have left the workings.

Safety Lamps.—In any mine where as great a quantity as 2 per cent. of explosive gas has been found in the return air, safety lamps should be employed. Although such a mine may be perfectly free from gas for a considerable length of time, there is liable to be at any time a further outburst and any disarrangement of ventilating appliances will then render the mine dangerous.

Mine Employes Other Than Miners.—For positions that play an important part in the safe operation of the mine, such as cagers, trip riders, motormen, trackmen, timbermen, fire bosses, or mine examiners, shot firers, and all employes other than miners, only men whose courage in face of danger has been demonstrated, or who appear to be brave and cool headed, should be employed, as on any one of them at some time may depend the safety of the entire working force of the mine. Any single individual in a mine may by his own carelessness or criminal action at some time endanger the lives of all those underground. It is imperative, then, that for the responsible positions men of known courage and resourcefulness fully aware of all the dangers that surround coal mining should be chosen.

Foremen or Mine Managers.—The writer believes firmly that the person who is the greatest factor in the prevention of mine accidents is the mine foreman or manager, and has reached this conclusion after 7 years' experience gained as mine examiner, assistant mine foreman, mine foreman, mine manager, and superintendent. Although his length of service is not as long as that of some others here present, still he lived hard while in the service, and his conclusions should be given due weight. For 10 years the writer has acted as mine engineer and geologist, and in that time has visited many of the important coal mines in this country, and has been brought into close personal contact with mine foremen and superintendents of various kinds. The following, in the writer's judgment, are the qualifications of a safe and successful mine foreman or manager.

First, and foremost, he should have had practical experience in mining coal at the face; this should be insisted upon.

Second, he should be a man of good judgment, with ideas of his own, and strength of character sufficient to advance and defend his actions.

Third, and this is of great importance, he should have previously demonstrated in some subordinate capacity his ability to handle labor.

Fourth, he should be temperate in habits and truthful in character, and should possess executive ability and initiative.

Having the above qualifications, the mine foreman or manager should pass a strict examination, both oral and written, to determine his qualifications, and before he is admitted to the examination his character and ability should be certified to by at least three men of integrity and standing in the business of coal mining. The examination should be strict and impartial, so as to prevent the cheapening of the certificates of competency issued to those foremen or managers passing it; at the same time, practical knowledge or experience in mines should be given preference over the mere ability to give correct answers to technical questions relating to coal mining.

The candidate passing the oral and written examination should be given a certificate of competency as mine manager, and this certificate should be subject to annulment by the proper authorities upon the presentation of evidence of carelessness in mine management or non-compliance with the state mining laws on the part of the holder. But the state regulations should go further than this. If a man has shown himself capable of passing a strict examination as to his qualifications as mine foreman or manager, he should be given, in addition to his certificate, some insignia of authority by the state, such as a badge, and should have certain police powers in the mine over which he has charge. In this way the mining laws will be more carefully enforced and life and property conserved. If the powers thus given to the mine foreman or manager were abused or used for purposes of oppressing the miner, there would be prompt annulment of the certificate, after the facts had been established upon investigation by the State Mining Board.

In the advent of disputes relating to matters of mine discipline between the mine foreman or manager and the miners, the police power given him could not be abused, for his actions would be subject to review by the Mining Board. On the other hand, should the foreman or manager be handicapped or hindered in his work by the failure to receive proper supplies or equipment from his superiors, he would be encouraged to demand the same and would be insured against possible loss of employment by reason of his making such a demand, realizing that the law would back him up in making such demands.

It seems to me absurd to put a mine foreman or manager in charge of a large mine employing several hundred men, all of whose lives he must be responsible for, without giving him the proper means of enforcing such rules and regulations as he may make for the safe operation of his mine. Some may say that the mine foreman or manager can enforce his orders by discharging those employes who refuse to obey them; but though

this course is theoretically open to him at all times, it may be a difficult one to take. When men are scarce or strongly organized a discharge is not always a practicable solution of the problem.

A consideration of the difference in the attitude of the miner toward the foreman who represents the company and the mine inspector, who is backed by the authority of the state, will convince those who doubt the wisdom of the above suggested plan; clothe the mine foreman and manager with the authority and dignity which his position entitles him to possess, and the writer is sure that mine accidents will be very greatly decreased.

Superintendents.—The mine superintendent should be a man of practical experience, preferably with some technical training, and should possess great executive ability; he should have reached his position as superintendent by promotion from mine foreman or manager, as he will then have that knowledge of details which is essential to economical and safe operating. In matters of mine discipline he should be supreme, but he should always support the foreman when the latter is in the right and should see that the foreman works in harmony with him.

In the matter of supplies, the superintendent should disallow requisitions from the foreman, if after close investigation he ascertains that such supplies are not required, but he should never cancel any requisition for supplies that are absolutely required. Perhaps the greatest abuse of this sort is the cancellation of supplies which are required to make ventilation more effective. This strikes the foreman at a vital spot, for he is answerable to two other parties in this affair, besides the superintendent; viz., the inspector and the miners. Both place the blame on the foreman if the needed supplies are not forthcoming.

In such matters as pertain to mine management, when disputes arise between the general office and the mine officials, the superintendent and mine foreman, if in the right, should not hesitate to declare themselves, for subsequent developments will discredit their work if they fail to stand for what is right and proper.

Inspectors.—The mine inspector should be a man of mature years and of good judgment, who has a practical knowledge of coal mining and considerable strength of character. His position, in my judgment, should be under state civil service rules, and not subject to change by reason of changes in state government. He should possess police powers granted him by the state and should be empowered to conduct sworn investigations of mine accidents.

Conclusion.—The writer is fully aware of the fact that the adoption of the above suggestions will add materially to the cost of coal production at the beginning of operations, but he is quite sure that the final returns on investments in coal mines will be greater because of fewer accidents and more efficient mining.

TRADE NOTICES

Cia. Minera Santa Ana y Anexas, of Pachuca, State of Hidalgo, Mexico, has entirely equipped its property with electric apparatus, the power for operating being purchased from the Cia. Electrica é Irrigadora. The power is furnished to the mining company at 6,000 volts, 50 cycles, and in order to reduce this voltage to 400 volts, that required by the motors, a large bank of oil-insulated transformers with protection apparatus has been installed. A part of the equipment comprises a double-drum electric hoist fitted with brakes and clutches operated by compressed air supplied from a separate electric-driven air compressor. The capacity of this hoist is 9,000 pounds total load at a hoisting speed of 700 feet per minute. The hoist is fitted with a 225-horsepower, 400-volt motor of three-bearing type with usual controlling device as furnished by the Westinghouse Electric and Mfg. Co., the entire outfit being purchased through G. and O. Braniff & Co., of Mexico City, who represent the manufacturers.

The Joseph Dixon Crucible Co., of Jersey City, N. J., has just gotten out a very attractive little booklet of envelope size on their paint for steel cars. Any one interested in steel-car painting should send for a copy of this booklet which will be forwarded, free of charge.

The Bristol Company calls attention to its new model of recording pressure gauges equipped with 6-inch chart as illustrated and catalogued for the first time on page 8 of Bulletin 141. Radically new lines of instruments are also illustrated in Bulletins 145, 146, and 147. A description of the Bristol–Durand radii averaging instrument is given in Bulletin 147.

The government mine rescue cars are being equipped with Hubbell electric mine lamps. Special cabinets have been designed for them, attached to which is a charging board which makes it possible to charge these lanterns from any voltage from 700 to 110. The cabinet consists of a locker wherein extra supplies are kept, such as reflectors, compound, and a ½-gallon bottle of solution, also two shelves each to hold four lanterns, which can be charged without moving them from their places. The weight of the type of lantern used is a trifle over 3 pounds. An advantage claimed for these lamps is that, owing to the nature of the battery with which they are equipped, they can be charged and put away for a long period of idleness without losing their efficiency or diminishing their storing capacity. An additional feature of the lighting equipment for this work is the portable electric searchlight which has proved very valuable for brattice work or for searching for bodies after an explosion. This light is equipped with two powerful batteries and a 20-candlepower light on a parabolic reflector which throws a long piercing ray as well as a generally diffused light. The construction is so arranged that one battery can be charged while the other is in service, and the light can be in service for 24 hours in a day. These lights were supplied by the Servus Rescue Equipment Co., Keenan Building, Pittsburg, Pa.

The consolidation of the Terry Core Drill Co. with the McKiernan Drill Co., under the name of the McKiernan-Terry Drill Co., is announced. The new company will manufacture the various products of both companies, including rock drills, hammer drills, core drills, core cutters, sheet-pile drivers, compressors, etc., and the business office will be at 115 Broadway, New York City.

Goyne Bros., of Ashland, Pa., recently shipped to the Buck Mountain colliery of the Lehigh Valley Coal Co., at Mahanoy City, Pa., two especially large triple-expansion duplex mining pumps, the shipment requiring four large-size cars; and this shipment constituted the first one of the large pumps produced at the greatly enlarged plant in that section popularly termed the "East End." The enlargement of these works has been held back by an extremely busy season, requiring the capacity product of both plants. While orders at present on the books will keep both plants rushed throughout the early months of 1911, the addition to the machine shops and erection of a commodious office building and other structures that will comprise one of the most substantial exclusive mine pump manufacturing plants in the country will go forward with the dispatch characteristic of Goyne Bros.

Frank Toomey, Inc., 127-131 N. Third St., Philadelphia, Pa., is placing upon the market the fomous Reilly steam pumps.

This line comprises mine pumps, fire pumps, water works pumps, circulating pumps, pumps for wrecking, irrigating, artesian wells, oil lines, quarries, etc.; also air pumps and air compressors for various purposes.

ANSWERS TO EXAMINATION QUESTIONS

Written for Mines and Minerals, by J. T. Beard

Mine Bosses' Examination Held at Terre Haute, Ind., August 17, 18, 1910

QUES. 1.—Omitted.*

QUES. 2.—(*a*) What are the requirements of the Indiana statute relative to the visiting of working places by mine bosses? (*b*) What are the benefits that should be derived from conformity with this statute? Points 6.

ANS.—(*a*) See Section 33 of the Indiana mining laws. (*b*) The working places will be kept in better condition and greater safety will be assured in every part of the mine. Fewer lives will be lost and fewer persons injured by falls of roof and coal. Work will be done with more care and system and the daily output of coal increased. The mine boss will be better acquainted with his men and the needs of the work.

QUES. 4.—(*a*) What are the factors necessary to insure efficient mine ventilation? (*b*) What are the conditions that should determine the quantity of air to be circulated through a mine? (*c*) What are the conditions that frequently arise that necessitate an increase in the quantity of air circulated through a mine? Points 9.

ANS.—(*a*) A sufficient volume of air must be circulated throughout the mine to make it healthful and safe. The air-current must be made to sweep the working face, and must be so distributed that the volume and velocity of the air passing in each section of the mine shall be sufficient for the needs in that section. All abandoned workings must be sealed off or ventilated to prevent a dangerous accumulation of gas; and all void places and headings must be kept free from gas by the use of temporary brattices if necessary. (*b*) The quantity of air to be circulated in a mine should be determined, aside from the requirements of the law, by the number of men and animals employed, the gaseous condition and size of the workings, the size of entries and rooms, method of working, depth of the seam below the surface, and area of abandoned workings requiring to be ventilated. (*c*) Any sudden outburst of gas, or increase in the flow of gas into the workings, due to a fresh blower, a fall of roof, or striking a fault where gas is given off freely. The opening up of a new area or section of a mine; or the occurrence of a mine squeeze, which may cause a loss of air due to leakage of doors or stoppings. All of these may require an increase of the circulation.

QUES. 5.—An airway 14.5 feet wide and 6.5 feet high is passing 47,125 cubic feet of air per minute; what is the anemometer reading? Points 4.

ANS.—The sectional area of this airway is $14.5 \times 6.5 = 94.25$ square feet; and the average velocity of the air-current is $47,125 \div 94.25 = 500$ feet per minute. Assuming the anemometer is exposed to the air-current for exactly 1 minute in such a manner as to obtain an average reading, this reading should be 500, making no allowance for the inertia in starting and friction. As proof, $Q = a\,v = 94.25 \times 500 = 47,125$ cubic feet per minute, which is the volume of air passing.

QUES. 6.—(*a*) Which should be the larger, the intake or the return airway, in a mine? Give reasons for your opinion. (*b*) What are the benefits to be derived from splitting the air-current? (*c*) When would you deem it not advisable to split an air-current in a mine? Points 7.

ANS.—(*a*) The return airway should have a larger sectional area than the intake for the reason that the volume of the return air-current is commonly increased by the addition of gases generated in the mine; also, by the natural heat of the mine, and by the decrease of the ventilating pressure as the current moves from the intake opening toward the return. All of these causes act to expand the air passing through the mine, which would increase the velocity of the return current and produce a greater mine resistance, but for the enlargement of the sectional area of the return airway. (*b*) By splitting the current one or more times a larger volume of air is circulated in the mine with less power; purer air is obtained at the working face; the velocity of the current is reduced; and the distribution of the air is made to conform to the needs in the several splits; in other words, the entire circulation is under better control. An explosion of gas or dust is often limited to the split in which it occurred. (*c*) An air-current should not be split when the velocity of the air would thereby be reduced below what is required for efficient ventilation; or when it would be necessary to build an air bridge, and the development of that section of the mine is not sufficient to warrant the expense.

QUES. 7.—Two mines, No. 1 and No. 2, have an equal number of square yards excavated, and employ an equal number of men and mules; all conditions are similar except the height of the coal in the seam. The coal in mine No. 1 is 3 feet thick, while that in mine No. 2 is 8 feet thick. Would the same minimum quantity of air required to properly ventilate No. 1 furnish efficient ventilation for No 2? Explain fully the reasons for your opinion. Points 4.

ANS.—No. 2 mine opened in the thick 8-foot seam would require a larger quantity of air in circulation to produce equally efficient ventilation with that in No. 1 mine in a 3-foot seam; for the reason that its openings, airways, and chambers all have a larger sectional area than those in the thin 3-foot seam and with the same quantity of air in circulation the velocity of the air-current in No. 2 would be much less than that in No. 1, which the question states is only sufficient to properly ventilate that seam. Efficient mine ventilation requires a certain velocity of the air-current, sufficient to sweep away the gases generated, but not sufficient to render the workings uncomfortable.

QUES. 8.—A certain mine is ventilated by means of a blowing fan located at the intake opening. The fan furnishes a volume of air that, under ordinary conditions would be more than ample for the needs of the mine. The air-courses, however, pass through extensive old workings, which contain large quantities of blackdamp that mixes with the air-current. Conditions are such that it is practically impossible to seal off the old workings entirely and prevent the flowing of the air. Can you advise some method by which this mine can be properly ventilated, and avoid carrying the blackdamp to the working faces? Points 5.

ANS.—Alter the fan so as to make it exhaust the air from the mine, thereby reversing the air-current and causing the intake air to traverse the working faces before it reaches the old workings and becomes fouled with the blackdamp. This would necessitate changing the doors throughout the mine so that they will close with the air. The blackdamp would then mix with the return current and be carried by it out of the mine.

QUES. 9.—(*a*) How many stages of underground haulage are there? (*b*) How should each stage be arranged? Points 5.

ANS.—(*a*) In all the larger mines there are two stages in the underground haulage; namely, the first or gathering stage in which the coal is hauled from the working face to the main inside parting where trips are made up to be hauled to the shaft or slope bottom, or out of the mine; this last or main haul constitutes the second stage of the underground haulage. (*b*) The main haul should be of such length that the trips can be hauled only as fast as the coal can be brought from the face to the parting. As the working face is advanced and the distance to the parting or the gathering haul becomes too great to put the coal on the parting as fast as the trips can haul it away, the main haul must be extended by making a new parting or gathering station nearer the face and thus reducing the gathering haul.

QUES. 10.—In selecting a location for a double parting to be used as a gathering point for mules, there are two available

* Questions 1, 3, and 14 were omitted, as to answer them the mine law must be quoted. Every miner should possess a copy of the mine laws of his state and become familiar with them.

sites, one being where the seam dips with a 1-per-cent. grade against the loaded cars. About 300 feet outside of this point the road passes over a slight rise and then has an inclination of 1 per cent. toward the shaft. (*a*) If motor haulage is to be used on the main road, from the parting to the shaft, at which point would you locate the parting? (*b*) Which of these two locations would you choose if mule haulage is to be used for this stage? Explain in detail the reasons for your choice. Points 6.

Ans.—(*a*) If motor haulage is to be used on the main road it would be well to select the inside location first mentioned; for the reason that the motor when pulling the empties into the mine can take the switch at the outby end of the parting and be in readiness to pull out the loaded trip as soon as the empties, running now by gravity, clear the switch. The motor will do the same at the shaft bottom, taking the switch, and allowing the loads to gravitate toward the shaft while it is coupled to the empty trip ready to start back into the mine. (*b*) In the use of mule haulage, it would be difficult for the mules to start the loaded trip against a grade however slight. The outside location should therefore be chosen in this case so as to give an easy start where the grade favors the loaded cars.

Ques. 11.—(*a*) Name the chief factors essential to successful mine haulage. (*b*) What arrangements would you make for the safety of drivers or other persons along haulage roads? Points 7.

Ans.—(*a*) The chief points are a good road bed, sufficiently heavy iron for the traffic, good solid track, ties well ballasted, straight track as far as practicable, well rounded easy curves, light grades, as far as practicable favoring the loaded cars, automatic switches at turnouts, and a good signal system. Keep rolling stock in good condition; use large, self-oiling wheels (not less than 16 inches in diameter), and well-built, tight cars that will not scatter the coal along the roadways. (*b*) Avoid doors on haulage roads wherever possible and never locate a door at the foot of a sharp grade where the driver cannot always have control of the car. Leave a clear space on one side of the track for persons to pass cars without danger; ample room should be provided where it is necessary for drivers to sprag their cars and on each side of doors. Where trappers are employed there should be refuge holes cut in the sides of the entry where the trapper will be protected in case of accident. Refuge holes should also be provided not exceeding 25 yards apart on all haulage roads used as traveling ways, and on slopes, and at slope bottoms.

Ques. 12.—Referring to Ques. 10, suppose the double parting or gathering station is 1 mile from the shaft, and there is a single-track road to the shaft bottom. The mine cars are 7.5 feet in length, out to out, and have a capacity of 2.5 tons of coal. Two 10-ton motors are used to haul the coal from the parting to the shaft bottom, each hauling 25 cars in a trip, which is made in 25 minutes. There are 10 gathering mules, each hauling a single car and requiring an average of 5 minutes to make the trip from the face to the parting. (*a*) What size of iron should be used on the motor road? (*b*) What arrangements should be made for the motors to pass each other between the shaft bottom and the parting? (*c*) What should be the length of the shaft bottom? (*d*) What should be the lengths of the turnouts and the inside parting? (*e*) How many mine cars will be required to keep drivers and motors going and avoid loss of time from having to wait on one another (not counting cars being loaded at the working face)? Points 12.

Fig. 1

Fig. 2

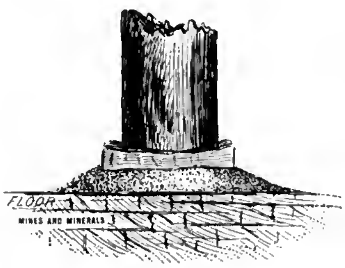


Fig 3

Ans.—(*a*) Use 40-pound rails with fish-plates and well bonded on the motor road. (*b*) A turnout should be made at the meeting point, half way between the parting and the shaft bottom. The switches at each end of the turnout should be automatic, spring-pole switches, so that the switches will always be set for traffic in one direction, but will be operated automatically by cars moving in the opposite direction. One track of the turnout will thus always be used by the loaded and the other by the empty trip in passing to and from the shaft. (*c*) The shaft bottom should be long enough to accommodate two loaded and two empty trips, in case this becomes necessary from some delay either in the hoisting or the haulage of the coal. In this case, the length of the trips hauled is 25 cars, and the shaft bottom should afford standing room for at least 50 cars, or have a clear length of $50 \times 7.5 = 375$ feet. (*d*) The turnout need only be long enough to hold one trip of 25 cars, or say 190 feet of clear track room. To enable the inside drivers and miners to continue work as long as possible, in case of trouble in hoisting, the inside parting should be able to hold two trips, or its length should be the same as that of the shaft bottom; namely, 375 feet of clear track room. (*e*) To avoid as far as possible any delay in hoisting, or stopping work at the face, there should be cars sufficient to form five loaded trips, or $5 \times 25 = 125$ cars, which would stand finally two trips on the bottom, one on the turnout, and two on the parting inside. This would not include the cars in process of loading at the face, which would depend on the number of working places in operation, and a few extra cars undergoing repairs, and in use for delivery of timber, rails, hay, and other daily supplies.

Ques. 13.—Again referring to Ques. 10, an average of 85 kegs of powder are delivered each day to the miners, in their working places, by the company. What arrangements should be made for the safe delivery of this powder? Points 5.

Ans.—This powder should be delivered at the top of the shaft, at a certain specified time each day, and placed in mine cars, lowered down the shaft, and hauled at once to the inside parting, from which point it should be distributed carefully to the men for whom it was intended. The handling and transportation of the powder should be done with great care and no lamp should be brought near the powder; no person but the driver should be allowed to ride on the trip carrying the explosive As each keg is delivered to the miner it should be placed at once in his box and not left at the side of the track.

Ques. 15.—If a prop 5 inches in diameter is required in a 5-foot seam, what should be the diameter of a prop, under the same conditions, in a 7-foot seam? Points 4.

Ans.—For post timber in mines a good rule is to make the diameter of the small end of a post, in inches, equal to its length in feet. Under the same conditions, the diameter of a mine post should be proportioned to its length; or in this case, the diameter of post in the 7-foot seam should be 7 inches.

Ques. 16.—Name the different conditions met with in coal mines in relation to the timbering of rooms, and state how props should be set under each of these conditions. Points 6.

Ans.—There may be a hard roof and floor, or vice versa a soft roof and hard floor, or both roof and floor may be soft. Also, the seam may be level or inclined. A hard-roof-and-floor condition is most destructive of post timber because of the unyielding character of the strata. Under these conditions posts should be provided with thick cap pieces of soft wood, and should not be driven tighter than is required to hold them

in place till they take the weight (Fig. 1); or they are frequently tapered at the foot so as to yield gradually to the weight without destroying the post (Fig. 2); or the post is sometimes set on a slight mound of dirt, for the same purpose (Fig. 3). When the floor is soft a foot-board may be required, the size being adapted to the condition of the floor; or in some cases, it may be sufficient to stand the prop with the large end down. With a weak roof a good cap piece must be used and the post is usually stood with the large end up. If the roof is very frail, or contains slips, planks are often used above the posts to secure greater safety, and the posts must be set closer together and nearer the face. The same precautions must be adopted when both roof and floor are soft. In a practically level seam posts are stood vertically, but when the seam is considerably inclined the post should be (underset) inclined a few inches up the pitch from the perpendicular to the seam.

A NEW SMOKE HELMET FOR MINE-FIRE FIGHTING

A new form of oxygen helmet, known as the Eveready Smoke and Ammonia Helmet, is receiving attention from the mining industry in connection with rescue work. It consists of a hood reinforced by a steel frame which fits snugly over the head and rests on the shoulders, and is made tight around the neck by a steel spring. The wearer breathes into a small mouthpiece and the exhaled air is passed through and over an oxygen-producing compound which utilizes the moisture from the breath and absorbs the carbon dioxide, and liberates oxygen in proportion. The simplicity of the device commends it for many purposes.

Several of these helmets have been used successfully in fighting mine fires, for which they are particularly efficient because the leather does not readily radiate heat. To recharge the apparatus it is only necessary to insert a new cartridge of chemical, and each cartridge, if cleansed after 1 hour's service, will render continued service. The Alabama Fuel and Iron Co., as well as the Pratt Consolidated Coal Co., both of Birmingham, Ala., have recently been equipped with this device, which is made by the Servus Equipment Co., Keenan Building, Pittsburg, Pa.

COAL NOTES

The Harding tipple of the Davis Colliery Co., at Harding, Randolph County, W. Va., which was totally destroyed by fire on September 17, has been rebuilt. The charging of ovens was restarted October 1, and the loading of railroad cars October 8, since which time the tipple has been in full use. Run-of-mine, lump, nut, and slack coal are loaded as before the fire.

On February 17, of the present year, the Legislature of Illinois enacted a law establishing three mine rescue stations and making an appropriation for their complete equipment for the work of rescue following a mine disaster. These stations it has been decided will be at Benton, in the Southern Illinois coal field, at Springfield in the Central coal field, and at La Salle in the northern coal field. Two men will be appointed for each station, one as general manager and the other as superintendent. Eight men have recently passed preliminary examinations for these positions, and have been in training for this work at the government rescue station at the University of Illinios. These men have had unexpected additional practice in a mine fire which occurred during their stay at the University of Illinois.

CATALOGS RECEIVED

In writing for catalogs, please mention Mines and Minerals

Raymond Brothers Impact Pulverizer Co., Chicago, Ill., Catalog No. 10, Grinding, Pulverizing and Separating Machinery, 80 pages.

Mine and Smelter Supply Co., 42 Broadway, New York, N. Y., Bulletin No. 5, The No. 5 Wilfley Concentrator Timber Frame, 19 pages.

Goldschmidt Thermit Co., 90 West Street, New York, N. Y., Thermit Rail Welding, 16 pages.

Chicago Pneumatic Tool Co., Chicago, Ill., Catalog No. 32, Air Tools, 104 pages.

The Cyclone Drill Co., Orrville, Ohio, Catalog A-50, Cyclone Drills, 128 pages.

John A. Roebling's Sons Co., Trenton, N. J., Price List of Galvanized Wire Rope Clips.

Baldwin Locomotive Works, Philadelphia, Pa., Record No. 67, Locomotives for Passenger Service, 32 pages.

Taylor Instrument Cos., Rochester, N. Y., Vol. 1, No. 7 "Tycos"-Rochester, 12 pages.

General Electric Co., Schenectady, N. Y., Bulletin No. 4766, Tantalum Incandescent Lamps, 16 pages; Bulletin No. 4771, Hand-Operated Starting Compensators for Alternating-Current Motors, 12 pages; Bulletin No. 4772, Electric Automobile Appliances, 12 pages; Bulletin No. 4773, Thomson High Torque Induction Test Meter Type, IB-4, 4 pages; Bulletin No. 4774, Centrifugal Air Compressors for Industrial Air Blast and Exhauster Service, 9 pages; Bulletin No. 4776, Engine Type Continuous Current Generators, Forms RB and RBO for Lighting and Power, 8 pages; Index to Bulletins, 10 pages.

Stromberg-Carlson Telephone Mfg. Co., Rochester, N. Y., Folder Describing Stromberg-Carlson Bargain Telephones.

Sullivan Machinery Co., Chicago, Ill., Bulletin No. 58D (Second Edition), The Sullivan Tandem Corliss Air Compressor, Class "WC," 20 pages; Bulletin No. 58F, Sullivian Small Air Compressors, Power-Driven: Classes "WG-3," "WG-4," "WK," "WK-2," "WK-3," 20 pages; Bulletin No. 58G, Sullivan Duplex Air Compressors, Power-Driven: Classes "WJ," "WI," "WN," Steam-Driven: Classes "WE" and "WF," 16 pages.

The Aldrich Pump Department, Allentown, Pa., Pump Data No. 13-A, The Aldrich Vertical Triplex Electric Pump, 24 pages; Pump Data No. 22, 16 pages; Pump Data No. 23. The Aldrich Triplex and Quintuplex Wood or Cement-Lined Electric Mine Pumps, 16 pages; Pump Data No. 24, The Aldrich Vertical Triplex Belt-Driven Slimes Pump, 16 pages; Pump Data No. 25A, The Aldrich Irrigation System, 16 pages; Pump Data No. 26, The Aldrich Vertical Triplex Electric- and Belt-Driven Trade Pumps, 8 pages; Pump Data No. 27, Aldrich Pump Accessories, 20 pages; Pump Data No. 28, The Aldrich Pressure Pumps, 16 pages.

The Bristol Co., Waterbury, Conn., Bulletin No. 126, Bristol's Class II Recording Thermometers, 32 pages; Bulletin No. 128, Bristol's Round Form Class II, 8-inch Recording Thermometers, 4 pages; Bulletin No. 129, Bristol's Thermometer-Thermostats, Classes I, II and III, 8 pages; Bulletin No. 135, Wm. H. Bristol's Recording Shunt Ammeter, 8 pages; Bulletin No. 141, Bristol's Round Form Recording Pressure Gauges, 12 pages; Bulletin No. 145, Bristol's Class II, Indicating Thermometers, 2 pages; Preliminary Bulletin No. 146, Bristol's Long-Distance Recording Tachometer, 2 pages;Bulletin No. 147, The Bristol-Durand Radii Averaging Instrument, 4 pages.

MINE ACCIDENT INVESTIGATIONS

*By George S. Rice**

While it is evident that there is a great diversity of opinion among American mining men as to the proper functions of the new Federal Bureau of Mines, on one point, at least, there appears to be general agreement, namely: That the investigation of accidents in coal mines, started under the jurisdiction of the Geological Survey, should be continued, and a similar investigation should be made of accidents in metal mines

Method of Tabulation of Accidents in Metal and Other Mines Suggested by the Bureau of Mines

A prominent English mining engineer, while speaking of accidents in metal mines, remarked that coal miners were fortunate in having disasters occur in their mines. This, at first, seems like a brutal statement, but when the matter is considered carefully, the meaning becomes clear, since in spite of the large number of men who have been killed and injured in the great colliery disasters that too frequently have appalled the world, the number of deaths and injuries from this cause have formed only a small part of total number killed and injured in coal mining by other causes, particularly those due to falls of coal and rock.

In 1907, the year of the series of greatest American colliery disasters, Mr. E. W. Parker reports 947 men were killed and 347 injured in gas and dust explosions. During the same year in all the coal mines of this country, there were 1,122 killed and 2,141 injured from falls of roof or coal; and from all causes the totals for that year were 3,125 killed and 5,316 injured. Therefore, of the total number killed and injured in that year of greatest mine explosions, less than one-third were killed and one-fifteenth were injured by explosions.

In more normal years like 1908, during which 2,450 were killed and 6,772 injured, only one-sixth of the killed were lost in disasters and one-twentieth of the total were injured by explosions of gas and dust.

Nevertheless, as previously indicated, the horror produced by the explosions has caused attention to be directed to accidents in coal mines, and the precautions taken on this account have led to a general improvement in colliery conditions, and in all civilized mining countries except the United States the accident rate has been steadily decreasing.

In the United States the annual accident rate of coal mining nearly doubled in the 10 years from 1897 to 1907, when we attained the high annual death rate of 4.86 deaths per 1,000 employes. Fortunately, the year 1908 showed a reduction to 3.60 deaths per 1,000, and in 1909 the death rate was still further decreased; we can only hope it will continue to decrease. The present rate is most appalling when one considers it in comparison with that of France or Belgium, where the annual mortality rates are but 1 per 1,000.

In view of the high accident rate in the collieries of this country, it might be questioned if the attention attracted by the disasters has led to much improvement of conditions. Undoubtedly there has been improvement, but it has not, until 1908, kept pace with increased dangers due to rapid development, and greater natural dangers arising from greater depth of workings with consequent increase in flow of explosive gas and the increased use of black powder in lieu of pick work.

Metal-Mining Accidents.—When attention is directed to metalliferous mining accidents, we are surprised to find that many of the states in which mining is done make no record of the deaths and injuries in metal mining, and the Federal Government has hitherto made no attempt to gather such statistics. This strange lack of information cannot be because the metal-mine operators as a rule are less interested in the welfare of their employes; there are many instances to show the contrary.

It is noticeable that the states that do not gather mine accident statistics are chiefly those in which there is no coal mining. Coal miners and operators, in going from the old-established coal-mining centers of the East and from Europe have carried with them their customs and systems of law and regulations, which include the recording of mine accidents.

Metalliferous miners in this country have largely formed their own precedents, and, probably due to the wide diversity of mining systems, in turn due to great differences in natural conditions, there has not been the same community of interests among metalliferous miners until more recently. Your committee on uniform mining laws and prevention of mine accidents in its report to this Congress, has clearly brought to attention the lack of metal-mining statistics, except in certain states, and those states have the best mining laws.

Mr. Frederick L. Hoffman, the well-known statistician, has done valuable service to the metal-mining industry in attempting to obtain, in spite of lack of official statistics, the fatality rates in different sections of the country. As his report, published in the *Engineering and Mining Journal* of March 5, 1910, has been republished by the committee on uniform laws, I will not attempt to comment on it, except to call attention to the fact that he finds the average rate is practically the same as in coal mining, despite the general acceptance that the natural conditions are less hazardous. His figures also show that there is a greater variation in different metal-mining districts than in different coal-mining districts, and that the accident rates in both are appallingly high as compared with those of European countries.

Attacking another phase of the conditions under which miners work, Mr. Hoffman in an article in the *Engineering and Mining Journal*, July 2, 1910, has commented upon the excess of sickness and ill health of miners as a class, compared with those who work in the open air.

Haldane and Thomas Paper on Miners' Diseases.—Messrs. J. S. Haldane and R. A. Thomas, in an article which appeared in the Transactions of the Institute of Mining and Metallurgy, Volume 13, reached the following conclusions after a study of the health statistics of miners and others living in England, Wales, and Cornwall:

"1. Apart from lung diseases and a slight excess of fatal accidents (many of which are not underground), metalliferous mining in England is a very healthful occupation.

"2. After the age of 25, there is an extraordinary contrast as regards the death rate from lung diseases between colliers and ironstone miners on the one hand and metalliferous miners on the other. Since 1892, the death rate from lung diseases has so increased that now the deaths due to lung diseases in the metalliferous mines are 8 to 10 times as great as that of colliers or ironstone miners of the same age."

The authors further state "the variety of lung diseases which is chiefly responsible is returned as phthisis on the death certificate. Among older men deaths from bronchitis are in excess. * * * the following causes have been attributed in the past as wholly or in part the cause of miners' phthisis:

"1. Absence of sunlight. This cause dismissed as having nothing to do with the disease.

"2. Exposure to high temperature and sudden variations in temperature. This cannot be an important cause as coal miners are similarly exposed.

"3. Gaseous impurities. Surprisingly little attention is given to the question of ventilating metal mines as compared with coal mines. Consequently, lung diseases in metal mines have been largely attributed to poor ventilation."

Messrs. Haldane and Thomas did not agree with the foregoing suggested causes as they found that the mines that they examined in Cornwall were freer from gaseous impurities than the air of coal or ironstone mines. They discussed three possible causes:

1. The stone dust produced.
2. The smoke produced by explosives.
3. Infectious bacteria.

* Paper presented before the American Mining Congress, Los Angeles September, 1910.

They decided that infection by tubercular bacillus will not explain the facts relating to miners' phthisis. They considered that the smoke from explosives and candles was "probably inappreciable." The final conclusion of Messrs. Haldane and Thomas was that the inhalation of the stone dust was the cause of the lung disease which is common among the Cornish and other miners. They state that out of 142 men who had worked rock drills in the Cornwall district, 94 per cent. died of lung diseases and the average age at death was 37 years, and of 178 men who did not work with rock drills, 65 per cent. died of lung diseases, but the average at death was 50 years.

They, therefore, considered that the introduction of rock drills in Cornwall is responsible for the recent great increase in the mortality of Cornish miners between the ages of 25 and 50 years. They thought that there was no reason for believing that the dust produced its effects upon the lungs in any other way than mechanically by its hardness, sharpness, and insolubility, as there seemed to be little difference in effect whether the dust was from quartz, granite, flint, gannister, or hard sandstone.

In a discussion which followed the Haldane and Thomas paper, Doctor Oliver stated "that miners' phthisis belonged to a group of diseases in which the spongy texture of the lungs became converted into a hard solid tissue."

Mr. R. J. Watkins, who had charge of a sampling mill, stated that when certain kinds of dust were sampled, men were knocked out often for many days, and that a certain sulphide ore of certain composition gave the most trouble.

South Africa.—Much attention has been paid in South Africa to the miners' phthisis, and in 1902 prizes were offered for "the three best practical suggestions and devices for obviating, minimizing, or combating the causes leading to miners' phthisis." The advertisement offering these prizes stated that while no definite information was before the Commission, it was generally assumed that the cause of the disease lay in the inhalation by the men of fine dust produced by rock drills.

Two hundred and twenty-nine competitors responded. Most of the suggestions were regarded as impracticable methods for wetting, through jets, the dust as made by the drill, and atomizers or sprinklers for laying the dust when made, received most attention, and some of the devices were regarded by the Commission as promising.

Gases From Blasts.—In addition to the menace of stone dust and despite the statements in the foregoing quotations, it is evident that in some mines and in some places, where there is no positive, or little, ventilation, the breathing of gases from the combustion of explosives produces an unfortunate effect upon the men, which at times has been accumulative in its effects. We know from the work which has been done by Haldane and others the poisonous effects of minute quantities of carbon monoxide which is absorbed by the blood in lieu of oxygen, and if a man is continuously exposed he rapidly succumbs. This is the case when there is over .3 of 1 per cent. in the atmosphere. Moreover, recovery of one partly poisoned is slow. It seems probable that when a man is regularly exposed to small quantities in the air, he may become in time more or less incapacitated.

With nearly all high or quick explosives on the market, carbon monoxide forms a large portion of the gases of combustion. From time to time we hear of men being overcome in the mines by such gases and in some instances even being killed—notably the case that occurred in the Gunnison tunnel, in Colorado, on January 16, 1910, when a shift of men while waiting for the smoke from a round of 50 holes to clear, were overcome and nine of them died, evidently due, according to the facts as reported, to monoxide poisoning.

Investigation of Mine Accidents by Bureau of Mines.—Under the terms of enactment of the Bureau of Mines, the investigation of coal-mine accidents, started under the jurisdiction of the United States Geological Survey, will be continued by the Bureau of Mines. This investigation consists of testing of explosives for use in gaseous or dusty mines (those that pass being listed as "permissible explosives"), the investigation of safety lamps and of rescue apparatus of all kinds; the establishment of rescue stations for the education and training of miners in the use of rescue apparatus; the investigation of mine disasters by mining engineers, with a view to finding preventives; the study of the humidity of mine air in its effect upon the drying or moistening of coal dust; the collection and analysis of mine gases and study of their sources and changes of efflux from the strata; the study by electrical engineers of the different phases of mine installations with a view to lessening the dangers therefrom, and the collection by the mining engineers of samples of coal throughout the country, which, while partly for governmental fuel supplies, is also of service in connection with mine accident work, through the relation of the coal to the production of dust and gas.

The foregoing work has in the past been carried on by a well-equipped laboratory at Pittsburg, aided by a small field force of mining engineers. Much time has been occupied in purely educational work among the miners and foremen. It is proposed to continue all this work and possibly to enlarge slightly from time to time as funds may be provided by Congress, to better cover the widespread coal fields of the country.

Under the law establishing the Bureau of Mines, the scope of the work has been enlarged, and the investigation of mine accidents has been extended to cover both quarries and metalliferous mines. It is expected that mining engineers who have been already connected with the mine-accident investigations on coal mines will extend their observations so far as possible to all kinds of mining and quarrying.

The appropriation for this year will allow but few additions to the force, but if the results justify, it is not improbable that Congress will provide for extension of the work in the future.

It is manifest that there is ample room to do good work in the investigation of the metalliferous mine accidents, particularly in studying in the field the effects of stone dust and of gases arising from the discharge of explosives and, at the Pittsburg station, in the study of the explosives used in metal mining with a view to obtaining such explosives that the products of combustion will be less harmful to the health of the miners.

In another direction, in quarries and in metalliferous mines, the gathering of statistics of accidents by a Federal Bureau, working in conjunction with state bureaus, a work hitherto ably done by Mr. E. W. Parker for coal-mine accidents, will be of great value, inasmuch as it will lead to precautions being taken to prevent many accidents, the causes of which are often overlooked. Unification of the methods of gathering such mine-accident statistics will also be of value, inasmuch as it will enable the intelligent comparison of similar kinds of accidents in different districts and countries and thus lead to betterment of conditions.

Beside this valuable feature there is another. It is the tendency of the times to look forward to a time when each industry will bear the load of caring for its killed and injured. Already a number of progressive mining companies have started insurance organizations, the cost of which is borne partly by the employes and partly by the companies. In any case, there can be little question that if an industry is in a thoroughly healthy condition, it should be able to bear the cost of caring for those injured in its business, and for the pensioning of needy widows and orphans, to a greater or less extent. As a matter of common humanity, this proposition can hardly be disputed. If prevailing conditions prevent, it means that the price obtained for the product should be raised to meet these conditions. No doubt this movement must be gradual to not disarrange business affairs.

Accompanying this paper, there are forms giving suggested classifications of accidents in metal mines and coal mines. These classifications are not in any sense final, but are put

forward for discussion. They follow to some extent the English system, though a little more elaborate. It is only by close subdivision that analysis of the causes of accidents can be made satisfactorily.

In another matter the Bureau of Mines may be able to give valuable aid, namely: In promoting the unification of mining laws and regulations, such as those proposed by the several committees reporting to this Congress.

There are doubtless many problems in connection with mine accidents which are not mentioned in this paper, and of which members of the Bureau of Mines staff will be glad to learn from the mining engineers of the country. While the question is complex, it is worthy of our best efforts since it deals with the lessening of loss of life and limb.

SUGGESTED FORM FOR TABULATION OF METALLIC MINE ACCIDENTS

I. Underground Accidents

(*a*) *Falls of Roof, Wall, or Timber:*
- 1. Fall of roof or hanging wall in headings.
- 2. Fall of ore or wall in stope or winze.
- 3. Slippage or breakage of timbers.
- 4. Run of ore in chutes.

(*b*) *Miscellaneous Underground:*
- 1. Explosives:
  - (*a*) Handling loose explosives.
  - (*b*) Thawing.
  - (*c*) Premature fires.
  - (*d*) Digging out misfires.
  - (*e*) Flying pieces from blast.
- 2. Suffocation:
  - (*a*) By natural gases.
  - (*b*) By gases from fires.
  - (*c*) By gases from blasts.
- 3. Irruptions of water.
- 4. Haulage.
  - (*a*) Breakage of rope.
  - (*b*) Run over or crushed by cars.
- 5. Electric shocks.
  - (*a*) Trolley.
    - 1. Below 300 volts. 2. Above 300 volts.
  - (*b*) Power cable.
    - 1. Below 300 volts.
    - 2. Above 300 volts.
  - (*c*) Switches.
  - (*d*) Electric machines or pumps.
- 6. Man falling down: (*a*) winze. (*b*) chute. (*c*) stope. (*d*) manway.
- 7. Drilling accidents (from machine or drills).
- 8. Miscellaneous.

II. Shaft Accidents

- 1. Overwinding.
- 2. Connections or ropes breaking.
- 3. While ascending or descending shaft by machinery.
- 4. Falling into shaft.
  - (*a*) From surface.
  - (*b*) From part way down.
- 5. By things falling down shaft:
  - (*a*) From surface.
  - (*b*) From part way down.
- 6. Miscellaneous shaft accidents.

III. Surface

- 1. Head-house accidents.
- 2. Outside tramway.
- 3. Power-house accidents.
  - (*a*) Boiler and engine. (*b*) Electric shocks.
- 4. Miscellaneous surface accidents.

Note.—It is proposed gathering the statistics for the number of accidents and number of men killed and injured by accidents, tabulating each of the three classes separately.

SUGGESTED FORM FOR TABULATION OF NON-METALLIC MINE ACCIDENTS

I. Underground Accidents

(*a*) *Explosions:*
- 1. Coal Dust.
- 2. Firedamp.
- 3. Coal dust or firedamp (cause undertermined).

(*b*) *Falls of Ground:*
- 1. Fall of roof at face.
- 2. Fall of roof in drawing pillars.
- 3. Fall of coal at face.
- 4. Fall of roof in roadways.

(*c*) *Miscellaneous Underground:*
- 1. Explosives:
  - (*a*) Handling loose explosives.
  - (*b*) Thawing.
  - (*c*) Premature firing.
  - (*d*) Digging out misfires.
  - (*e*) Flying pieces of rock or coal from shots.
- 2. Suffocation.
  - (*a*) By natural gases.
  - (*b*) By gases from fires.
  - (*c*) By gases from blasts.
- 3. Irruptions from water.
- 4. Haulage.
  - (*a*) Rope breaking.
  - (*b*) Run over or crushed by cars or locomotive.
- 5. Electric shocks from:
  - (*a*) Trolley wire.
    - 1. Below 300 volts. 2. Above 300 volts.
  - (*b*) Fixed power cable:
    - 1. Below 300 volts.
    - 2. Above 300 volts.
  - (*c*) Switches.
  - (*d*) Machines or trailing cables.
- 6. Man falling down:
  - (*a*) Winze.
  - (*b*) Chute.
  - (*c*) Stope.
  - (*d*) Manway.
- 7. Drilling or mining machine accidents.
- 8. Miscellaneous.

II. Shaft Accidents

- 1. Overwinding.
- 2. Rope or connections breaking.
- 3. While ascending or descending on cage.
- 4. Falling into shaft.
- 5. By things falling into shaft.
- 6. Miscellaneous shaft accidents.

III. Surface Accidents

- 1. Tipple or tower accidents.
- 2. Outside inclined-plane or tramway.
- 3. Power-house accidents:
  - (*a*) Engine or boiler accidents.
- 4. Electric shocks.
- 5. Railroad accidents (incident to loading and switching).
- 6. Washery and coke-oven accidents.

Note.—It is proposed gathering the statistics for the number of accidents and number of men killed and injured by accidents, tabulating each of the three classes separately.

According to Professor Phillips of the University of Texas, 10,767,866 tons of coal and 5,488,218 tons of lignite have been mined in Texas since 1884. There are 8,000,000,000 tons of lignite and 23,000,000,000 tons of coal remaining in an area of 68,500 square miles,

TURBOBLOWERS AND COMPRESSORS *

One of the newest uses to which steam turbines have been successfully put is the driving of turboblowers and turbocompressors.

Construction and Operation of Air Compressors on the Turbine Principle for High Pressures

The turboblower and turbocompressor, is a suitable machine for exhausting or compressing large volumes of air or gas. It is suitable for ventilating mines, for supplying the blast to cupolas and coke ovens, and for respectively supplying compressed air or gas to, or for exhausting air or gas from, the various apparatuses and processes used in chemical works and similar undertakings.

Although only a short time ago it was usually considered to be impossible to make a commercially efficient and serviceable blower for any but the lowest pressures, it is claimed that even at pressures as high as 150 pounds per square inch the new type of turbocompressor is at the very least equal in thermal and mechanical efficiency to the best kind of reciprocating air compressor, whilst from a commercial point of view, the turboblowers or compressors are often preferable on account of advantages attainable through their introduction. The following are some of these advantages:

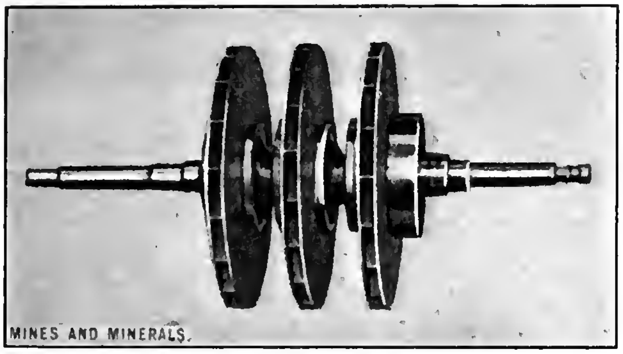


Fig. 1. Impellers for Turboblower

1. The turboblowers and compressors deliver a steady (non-pulsating) current of air or gas.
2. They run practically without noise at all loads.
3. They are simple in construction, have all parts easily accessible, and are reliable in action.
4. They not only require very little attendance and lubrication when at work, but relatively speaking, a minimum of space for their accommodation.
5. They require a minimum of power to drive; they are also easily governed through a wide range of variation of speed, without materially affecting their economy.
6. They are perfectly balanced both dynamically and in respect of axial thrust.

As already stated, turboblowers are adopted for a variety of purposes. It is possible they are preferable to reciprocating engines in all cases where the volume of the air or gas to be handled is sufficiently great, quite irrespective of whether they are required to produce a rise of pressure or a partial vacuum. The turbo compressor, in a like manner, is suitable for all pressures up to 120 pounds per square inch, if the volume of gas to be handled is sufficiently large. Both the blowers and compressors are suited for direct coupling to steam turbines, but they may equally well be driven by electromotors of suitable size and speed.

The impeller shown in Fig. 1 is similar in design and detail, whether intended for a blower or for a compressor. To avoid the high stresses to which it is subjected the impeller is built up of those materials which are most suitable for the various parts of the wheel. Thus, the bosses are steel castings, whilst the cheeks are of special steel plates. The latter, as shown in Fig. 2, are conical in cross-section, and attached to the boss in such a manner as to allow free expansion of both cheeks under

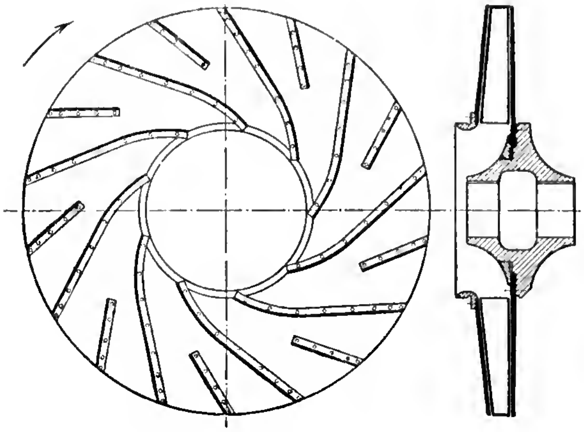


Fig. 2. Sectional View of Impeller

the influence of centrifugal force. As a result of this construction, the impeller runs true and steady at all speeds, and permits of the running clearances being reduced to a minimum, with the result that internal circulating losses are small.

The housings of these turboblowers and compressors are respectively designed to suit the requirements of the two types of machine. In both cases the housing is fixed at one (the driving) end only, and is free to move with variations of temperature at the other.

In the turboblower shown in Fig. 3 the inlet cover is provided with substantial feet solidly connected to the bedplate. The pressure end and intermediate parts of the casing overhang the bedplate, but are well connected to the inlet cover by

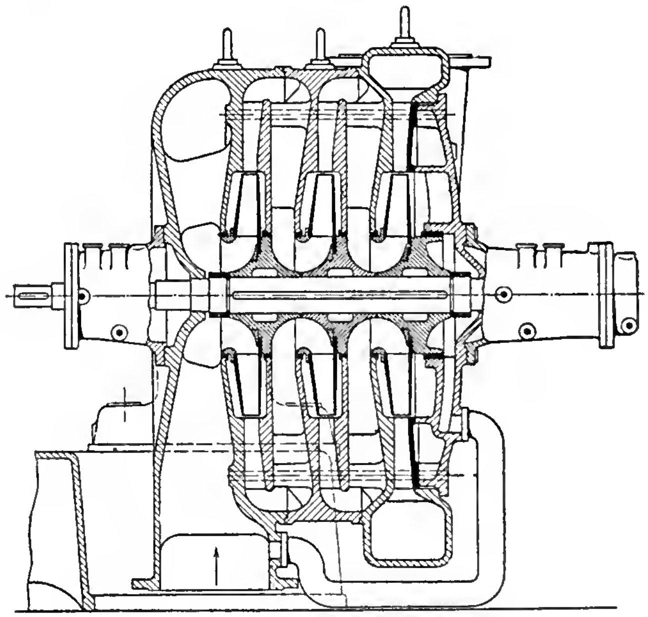


Fig. 3. Section of Turboblower

vertical, cylindrical, spigot, and faucet joints. The general design is such that all parts of the blower are easily reached for examination or adjustment.

In the case of the turbocompressors, the same general principles of design are followed. As, however, a compressor is usually of considerably greater length than a blower, the two

* *Iron and Coal Trade Review.*

bearings of the former, situated one at each end of the machine, are designed to support the casing as well as the rotor at both ends. Here, too, on account of the greater length of the unit, although the vertical divisions of the casing are maintained, a further horizontal division is introduced, as shown in Fig. 4, so that, after removing the upper half of the housing, each impeller and its corresponding part of the casing are freely accessible for inspection.

The ease with which air may be cooled whilst being compressed in a turbocompressor is to no small extent answerable

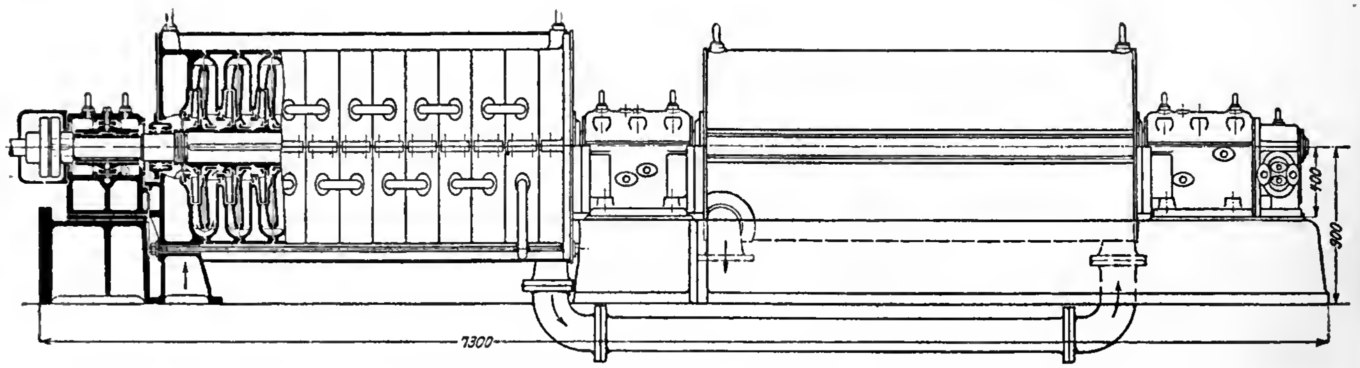


Fig. 4. Sectional Elevation of Two-Cylinder Turbocompressor

for the good efficiency of the unit. When reciprocating engines are employed, the effect of water-jackets decreases as the cylinder volume increases; for which reason, if low final temperatures are to be obtained, multistage compressors must be resorted to and intermediate coolers employed. In the case of turbo machinery, the form of the housing and the condition in which the air or gas passes through the machine are both favorable to the efficient use of water-jackets. The rise of temperature can therefore be reduced to a minimum.

The easy and economical method of regulating the delivery and pressure of these blowers compares favorably with the methods—such as by-pass mains and variable clearance volumes, etc.—necessary with some types of blowing machinery. Delivery volume, pressure, and rotary speed vary, as is the case with centrifugal pumps, in fixed proportion to one another; consequently, delivery volume and pressure may be changed by simply varying the rotary speed. At constant speed, volume and pressure may be varied by adjusting the regulating valve usually situated in the suction main. By this means the delivery volume may be varied between its maximum value and zero, the power consumed falling with the reduction of volume, while the efficiency, as shown in the curve, Fig. 5, is but slightly affected, even by comparatively large changes of volume.

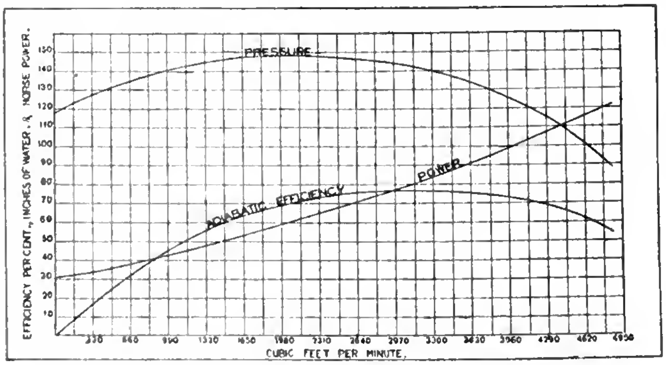


Fig. 5. Pressure, Efficiency, and Power Curves of Turboblower

If turbocompressors are directly coupled to steam turbines, a pressure-regulating device can be made to govern the speed of the turbine in such a way as to maintain a constant air or gas pressure in spite of variations of delivered volume. The high efficiency already obtained (78 per cent.) as referred to adiabatic compression, places these turboblowers very favorably in comparison with the best reciprocating blowing engines. Within the past three years electricity has been applied to air compressors attached to coal cutters and its adoption would be in line as power for multistage blowers.

SOCIETY MEETINGS

About 120 members of the American Institute of Mining Engineers left New York on the steamer Prinz August Wilhelm, on October 21. The itinerary of the canal zone meeting includes Havana, Cuba; Kingston, Jamaica; Colon, Panama; Port Limon, Costa Rica; Colon, Kingston, and Fortune Island; arriving in New York, Tuesday, November 15. An account of the trip will appear in the January issue of Mines and Minerals.

The West Virginia Coal Mining Institute will hold its next meeting at Wheeling, W. Va., December 6, 7, and 8. An interesting and instructive program has been arranged, which with the entertainment features will make this meeting one to be remembered.

The 1911 annual meeting of the Canadian Mining Institute will be held on Wednesday, March 1. Members contemplating contributing papers to the proceedings will greatly oblige the secretary by informing him as soon as possible. It is desirable that the manuscript of all papers should be in the hands of the secretary not later than the 15th of January next, to admit of the issuance of advance proofs for discussion purposes. If manuscript is received later than the above date, its publication in advance of the meeting cannot be guaranteed.

Coal Mining Institute of America will hold the next meeting in the Engineering Building of the Carnegie Technical Schools, Pittsburg, December 15 and 16. Membership in this institute is open to all mining men, and all members receive the proceedings each year in a bound volume of about 400 pages. Membership $2. Bound volume to non-members $2. Charles L. Fay, secretary, Wilkes-Barre, Pa.

Coal Mining Institute of West Virginia will hold the next meeting in Wheeling, W. Va. Membership in this institute is not confined to West Virginia. The proceedings are published annually and can be purchased from Edward B. Day, secretary, 108 Smithfield Street, Pittsburg, Pa. The annual membership fee is $2.

Consul Frederic W. Goding writes from Montevideo, Uruguay, that arrangements are being perfected to develop the manganese deposits in and around Zapucay, Department of Rivera, which consist of two large mountains of that mineral, one of which, according to British geologists, is among the richest known, containing 150,000,000 tons of first-grade manganese. As the current price is $15 per ton, the mineral contained in the measured area represents $2,250,000,000.

FALCONE OVEN TILE-LAYING MACHINE

Written for Mines and Minerals

In the October issue of MINES AND MINERALS there was a brief article by Bert Lloyd on "The New Tile-Laying Machine," which was introduced at the Cokedale plant of the Carbon Coal and Coke Co., in Colorado. The tool, which was invented by Michele Falcone, of Cokedale, is strictly an expense reducer in laying tile where beehive or Welsh coke ovens are in operation.

Operating Data From the Cokedale Plant, Las Animas County, Colorado

The cost data given below relative to this machine are authentic in every respect and will be of special interest to all who are connected with or interested in the operation of coke oven plants. Frederick P. Bayles, E. M., superintendent of the Cokedale plant has kindly furnished the data relative to this machine which has been in use since July, 1909, during which time 140 hot ovens have received new bottoms without the loss of a single charge of coke from the oven cooling. Information concerning these ovens is that they are of the standard 13 feet diameter, 7½ feet high, beehive type with one hundred and thirty-five 4″×12″×12″ tile per oven, therefore the tool has replaced 18,900 tile since its introduction, and withdrawn nearly as many more. At the Cokedale plant the ovens are pulled by a machine, similar to those shown in Fig. 1, and are charged with 11,466 pounds of washed slack containing 9.4 per cent. moisture, but whose dry weight is 10,388 pounds (5.294 tons) and whose coke yield is 7,593 pounds (3.797 tons) or 73.09 per cent.

FIG. 1

The Cokedale ovens are of special interest because of the ring and wharf walls being of concrete, while the oven walls, Fig. 2, and the piers, Fig. 3, are of reinforced concrete. It is believed that the remarkable yield of coke, 75 per cent. of the charge, is due in a measure to the oven construction, which prevents the entrance of excess air, and also to the close regulation and control of the oven draft.

The ovens are pulled from 1 A. M. until about 10 A. M., and as the day shift, who repair oven bottoms, do not come on until 7 A. M., there is a delay which is an operating convenience and not a necessity. After pulling the coke, and before cleaning out the old bottom tile previous to repairing, there is an average delay of 3.60 hours. From the time the coke is pulled until the oven is charged after repairing, the average is about 12.23 hours, divided as follows:

	Hours
Before cleaning out the old bottom	3.60
Cleaning out the old bottom	2.01
Placing and leveling sand	.77
Laying new tile by machine	1.15
Delay after repairs	4.70
Total	12.23

The delays before and after repairs are due to normal working conditions and could be eliminated were the saving in so doing of sufficient moment. The time necessary for the charge to catch on fire, after an average delay of 12.23 hours, using an average charge of 11,466 pounds of washed slack containing 9.4 per cent. of moisture is 1.60 hours.

TABLE 1

Leveling Sand		
1 machine man at 30 cents for .77 hour	$.2310	
1 helper at 17½ cents for .77 hour	.1347	
	$.3657	$.3657
Laying Floor		
1 machine man at 30 cents for 1.15 hours	$.3450	
1 helper at 17½ cents for 1.15 hours	.2012	
	$.5462	$.5462
Total cost of machine laid floors		$.9119

In the year 1909, 37,862 ovens were pulled at Cokedale, producing a total of 190,093 tons of coke, or an average of 5.02 tons coke per oven pulled. The loss in coke yield due to placing a new bottom with this tile machine was 1.223 tons or 24.5 per cent. of the normal yield, which is practically $\frac{1}{30}$ of the loss which occurs when laying tile by hand.

The total loss in the production of coke from the delays caused by the repairing or relaying of 140 new floors amounted to 171.2 tons, which in the production of 466,254 tons is .0003 per cent. The cost of removing the old floors, hauling and placing sand in the oven, would be practically the same with machine or hand practice. See Table 1

To appreciate the difference in saving in time and money it is to be borne in mind that with hand-laid floors the oven must be cooled down sufficiently to permit a man to enter; this calls for a shut-down of from 10 to 12 days before repairs can be made and the oven again put in coke.

On account of the heat in the oven it will take a brick mason and helper one-half day and often longer to relay a floor by hand,

FIG. 2

exclusive of the time necessary to remove the old floor and place sand.

1 mason at 30 cents, 5 hours	$1.500
1 helper at 17½ cents, 5 hours	.875
Total	$2.375

Before comparison between machine and hand work is made the cost of reheating the ovens after repairing by hand is to be considered, and this Mr. Bayles gives as follows:

½ cord wood, $4 per cord	$2.00
Hauling, splitting, and placing in oven	1.50
3 tons coal, $1 per ton	3.00
Stirring, building, and tending doors, pulling and disposal of sweat coke	3.00
Total	$9.50

This makes the total cost of hand work $11.88 as against 91 cents by machine.

With machine-drawn ovens the bottoms should be replaced once every year and with hand-drawn ovens once every 2 years, if the ovens are to be kept as they should be.

The loss in production of coke while the oven is out and from faulty and inferior coke from the first and second charge will bring the operating cost to very close to $15 per oven per annum for machine-pulled ovens, and about $7.50 per oven per annum for hand-pulled ovens, against a cost of $1 per oven per annum for machine-repaired machine-pulled ovens, and 50 cents per annum for machine-repaired hand-pulled ovens.

Fig. 3

DETERMINING VERTICAL DEPTHS IN DRILL HOLES

In the construction of the new aqueduct of the city of New York, which crosses the Hudson River just north of West Point, it became necessary to determine accurately the vertical depths of the narrow diamond-drill bore holes which are being sunk under the Hudson River at various angles and which have already reached depths of 1,500 feet.

This problem has been solved by Prof. G. N. Lewis and Prof. H. T. Kalmus, of the Massachusetts Institute of Technology, who have devised a self-recording pressure gauge which has a diameter of less than ¾-inch and can therefore be placed directly in the terminal section of the drill rod. Since the bore holes are constantly filled with water, the maximum hydrostatic pressure recorded is a direct measure of the vertical depth. It seems probable that this instrument may be of service in other engineering and mining work where it is desirable to determine the vertical depth of bore holes used in prospecting.

The pressure gauge proper consists of a very thin strip of tempered steel bent into the form of a hollow rectangular tube approximately ⅝-inch wide, $\frac{1}{32}$-inch thick, and 18 inches long. The edges and the lower end of the tube are welded together with the oxyacetylene flame. Into the upper end is welded a small steel tube which in turn is sealed to a straight glass tube, the upper end of which enters an air-tight chamber. The gauge and connecting glass tube are filled with mercury and when subjected to an external pressure the thin steel walls of the gauge undergo considerable temporary deformation, thus diminishing the total volume of the gauge and forcing the mercury from the upper end of the glass tube into the surrounding chamber. When the pressure is released the gauge resumes its original volume and the mercury level in the glass tube falls through a distance which measures directly the pressure to which the apparatus has been subjected. The mercury may now be returned to the glass tube from the surrounding chamber by a simple device.

The instrument gives remarkably reproducible results and is able to record vertical depths up to 1,500 feet with an error of no more than 2 or 3 feet. On account of the extremely small volume of the thin steel gauge, the effect of temperature changes, even amounting to 30 or 40 degrees, is entirely negligible.

THE INNOKO DISTRICT, ALASKA

The discovery of gold placers on Otter Creek, a northern tributary of Haiditarod River, in the Innoko Valley, caused a movement of population in 1909 which promises to be important. Thousands of prospectors and miners flocked to this district from all parts of Alaska, as well as from points outside the territory. Although the district may not support the large population it has acquired, it seems to offer a promising field for exploitation. The output of the year in the Innoko is variously estimated at $300,000 to $400,000 in gold. A government survey of the district is now under way.

ELECTRICALLY DRIVEN PUMPS

The electrically driven three-stage centrifugal pumps, shown in Fig. 1, are of particular interest to coal miners where electricity is in use for driving machinery.

One of the most up-to-date plants of this kind is at the Eureka mines of the Berwind-White Coal Co., at Windber, Pa.,* where there is a central pumping station which drains a large area. The Windber plant pumps 4,000 gallons of water per minute against 330-foot head.

Another centrifugal stage pump is installed in the Keyser Valley central pumping station near Scranton, Pa. Each unit of the pumps shown in Fig. 1 comprises a Westinghouse induction motor designed for 500 horsepower at 750 revolutions per minute, with three phase-current at 400 volts direct-connected to the pump. There are three of these units although but two are shown in the illustration, and after being installed at the pumping station for which they were designed they were tested.

Each unit was run continuously for 15 hours, during which time readings of the electrical input were taken every 5 minutes; readings of the Venturi water meter every 10 minutes; pressure gauge every 5 minutes, and revolutions of pump every 15 minutes. The results of one test will answer for all.

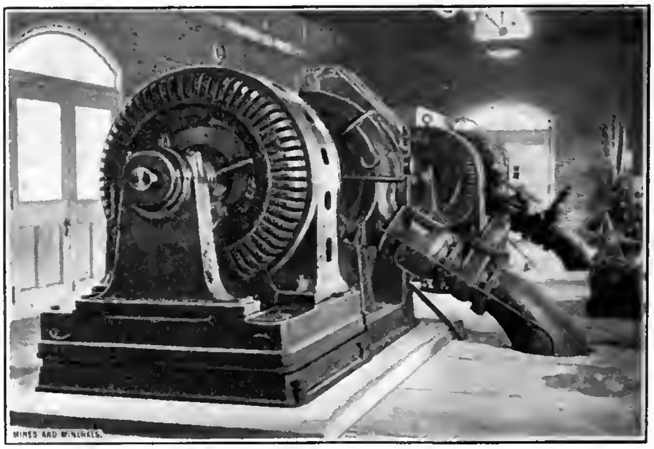


Fig. 1. Electrically Driven Pumps, Keyser Valley Plant

Maximum revolutions per minute	746
Minimum revolutions per minute	738
Average revolutions per minute	743
Maximum gauge pressure, pounds	143.5
Minimum gauge pressure, pounds	119.5
Average gauge pressure, pounds	136.9
Average head, feet	315.8
Gallons pumped per hour	238,000
Gallons pumped in 24 hours at this rate	5,712,000
Kilowatt input per hour	343.4
Gallons per kilowatt hour	1,454
Hydraulic horsepower	318
Electric horsepower	456.6
Efficiency per cent	69.6
Duty, foot-pounds per 1,000 kilowatts	30,800,000
Degrees temperature rise in 15 hours	41

* Mines and Minerals, Vol. XXX, page 457.

PROSPECTING FOR SILVER

Silver, once called a precious metal, is now an ordinary commercial substance, whose intrinsic value is so low, about 20 cents per ounce, that it it not sought so eagerly as formerly.

Tests By Which It May Be Identified Methods of Occurrence and How Deposited

It would be folly on the part of the prospector to pass it by simply because it is so common a substance, for there are possibilities that it may be in large quantities or associated with other metals which will enhance its value. While silver is found occasionally in metallic form, it is more frequently found associated with other metal minerals, for instance, copper, lead, and gold. When native it resembles the silver in coin, although possibly a little brighter or darker in color. It can readily be distinguished in this form, but if there is doubt it can be melted with a blowpipe on charcoal, or if that useful little instrument is not available it will dissolve in nitric acid. If a little salt be added to a silver nitrate solution white silver chloride will be precipitated. If a few drops of hydrochloric acid be added to a silver nitrate solution silver chloride will be precipitated, which is soluble in ammonia, while lead chloride is not. Silver chloride is insoluble in hot water, while lead chloride is soluble, but will precipitate on the solution becoming cold.

With these few simple tests the novice can distinguish lead from silver, although frequently the two are associated in nature; in fact it is a rather rare occurrence to find lead sulphide free from silver.

The greater part of the silver produced in the United States is obtained when smelting lead and copper ores.

The greater part of the silver produced in Mexico comes from lead-silver ores. These ores also carry some gold, which oftentimes is sufficient to pay for their treatment, although not in such quantity that they could be classed as gold ores.

The greater part of the silver produced in Ontario, Canada, is almost native, although associated with cobalt and nickel minerals.

The first large silver deposit found in this country was the celebrated Comstock lode, in Nevada. The black sulphide of silver was at first thrown away by those who were washing for gold, and even broken up to obtain the free gold in the lumps. As soon, however, as an assay disclosed its nature mining commenced in earnest. The gangue in this deposit was mostly quartz and the vein has been worked to a depth of 3,100 feet, making it evident that the solutions forming the vein came from great depths. Argentite, or silver sulphide, contains 87.1 per cent. of silver and 12.9 per cent. of sulphur. It occurs in cubes and octahedrons, massive; as scales, coatings, and tree-like markings. It has a blackish lead-gray color. Its streak on porcelain has a lead-gray color and a shining metallic luster. It is partly soluble in nitric acid, and when in solution will silverplate a copper strip. The mineral is as readily cut as lead; will flatten under the hammer, and is easily melted.

Other silver deposits were found in Nevada, although they were not remarkable, and it was not until 1878 that any bonanza silver deposits attracted attention. These were at Leadville, Colo., where the lead sulphides and carbonates were found in imestone adjacent to porphyry. In places very rich silver ore was found and often associated with it was gold. Closely following this were the Butte, Mont., deposits, which were in granite that had been cut by intrusive dikes, forming fissures that were afterwards filled by solutions carrying copper sulphides, gold, and silver. The copper sulphides were weathered, leaving the gold and silver in great masses near the surface. The gangue in this instance is quartz, indicating that the mineral solutions came from below.

Following this discovery came the Mollie Gibson and Aspen silver deposits in Colorado. These mines were in limestone adjacent to porphyry, and like the Leadville mines, were contact deposits. Probably the next most noted silver deposit was found by a burro in the Couer d'Alene district at Wardner, Idaho. The silver-bearing galena seems to be as rich in the Bunker Hill and Sullivan Mine at a depth of 2,100 feet, as at the surface. The mine is said to have paid over $7,000,000 in dividends since its discovery in 1885. It is at present the richest silver mine in the United States, and other silver-lead deposits are being discovered in the same district almost every year. The deposits are in a mineralized zone, having quartzite foot wall and an impregnated hanging wall of brecciated quartzite. The rocks in this section are quartzite and schists that have been folded and otherwise distorted.

In the Cobalt district in Canada there are some exceedingly rich silver veins in diabase. The outcrop of these veins is colored pink by cobalt bloom, and if nickel be present a greenish tinge is observed. The silver in this district is almost native and is associated with calcite gangue showing that the ascending solutions carrying silver were met by descending solutions carrying carbonate of lime.

While silver is found in the older formations in quartz veins, and in the Cobalt district with calcite gangue, the largest deposits are found in limestone adjacent to porphyry. It seems remarkable at first that this should be the case until the same phenomenon is noticed in the case of copper sulphides and galena.

Both copper and galena when in sulphide form are associated almost invariably with silver. If they become weathered or taken in solution they leave the silver to be absorbed by some other solution, and this traveling along crevices in limestone eventually finds some mineral or solution which precipitates the silver.

For example, silver sulphate in solution could be precipitated by a salt solution forming insoluble silver chloride, known also as horn silver or cerargyrite. Large deposits of this mineral are rare, but it is found associated with other silver minerals. It is soluble in ammonia but not in nitric acid. Being soft and waxy it is readily melted, and will, when wet and placed on zinc, turn black, swell up, and show metallic luster if pressed with the point of a knife. It is usually found near the surface, but also in some depth in deposits formed from other minerals, being oxidized and taken into solution. Horn silver contains 75 per cent. silver and 25 per cent. chlorine. It is whitish with other shades of color at times.

Should the silver be in solution and meet another solution containing antimony, ruby silver would be formed. Ruby silver is a black mineral with a dark red tinge. It generally looks massive, but under the glass it is seen to be finely crystallized. It gives a dark red streak on porcelain; is easily melted, giving off sulphur fumes, and is soluble in nitric acid. It might be mistaken for proustite or light red silver, but to the prospector this makes little difference. Light red silver contains sulphur and arsenic instead of sulphur and antimony. It is soft, gives a bright red streak and has a bright red color. It is decomposed by nitric acid and melts easily. Both minerals are associated and at times carry considerable gold. Several other silver minerals are known, but usually are associated with other metals or non-metals to such an extent as to be unrecognizable, although at times they can be determined by tests.

Limestone is generally bluish-gray or drab, but cannot always be recognized with the eye; however, a drop of acid will cause it to froth. Silicious limestone will not effervesce, and in all probability silver deposits will not be found in it. Porphyry weathers more or less at the surface unless highly silicious, and is generally white or gray with crystals of some other minerals prominent in a fine mass. It is an intrusive rock that often forms great dikes. It is presumed that solutions bearing minerals accompany the intrusion, and these, finding their way into limestone, deposit their mineral constituents. Another reasonable supposition is that the mineral in porphyry weathers and is carried down by surface waters and is then deposited.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

WESTERN EDITORIAL OFFICE: 728 Equitable Building, Denver, Colo.
CHICAGO OFFICE: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to MINES AND MINERALS, Scranton, Pa.
CABLE ADDRESS, "RETSOP, SCRANTON"

RUFUS J. FOSTER........Manager
EUGENE B. WILSON, Scranton, Pa........Editor
GEORGE F. DUCK, E. M., Denver, Colo........Western Editor
P. G. MOORE........Circulation Manager
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., Ltd., Dryden House, 43 Gerrard St., Soho, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 Pitt Street, SIDNEY.
SOUTH AFRICAN AGENTS: J. AFFLECK ROBERTSON, CAPE TOWN, Mansion House; JOHANNESBURG, Trust Building; DURBAN, Natal Bank Chambers; BLOEMFONTEIN, Florence Buildings.

SUBSCRIPTION RATES

Single Copies........25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada........2.50
To all other Countries........3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always give your FORMER as well as your present address.

TABLE OF CONTENTS

Retort Coke Ovens in Mexico, by E. B. Wilson........257
The Starkville, Colo., Explosion........261
Electric Air Compressor, by Jas. A. Seager........263
Chemistry of Coal Dust Explosions........264
Handling Mine Supplies, by H. H. Fitch........267
The Taylor Concrete Breaker, by E. B. Wilson........272
Personals........273
Mine Fires, by Thomas K. Adams........274
Mine Accident Prevention, by J. J. Rutledge........276
Trade Notices........278
Answers to Examination Questions........279
Catalogs Received........281
Mine Accident Investigations, by Geo. S. Rice........282
Turboblowers and Compressors........285
Society Meetings........286
Falcone Oven Tile-Laving Machine........287
Determining Vertical Depths in Drill Holes, by Professors Kalmus and Lewis........288
Electrically Driven Pumps........288
Prospecting for Silver........289
Editorial—
The Mexican Situation—Mine Promotion Literature—Educating Anthracite Miners—How About This........290
Book Review........291
Books Received........292
Correspondence—
Rules for Mine Foreman—Fernie Slack Bin Fire—Miscellaneous Questions........293
Georgia Brown Iron-Ore Washeries, by E. F. McCrossin........294
Drilling in Upper Peninsula........295
Utah Metals Co. Tunnel, by Leroy A. Palmer........296
Lake Superior Iron Ore........297
Evolution of Hoisting, by E. B. Wilson........298
Hydrometallurgy of Cobalt Ores........303
Tungsten Ore in Washington, by August Wolf........307
Tin Sluicing in Tasmania, by Ed. Edwards........309
Chitina Valley Copper Deposits, by E. Jacobs........315
Ore Mining Notes........318
Obituary........318
The Ruby, by Morris R. Ward........319
New Inventions........320
Squibs—
Coke Oven Gas, 266—No Progress in Alaska Coal Mining, 272—English Slag Paving Blocks, 273—Cost of Sinking a Mexican Mine Shaft, 275—A New Smoke Helmet for Mine Fire Fighting, 281—Coal Notes, 281—Coal Tonnage of Texas, 284—The Innoko District, Alaska, 288—Competition Among Smelting Companies, 293—Report of American Zinc, Lead and Smelting Co., 293—Duralium, 314—Air-Nitrate Fertilizers........320

THE MEXICAN SITUATION

PEOPLE in the United States when asked to invest in Mexican enterprises, have stated that while under President Diaz their investments would be safe, there are a large number of malcontents who under less firm management would stir up internecine war. The educated and refined Mexicans, the men who have made their wealth by their brains, voice the same opinion, all of which reflects on the integrity of Mexico as a nation.

The answer to this has been, that so much foreign capital is now invested in Mexico that the United States under the Monroe Doctrine will be obliged to protect it.

While this is undoubtedly true, the uprising against United States citizens who have transgressed no law of the Republic, and the insult to the people of the United States as a whole, argues that the self-imposed patriots are plunging their country into trouble.

People in the United States desire to live peaceably, and to that end accept a certain amount of imposition, but when their neighbors become a nuisance hard feelings are engendered and ructions occur.

The people in the United States know that the citizens of Mexico did not encourage the half-baked students and political shysters to insult them by trampling on their flag, in fact the press as a whole, held the Mexican Government blameless, in a matter which no European country would tolerate.

In nearly every locality in Mexico there is a politician who imagines himself the government in his bailiwick. This species hates the gringo who invades his bastion and pays out real pesos that accomplish results, for the reason that his importance as the leading citizen is on the wane and his former constituents are climbing to an eminence which challenges his superiority.

The Germans have circulated scurrilous literature to foment this hatred and, as the politician has a certain following, progress toward improvement is slow. Bucolic and jealous, he is unable to realize that the foreign invasion adds to his country's wealth and enlightenment, and being ignorant of affairs he is unable to share in the general prosperity. He has a grouch because the foreigners do not place him in some position of importance which he is not able to occupy. The younger generation are in a better position and will occupy positions of trust and responsibility provided they take advantage of their opportunities to attend the schools and colleges in Mexico that make for progress. People who have invested in enterprises in Mexico would rather employ Mexicans than people from the States, for the latter are apt to leave at any time for home. The writer is sure of this for he left an excellent position in Mexico to be with his own people, and therefore from hearsay and knowledge there seems to be no question but that the educated Mexican will come into his own, always provided the "sour balls" can be subdued. There are certain states in the United States whose laws inflict death for ravishing women. As the

people make the law, they sometimes constitute themselves judge, jury, and inflictors of punishment.

If the man in Texas had not committed the crime he did, he would not have received the horrible death penalty. Even this was no excuse for mobbing inoffensive citizens whose only crime was that they were English-speaking Americans, and an abject apology is coming.

MINE PROMOTION LITERATURE

JULIAN HAWTHORNE claims that for the sake of the Hawthorne Mining Co., Ltd., whose properties are said to be in Canada somewhere, he has abandoned literature and journalism. The *Cobalt Nugget* says: "A perusal of the prospectus would seem to refute this, as he has not given up fiction and he still wields a fascinating pen."

The *Mining Investor* says: "Julian Hawthorne is a writer who has nobly overcome the handicap of a distinguished literary name." "In his pamphlet, 'The Secret of Solomon,' Mr. Hawthorne points out three well-known facts: First, that King Solomon was the wisest of men; second, that he was also the richest of men; and third, that he made his money from mining."

The mysterious veil which heretofore enveloped Solomon's title to supremacy in wisdom is now brushed aside—he was a mining engineer.

A poor man's riches consist in children, and as Solomon had a thousand wives who worked for him, and no children to speak of, he ought to have been as rich as Rockefeller at least. That he made his money from mining should be qualified to read, he obtained his money from mines. Some of the thought gems from Hawthorne's "Solomon" are the following:

"Unwrap from its napkin that talent in the safe-deposit drawer."

"Take the tide at its flood."

"Climb, and do not fall."

"Open the door to opportunity."

"Remember that Solomon looked before he leaped."

"Study the situation well."

"He that lingers till tomorrow buys dear what was cheap yesterday."

As a Recessional to those who hesitate to invest he furnishes this:

"He either fears his fate too much
Or his desert is small,
Who dares not put it to the touch
To win or lose it all."

EDUCATING THE ANTHRACITE MINER

COL. R. A. PHILLIPS, Superintendent of the Delaware, Lackawanna & Western Railroad Co.'s coal mining department, has reached the conclusion that it is easier to teach non-English speaking mine workers to protect themselves in the mine through their eyes than through their ears.

He is therefore having a series of about 80 subjects photographed showing the right and wrong method of doing things in mines.

Mr. Bunnell, the official photographer, takes these pictures inside the mines, thus making them realistic and difficult to be misunderstood.

Slides will be made of the photographs so that they may be projected on screens where all miners attending the monthly meetings of the Y. M. C. A. mining institute can see them.

The Y. M. C. A. is laboring faithfully to educate the foreign-speaking mine workers in order to reduce fatalities and injuries to a minimum.

Mine superintendents appreciate their endeavors and aid the representatives financially and in the practical way mentioned. Colonel Phillips proposes to supply a full set of slides to the anthracite committee of the Y. M. C. A. for each class of foreign miners.

HOW ABOUT THIS

BEFORE many years have passed the hard coal operators in Northeastern Pennsylvania will be confronted with a problem as serious as any that has occurred in anthracite mining. According to prevailing laws a man must work 2 years as a laborer, 5 years as a miner, and then pass a rigid examination as to his fitness to occupy the position of mine boss.

At present, with competent captains at the helm, the coal companies are sailing unruffled seas, but where are the master mariners' mates who will command the ships when nature asserts itself and the captains go over the side. Would it not be policy to import young educated English speaking miners from other coal fields; make it worth their while to work as laborers 2 years and miners 5, and at the same time tutor under the present mine foreman?

BOOK REVIEW

Smoley's Tables, sixth edition, has been issued by the Engineering News Publishing Co., price $3.50. Flexible leather, 4¾ in. × 7 in., 518 pages. The tables are combined in a way to obviate the tedious calculations in structural and other branches of engineering. Their convenience led to the immediate adoption of the early editions, and the work was soon regarded as indispensable to the designer.

In the later editions, tables of angles and logarithmic functions corresponding to any given bevel were added. Also a complete set of logarithmic and trigonometric tables were included. The present edition contains a complete table of natural trigonometric functions for every minute of the quadrant; a table of reciprocals, squares and square roots, cubes and cube roots, together with circumferences and areas of circles corresponding to diameters from 1 to 1,000. This latest table, by supplementing the tables of squares and logarithms of lengths in feet and inches, makes a desirable addition.

Compressed Air Plant, by Prof. Robert Peele, 502 pages, 8 vo., 209 figures. Price $3.50. This second edition is enlarged by 174 pages and 97 illustrations. The additions relative to

the construction and operation of rock drills, coal-cutting machines, and channeling machines occupy 90 pages. The theory of the compression of air is presented in greater detail and deductions from the more important formulas used to calculate the horsepower in single- and multiple-stage compression are given. Professor Peele has from time to time furnished articles on compressed air for Mines and Minerals and is therefore known to the older subscribers. However, for the newer subscribers it is stated that a more comprehensive and practical book on compressed air is not printed. Publishers, John Wiley & Sons, New York; Chapman & Hall, London.

Ore Mining Methods, by Prof. W. R. Crane, 8 vo., 219 pages, 60 illustrations; John Wiley & Sons, New York City, publishers. Price, bound in cloth, $3. In this book Professor Crane has not attempted to cover the machinery, tools, or other apparatus used in excavating ore. This elimination permits the concentration on the methods adopted in modern exploitation without digression; also it permits the author to discuss his theme connectedly without cross-references to subjects creeping into the text to distract the reader. The book forms a contrast to those usually put out on mining, and because of its valuable contents will appeal to ore miners. There is more on the methods of ore mining in this book than the number of pages would indicate for the reasons stated.

Practical Shaft Sinking, by Francis Donaldson, Chief Engineer of the Dravo Contracting Co.; 139 pages, 8 vo., 63 illustrations. Price $2. McGraw-Hill Book Co., New York City. This book is written by a practical man engaged in the business of shaft sinking, and covers the ground of modern shaft sinking from *a* to *z*. Chapter I includes contract agreement for excavation; the extra prices to be paid for excess of water pumped above 100 gallons per minute; and timbering specifications where shafts require to be lined. Chapter II covers the power plant required for sinking shafts to various depths; the disposal of excavated material; and an itemized cost of an entire sinking plant for a shaft 500 feet deep. Chapter III. In this chapter the author shows how to sink and support earth from the surface to rock; the various kinds of piling used, including steel; the use and construction of circular and oblong concrete caissons, with the construction of the shoes. Chapter IV covers the pneumatic process of sinking through soft ground; also the shield method practiced in Illinois. Chapter V covers rock excavations; the tools for drilling; placing holes for blasting; progress and probable cost per foot. Chapter VI. The processes followed abroad and the Poetsch-Sooysmith freezing process are described. Chapter VII describes the Kind-Chaudron process, and the cementation of water-bearing fissures. Chapter VIII is devoted entirely to lifting water during the progress of shaft sinking. Chapter IX deals with various kinds of shaft linings used abroad and in this country. Chapter X gives the cost per linear foot for rectangular, angular, elliptical, and quadrilateral cement-lined shafts.

BOOKS RECEIVED

United States Geological Survey Publications, Washington, D. C., Bulletin 381, Contributions to Economic Geology, Part II, Mineral Fuels, by Marius R. Campbell; Bulletin 426, Granites of the Southeastern Atlantic States, by Thomas Leonard Watson; Bulletin 429, Oil and Gas in Louisiana, with a brief summary of their occurrence in adjacent states, by G. D. Harris; Bulletin 435, A Reconnaissance of Parts of Northwestern New Mexico and Northern Arizona, by N. H. Darton. Water-Supply Paper 240, Geology and Water Resources of the San Luis Valley, Colorado, by C. E. Siebenthal; Water-Supply Paper 253, Water Powers of the Cascade Range, Part I, Southern Washington, by John C. Stevens; Water-Supply Paper 255, Underground Waters for Farm Use, by Myron L. Fuller; Water-Supply Paper 260, Preliminary Report of the Ground Waters of Estancia Valley, New Mexico, by Oscar E. Meinzer.

Department of the Interior, Bureau of Mines, Washington, D. C., Bulletin No. 1, The Volatile Matter of Coal, by Horace C. Porter and F. K. Ovitz.

Circular of Bureau of Standards, S. W. Stratton, Director, Department of Commerce and Labor, Washington, D. C., Bulletin No. 23, Standardization of Electrical Practice in Mines.

Canada Department of Mines, Eugene Haanel, Ph. D., Director. Ottawa, Canada, Report of Analysis of Ores, Non-Metallic Minerals, Fuels, etc., made in the Chemical Laboratories during the years 1906, 1907, 1908, arranged by F. G. Wait, chief chemist; Bulletin No. 3, Recent Advances in the Construction of Electric Furnaces for the Production of Pig Iron, Steel, and Zinc, by Eugene Haanel, Ph. D., Director of Mines; Peat and Lignite, Their Manufacture and Uses in Europe, by E. Nystrom, M. E.; Bulletin No. 1, Investigation of the Peat Bogs and Peat Industry of Canada, During the Season 1908-9, by Erik Nystrom, M. E., and S. A. Anrep, M. E.; Bulletin No. 4, Investigation of the Peat Bogs and Peat Industry of Canada During the Season of 1909-10, by Aleph Anrep, Jr., Peat Expert, to which is appended Mr. Alf. Larson's paper on Dr. M. Ekenberg's Wet-Carbonizing Process; Joint Report on the Bituminous or Oil-Shales of New Brunswick and Nova Scotia, also on the Oil-Shale Industry of Scotland, Part 1, Economics, Part II, Geology, by R. W. Ells, LL. D.

Bulletin No. 40, A Study in Heat Transmission, by J. K. Clement and C. M. Garland, University of Illinois, Urbana, Ill.

Friction in Small Air Pipes, by Prof. E. G. Harris, Elbert Park, and H. K. Peterson, Bulletin of School of Mines and Metallurgy, University of Missouri, Rolla, Mo.

Meteorite Studies, by Oliver Cummings Farrington, Curator, Department of Geology, Field Museum of Natural History, Chicago, Ill.

Twenty-Eighth Annual Coal Report of the Illinois Bureau of Labor Statistics, David Ross, Secretary, Springfield, Ill.

A Modern Method of Pumping, Starrett's System of Pumping by Compressed Air, by D. W. Starrett, Salt Lake City, Utah.

Annual Report of the State Geologist of New Jersey for 1909, by Henry B. Kummel, State Geologist, Trenton, N. J.

Scope and Progress of the Mining Industry in Colorado, Colorado School of Mines Quarterly, Golden, Colo.

A Commemorative Bulletin, No. 32, Ohio State University, Columbus, Ohio.

The Mineral Resources of the Philippine Islands, issued by the Division of Geology and Mines Bureau of Science, by Warren D. Smith, Chief, Manila, P. I.

North Carolina Geological and Economic Survey, Joseph Hyde Pratt, State Geologist, Chapel Hill, N. C.; Economic Paper No. 20, Wood-Using Industries of North Carolina, by Roger E. Simmons.

Map of Cobalt-Nickel-Arsenic-Silver Area, near Lake Temiskaming, Ontario, to accompany fourth edition of report, by Willet G. Miller, Provincial Geologist, in Part II of the Nineteenth Report of the Bureau of Mines, 1910.

Annual Report of the Consolidated Mercur Gold Mines Co. for the year ending June 30, 1910, by John Dern, President, Mercur, Utah.

Nineteenth Annual Report of the Bureau of Mines, Ontario, Canada, for 1910.

Mine and Quarry for October, published by the Sullivan Machinery Co., of Chicago, Ill., contains "Modern Methods of Mining on the Menominee Range," by L. J. O'Grady; "Diamond-Drill Boring on the Clackamas River, Oregon," by V. J. Hampton; "Notes on the Imperial Copper Co.," by F. C. Baxter; "Marble Quarrying in Arizona"; "Stoping Drill Steels"; "Recent Developments in the Undercutting of Coal by Machinery," by Edward W. Parker; "Atomizing Fuel Oil With Air," by R. E. Ellis.

CORRESPONDENCE

Rules for Mine Foremen

Editor Mines and Minerals:

SIR:—In my opinion the following rules should be carried out to make a successful mine foreman:

1. A mine foreman should be a man of good character, of cool disposition and sound in his judgment.
2. Should be courteous and discreet in hiring and handling men.
3. Should be able to gain the respect and confidence of his subordinates without sacrificing his individuality.
4. Always insisting on orders being obeyed.
5. Should be able to assert his personal, as well as his representative, rights when circumstances demand it.
6. Should be able and qualified to give instructions by example as well as by precept.
7. Always provide plenty of ventilation.
8. Always keep good roads for the transportation of coal.
9. When an order is given you by your superior, see that it is carried out.
10. Always keep in mind next thing to the protection of your men, the protection of the company's property.
11. Always give a man a civil answer.
12. Always use the company's money as if it were your own.

Carbondale, Pa. T. J. ASTON

Fernie Slack Bin Fire

A fire occurred at the Fernie slack bins, Fernie, B. C., recently. Mr. Ashworth writes "that he anticipated either a violent inflammation or explosion when the sides gave way and threw a large volume of coal dust into the air in close contact with the flaming woodwork, but no explosion took place." "If, as argued by some coal dust experts, a 'pioneering cloud' is necessary to originate a coal-dust explosion in a mine, he would like to ask why he did not have an explosion? With plenty of air, plenty of fine coal dust, and plenty of hot fire."

Miscellaneous Questions

Editor Mines and Minerals:

SIR:—1. What size of a box will it take to measure 1 ton of lignite coal?

2. For a 3-foot seam of lignite coal with only a moderate roof (area of coal 640 acres) what size of car would you prefer, and what size of car wheels and size of rails?

3. For a 4-foot seam of lignite coal with a very poor roof (area of coal to be taken out is 40 acres) what size and kind of coal car and size of rails would be preferred.

4. What is the horsepower of a boiler 42 inches in diameter and 42 feet long with two 12-inch flues?

5. What is the horsepower of a Cameron pump, diameter of the plunger 6 inches; length of stroke, 13 inches; size of steam cylinder, 10 inches in diameter; steam pressure at the boiler, 90 pounds?

6. What is the safe load for a 1-inch plow-steel rope, to haul up an incline 350 feet long; angle 35 degrees?

Bow Island, Lethbridge SCOTCHMAN

A DRY TIME IN SEWARD PENINSULA

All the placer camps were prosperous in 1909 except those of Seward Peninsula, where dry weather curtailed the output. In this region the methods of mining must evidently be adapted to meet the exigencies due to frequent seasons of low water. Important advances were made in dredging enterprises.

COMPETITION AMONG SMELTING COMPANIES

According to men who have arrangements with the International and allied smelting interests like the Amalgamated and Cole-Ryan people, the competition between the International and the American Smelting and Refining Co. is more bitter than the public has understood. With the erection of an International smelter in the Ely district of Nevada, as contemplated every field of the American Smelting and Refining Co. except Colorado will be invaded by the new company. Colorado is said not to be inviting for the Cole-Ryan people, as ore production has fallen off so greatly in that state that the Guggenheims can operate only three of their seven furnaces there. The International and its allied interests now have the smelters at Anaconda and Great Falls, Mont.; the plant at Tooele, Utah; the Calumet and Arizona smelter at Douglas, Ariz.; and the Cole-Ryan plant at Cananea, Mex., covering a large part of the field from which the American Smelting and Refining Co. draws its copper ores.

When the latter had a clear field its charges for ore treatment were so high as to allow a profit of $4 to $6 a ton. The excessive smelter charges made many enemies among Western miners, and the International interests have an advantage in that fact, while its smelting plants are new and modern. With these advantages the International will be satisfied with profits of $1 to $1.50 a ton and its purpose evidently is to prevent the American Smelting and Refining Co. from renewing any of its old and highly profitable contracts. The best contract the American Smelting and Refining Co. has is that of the Utah Copper Co., and it is said that the profits made by the smelting company on it are enormous.

The International interests expect to treat a normal tonnage of custom ore of about 5,000 tons a day. At a profit of only $1 a ton this would amount to $1,825,000 a year. It is expected that this will represent a tonnage that the American Smelting and Refining Co. will lose.—*New York Sun.*

REPORT OF AMERICAN ZINC, LEAD, AND SMELTING CO.

In the annual report of the American Zinc, Lead, and Smelting Co. for the year ending June, 1910, the net profits for the year amounted to $238,885.22. From this, dividends 6 and 7 amounting to $80,520 at 50 cents per share each were declared. The balance of the profits carried to surplus amounted to $103,365. $25,000 was added to a special reserve fund for further additions and betterments of the property.

The additions to the mines in Missouri consisted of the purchase of 16¾ acres of land adjoining the Davey properties, the installation of an air compressor and the completion of the equipment of the Vogey mill upon the December tract. The additions and betterments at the smelters consisted principally of new gas lands and leases acquired, development of gas lands and construction of pipe lines.

The property account is represented by 2,416 acres of mineral lands owned in fee in the Joplin district, and upon these lands six operating concentrating mills, two smelters at Caney and Dearing, Kans., gas lands and gas leases, and an ore separating plant at Platteville, Wis.

In mining operations during the year 11¼ acres were mined from the Davey tract, but this has been more than made up by developments and the purchase of the adjoining tract.

On the December tract operations during the fiscal year were not carried on at the full capacity of the plant. It is now being worked up to nearly its full capacity of 1,000 tons per day. At the Kansas smelters operations were carried on at the full capacity throughout the entire year. The operation of the Wisconsin properties has been unsatisfactory. The development of the Huff process has been steady. Progress is being made at Wisconsin, Utah, and elsewhere.

GEORGIA BROWN IRON-ORE WASHERIES

*Written for Mines and Minerals, by E. F. McCrossin**

Method of Handling the Ore and Cleaning It by Use of Log Washers Operated by Steam or Electricity

The brown iron ores in Georgia so far developed are in Polk, Bartow, and Floyd counties. In Bartow County the deposits are named as follows: The Eastern district, the Iron Hill district, and the Linwood district. Probably the most important of these three divisions is the Eastern district, which is confined to the eastern part of the county. The deposits in this part of the state begin about 2 miles south of Emerson and extend in a northerly direction to Sugar Hill, a distance of about 16 miles. Some of the most important brown ore mines in the state, such as Sugar Hill mines, are located in this district. The brown iron ores occur chiefly in the form of pockets in the residual clays and are quite variable in size. It is not uncommon to find the deposits extending over 5 or 6 acres, although the ore does not occur in equal abundance over the entire area. On the contrary it is often traversed and intersected by clay horses. The depth to which the ore extends is also variable. In some cases it is quite superficial, being only a few feet in thickness, while in other cases they appear to extend to more than 50 feet in depth. In some instances the deposits on the hill slope are found to extend into the valley below. The ore is found mostly in the form of boulders or pieces in the residual clays, and the proportion which the ore bears to the entire material varies greatly. In some of the most valuable deposits the ore will average 50 per cent. or more of the material mined, but generally it falls short of this average, and it becomes necessary to wash the deposits in order to concentrate the ore for transportation.

Fig. 1. Sugar Hill Iron Mine Plant

The ore, which evidently has had its origin from the weathering of pyrite, varies in phosphorus content from .363 to 1.583. The silica varies from 5.82 to 16.47.

Until recently it has been the policy of the larger corporations operating in the Alabama and Georgia brown iron-ore fields, to erect small instead of large plants, preferring to have two small washers rather than one large one. As is the case in all other iron-ore fields, the rapid exhaustion of the richer deposits has compelled the erection of larger and more efficient plants in order that the lower grade ore may be successfully handled.

The most elaborate and recent developments along this line have just been completed on the Bartow Division of the Georgia Steel Co. The division consists of about 1,300 acres in northwest Georgia. The company owns its own standard-gauge railroad running through the property and connecting with two interstate railroads. Along the railroad are located the four principal plants: the Sugar Hill, Big Springs, Bufford, and Guyton washers, situated in the order named, Sugar Hill being the terminus of the road.

The plant at the Sugar Hill Mine, shown in Fig. 1, consists of a steel six-log washer, equipped with overhead distributing screens, sand screens, and four picking belts, three of which are of the steel-pan type, the other a 30-inch canvas belt. On the sand screens the standard slotted sheets heretofore used have been replaced by perforated sheets with ⅞-inch circular perforations. The ore, after passing through the distributing screens and logs, reaches the sand screens, the oversize passing through and on to the picking belts and the undersize to the jig plant attached to the washer, where it goes through a concentration process; the silicious matter being discarded and the concentrate loaded into the bins, from which it is later loaded into cars with the ore from the picking belts. This washer has been in operation about 3 years and has a capacity of about 500 tons per day. The jig plant is a more recent addition and has a capacity of about 16 tons an hour.

The mines at Sugar Hill cover a very large area and have been worked more or less for a number of years. Ore was hauled from them in wagons to small stone furnaces in the western part of Bartow County, which were in blast long before the Civil War. The ore from the working faces is hauled to the washer with standard-gauge locomotives in 6-yard, side-dump cars, loaded direct by steam shovels, as shown in Fig. 2. A branch line from the railroad extends to a deposit about 2½ miles from the washer. On this line 12-yard side-dump cars are used with much success.

The ore is a high-grade brown hematite lying in large irregular deposits with a massive sandstone foot-wall. It is good all the way down until the local water level is struck, where, as a rule pyrite begins to show and causes the sulphur to exceed the limit allowed by the furnaces. It is possible that some time in the future even this ore will be extracted and roasted.

Seven miles from Sugar Hill, on the company's railroad, lies the town of Aubrey, where are located the division offices, the machine shop, forge shop, power house, and a number of residences for the officials and men. This is the nucleus of the division. The steam shovels, locomotives, cars, and all classes of equipment are brought here for repairs. The shops are fully equipped with lathes, drill presses, shapers, steam hammers, and all other tools necessary to repair and rebuild the mining machinery. All the shop machinery is driven by electric motors.

The power house is a modern brick building with steel trusses. The current to operate the shops, the Big Springs and Bufford washers, and the two 1,200-gallon-per-minute pumps for the washers, is generated by two direct-connected, 550-volt, 455-ampere, direct-current generators and one 350-ampere belted generator; all connected in parallel, so that any machine can assume the load from any line by simply alining the proper switches. One machine is always idle and held in reserve. The current is sent to the washers over heavy copper cables. Chestnut poles set 75-feet centers hold the cables. There are frequently heavy static discharges in this district, but so far there has been very little trouble from this source. A large bank of lightning arresters has been provided at the power house and at each plant a number are also scattered, along the transmission lines.

* Consulting Engineer, Cartersville, Georgia.

The Big Springs plant, shown in Fig. 3, consists of a wooden four-log washer and is situated 2 miles east of Aubrey on a branch line of the railroad. The logs and screens are driven by a 100-horsepower motor belted to a friction wheel, which in turn drives the line shaft. This is an innovation in this district and should the logs become overloaded or jammed, the entire washer can be stopped at an instant's notice, and then by

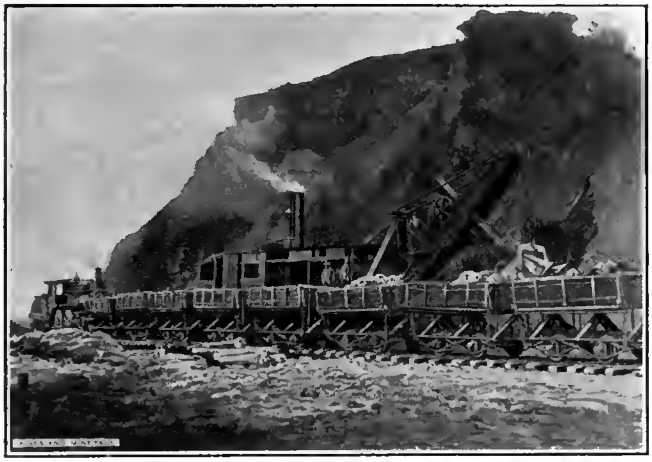


Fig. 2. Loading Iron Ore at Sugar Hill, Ga.

slowly applying the friction wheel to the shaft the logs can be relieved and loosened and gradually put up to their normal speed.

The ore is brought to this plant from the open cuts in narrow-gauge, 4-yard, side-dump cars, hauled by "dinky" locomotives. All the material is dumped into the hoppers on grates, the bars of which are set 6 inches apart The undersize from these grizzlies is flumed direct to the sizing screens. The oversize is separated, the foreign matter is thrown out and the heavy massive ore discharged into a No. 8 gyratory crusher from which it is flumed to the logs. The location of this plant is ideal and a minimum operative cost can be maintained.

The Bufford plant is not so favorably situated as the Big Springs washer. The slope of the hill upon which it was found necessary to erect the washer was so gradual that it necessitated the elevation of the mud-boxes, or hoppers, above the grade of the mine tracks. The mining conditions here are much the same as Sugar Hill, and the ore is brought to the washer in 6-yard side-dump cars. These cars are hoisted to the mud-boxes by a 70-horsepower electric hoist, and discharge direct into the hoppers. The crusher at this plant is elevated on timber bents to the level of the flume line, and the material follows the same course as it does at Big Springs. The washer is the same type as the one at Sugar Hill, except that it has four steel logs instead of six, and has no jigs.

Both of the washers have given very little trouble since their erection, and the cost of the operation of these washers and their pumps by electricity, compares most favorably with that of steam plants of equal capacity.

DRILLING IN THE UPPER PENINSULA

There has been a cessation of diamond drilling operations in all the iron fields of the upper peninsula.

Many drilling outfits have been transferred from the iron country to the Lake copper district, where exploration for new lodes is actively in progress. There will be much learned of Ontonagon County through the use of these drills, and many of them are being now employed in the district. Several promising discoveries have been made with the drill and the work is being supplemented with new borings to learn as much as possible concerning these deposits. Should one good copper deposit be proved it will well repay for all the carbons that have been used, because some day copper is to come into its own and find a market at a profit commensurate with the cost of getting it. Carbons at $85 per carat are pretty expensive agents for rock cutting, being about half the price of the ordinary-sized and ordinary-colored brilliant. There was a time when carbons used to sell at $6 to $10 per carat, but the demand has increased owing to the greater number of drills employed, and those who have the industry of carbon mining and selling in hand know how to conduct their affairs so as not to give their output away without proper compensation. At $85 per carat it is evident that the carbon men are reaping a rich harvest at this particular time.

Many substitutes have been tried for black diamonds, but up to this time none have been found worth mentioning. Bortz may be used if the ground is not too hard, and highly tempered steel has sometimes been employed. When it comes to boring the hard rocks of the Lake Superior country, however, the black diamonds are required, and it needs the best of these to cut the jaspers of the iron-bearing formations.

Much of the drill work in the lake region is done by contract with parties owning their drill equipments and having experienced men in charge. It stands to reason that with the high price of the black diamond a careless setter could easily waste a lot of money. The setter is a man of great importance. The runner also should know how to treat his bit of stones in the hole. He can easily destroy hundreds of dollars worth of stones in a single careless shift. The bit must be fed into the rock according to the hardness of the latter and it is only long practice that decides the proper handling of the machine by the operator. Good setters and machine men are always in demand and are well known to those using drills.

The diamond drill has been valuable in locating new iron ore deposits in the past few years. Probably there have been greater additions to the known iron-ore deposits through the use of the drill in this time than in any former similar period, this because they were being more used in new localities. It has been the practice, as it still is, for the mining companies to use drills in their mines, locating ore extensions and pockets on

Fig. 3. Iron Ore Washery, Big Springs, Ga.

the strike of the formation and contiguous to their underground operations. Nearly all the mining companies of the Lake Superior country own drills that are operated by their employes.

Shaft work will take the place of the drilling crews at many locations in the iron country, and it may be that the use of the drills in the Lake Superior copper country will be followed by new mines in that section already well known for its merit in a mineral way.

UTAH METALS COMPANY TUNNEL

Writen for Mines and Minerals, by Leroy A. Palmer

A Tunnel 11,000 feet long for Ore Transportation. Electric Driven Air Compressors

To reduce the cost of transportation to the Tooele smelter, a distance of 24 miles by rail, and only a few miles through the mountain the Utah Metals Co. is constructing a tunnel. This transportation project of the Utah Metals Co. starts in the property of the Bingham Central Standard Mine near the Utah Consolidated and pierces the mountain in a general westerly direction, coming out in Middle cañon at a point about 2,000 feet higher than the International smelter at Tooele, and 4 miles distant in a direct line, the topography being such that an aerial tram could be built without any great difficulty. The tunnel, when completed, will be 11,000 feet long with a grade of 3 to 4 inches in 100 feet. It will be 8 ft.×9 ft. in the clear with frequent passing tracks and, while the width of 9 feet would make it rather crowded, it could, if necessary, be double tracked without enlarging. It is being driven by compressed air generated by electricity in the company's hydroelectric plant. The water is drawn in two pipe lines, one from Middle Creek proper and the other from a fork of the creek just above the power house.

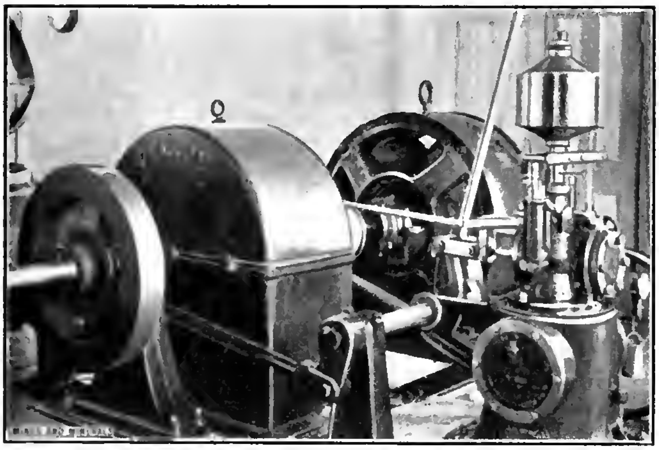


Fig. 1. Pelton Generating Unit, Utah Metals Co. Tunnel

The line from Middle Creek contains 5,600 feet of American spiral pipe; 2,400 feet of 12-inch, 3,000 feet of 10-inch, 100 feet of 8-inch, and, just before entering the power house, 75 feet of 5-inch. The water has a head of 750 feet that gives a static pressure of approximately 325 pounds per square inch, and the pipe is tested to four times this amount. The joints are of forged steel bolted type. A sleeve belled at each end is slipped over the pipe and a rubber gasket placed inside the bell. A flanged collar fits inside the bell against the gasket on which it is tightened by bolts through the flanges. This makes a joint which is tight under pressure and allows of some deflection without leakage. There is sufficient movement in the joints to take care of all expansion and contraction but they also call for very secure anchorage. This is secured by sinking a deadman 5 feet in the ground at every other joint. A clamp is put around the pipe and a bolt attached, the other end of the bolt being passed through the deadman.

The water from the fork is carried in a wrought-iron pipe 6,000 feet long, 1,420 feet of 12-inch, 1,078 feet of 8-inch, 1,807 feet of 6-inch, and 1,620 feet of 5-inch. The head is 800 feet, giving a pressure of approximately 400 pounds per square inch. This pipe has ordinary sleeve joints. It is buried for the greater part of its length which does away with part of expansion and contraction in length due to temperature and as it is on rather sharp curves such changes as do occur are taken up by lateral movements. The fact that it is buried relieves some of the necessity of anchoring, and the anchors, which are of the same style as these used on the spiral pipe, are placed only every 500 feet.

The spiral pipe discharges against a 30-inch Pelton wheel, Fig. 1, with deflecting nozzle and Pelton governor. This wheel is direct-connected to a 55-kilowatt direct-current generator making 875 revolutions per minute and generating at 5,550 volts and 10 amperes. The current is carried to the tunnel 1,100 feet distant where it is used for the haulage system and to drive a small fan.

The wrought-iron pipe line discharges against a 36-inch Pelton wheel belted to an Ingersoll-Sergeant two-stage compressor, 9½ in.×15 in.×10 in. which can be run at 140 revolutions per minute, thus giving a capacity of 400 cubic feet of free air per minute. The compressor works to 100 pounds. The air is carried in a 6-inch line to a point 200 feet within the portal of the tunnel where the line is reduced to 4 inches and carried to a receiver near the working face. From the receiver the machines are fed by a 2-inch line.

At this writing the tunnel is in 3,900 feet through quartzite with occasional belts of limestone. It is 8 ft.×9 ft. in the clear with a drain at the side in which is laid a flume 24 in.×24 in. The flow of water, 380 gallons per minute, is piped in a 5-inch line to the power house and directed against an 18-inch Pelton wheel direct-connected to a 3-kilowatt direct-current generator which supplies the lighting system. At present the company is getting more than sufficient power for its own use and no attempt is being made to utilize the tunnel water other than described, but when the work is completed this water can be led down the cañon to a point, still on Utah Metals ground, where sufficient head can be obtained to develop 500 horsepower.

Drilling is done by two Ingersoll-Sergeant 3¼-inch piston machines mounted on the same bar. The ordinary round consists of 12 holes, three cut holes with as much pitch as the ground will stand, three breast holes with a slight angle downwards, three uppers at a slight angle, and three lifters with just sufficient angle to carry water. This round is varied to suit the nature of the ground, and frequently three or four additional holes are drilled. The holes are charged with 40-per-cent. dynamite, 50 to 70 pounds to a round, and fired by hand. Three shifts are at work and each breaks a little over 4 feet in the regular sized drift.

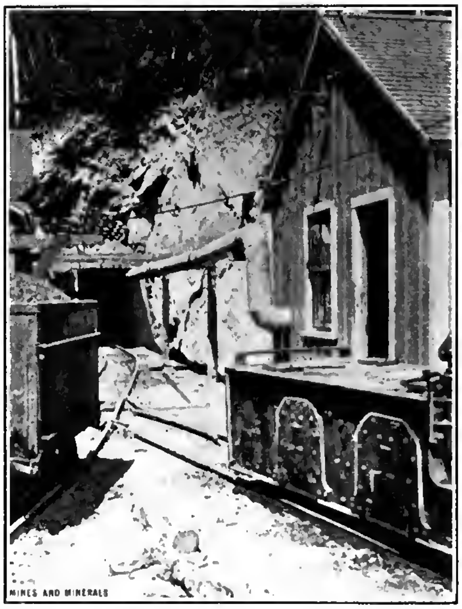


Fig. 2. Portal and Fan House, Utah Metals Co. Tunnel

The formation so far encountered is sufficiently firm to stand with little support. Whatever timbering is needed

consists of stulls or posts with head-boards and a few full drift sets. If bad ground is encountered, a drift set with the cap framed in three segments so as to form a rough arch will be used. The timbers, of which there is an abundance on the property, are undressed. The track is 24-inch gauge, of 30-pound rails on ties whose centers are 24 inches apart. The cars have a capacity of 42 cubic feet, with slope bottom, so that by throwing a catch one side is released and the rock slides out. They are hauled in trains of four by a Baldwin-Westinghouse locomotive. Ventilation is effected by a 48-inch Sirocco fan direct-connected to a 15-horsepower direct-current motor. It is used as a blower while the men are drilling and as an exhauster after blasting.

At first it was installed as shown in Fig. 2, but was changed and placed inside the tunnel 2,300 feet from the portal where it gives a much higher efficiency. The fan has a capacity of 5,300 cubic feet of air per minute and the object in changing its position was to balance to a certain degree the length of the intake and discharge pipes, and to use it as an exhaust fan all the time. The cost of driving 10½ feet in two shifts is as follows: Sixteen men at $3.50, $56; 90 pounds explosive at 16 cents, $14.40; two engineers, $7; blacksmithing, $7; superintendent and engineering, $10.67; total for 10.5 feet, $95.07, or $9.15 per foot.

The principal object of the tunnel is to afford transportation for the ores of the Utah Metals Co., which owns about 3,500 acres in Bingham, to the International smelter, but it will doubtless be used to handle the ores of other companies as well. It is about 50 miles from the upper Bingham mines to the Tooele smelter by rail. By the tunnel and an aerial tramway this distance will be cut to a little more than 6 miles. The water will develop a considerable amount of power which can be disposed of to advantage, and there is good reason to believe that mineral will be encountered. The formation is quartzite and lime which, on the other side of the range, frequently gives values on the contact and in the fissures in the lime, and the tunnel has encountered some mineralization in the lime but not of sufficient value to be classed as ore. No attempt is being made to follow up these indications, the company confining the work, at present, to getting the tunnel through the mountain as quickly as possible. At the present rate of progress the work should be completed in the next 18 months or 2 years and should prove a very convenient outlet for the Bingham ores which are to be treated at the Tooele plant.

The writer wishes to acknowledge the courtesy of those who have assisted in the preparation of this article, notably, S. H. Treloar, General Manager, and H. M. Hurst, Engineer of the Utah Metals Co.

LAKE SUPERIOR IRON ORE

That the Marquette range, from Ishpeming to Michigammee and beyond, will be tested with diamond drills during the next few years, is now certain. Some of the mining companies owning lands in that region have met with such success that continued and more extensive exploratory work is being planned. The Marquette range is the oldest of the Lake Superior iron districts, yet evidences multiply that it is far from the zenith of its development. Ore has been encountered in many drill holes.

It is expected that the Cleveland Cliffs Iron Co. will undertake the draining of North Lake, on the east shore of which the North Lake Mine is now being opened. This lake is 3 miles west of Ishpeming. In view of the fact that ore bodies also have been found on the west shore, it is believed that the deposits extend under the lake, which is three-quarters of a mile wide. Thus there is indicated an ore body of wide area, particularly since the diamond drilling has determined the deposit to be of considerable depth.

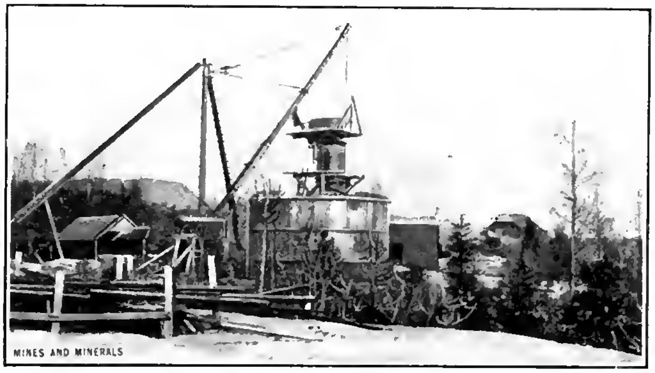
The district to the westward of Ishpeming promises to make good the loss of ore to which the older producing portions of the range have been subjected the past 60 years, during which many million gross tons have been sent to market.

The Cleveland Cliffs Co. was the first to begin a systematic exploration of the field. It mapped out the geology, beginning at the Cliffs shafts, Ishpeming, and with diamond drills it has tested the country westward for several miles. The discovery of the North Lake ore body was due to this carefully worked out plan. A large sum of money has been spent in exploring and equipping the property for the work of mining that soon is to be commenced. There is a concrete shaft to the ledge,* a fine plant of machinery in substantial buildings, and an attractive location of dwellings for employes.

Further west of North Lake, where the Cleveland Cliffs Co. is testing lands near the old Dexter hematite mine, a little village has been built and a shaft is well along toward the ore deposit. A steel head-frame and substantial buildings holding adequate machinery are also erected. This property, the Barnes, will soon be in condition to mine ore.

West of the Dexter Mine the next active property is the American Mine, where the Hanna interests are about ready to operate. The American Mine was operated years ago, but the ore bed was small and it was not worked advantageously.

The shaft at this mine has been deepened, new levels added, and the mine equipped with necessary machinery,



NORTH LAKE SHAFT, SHOWING LOCK FOR PNEUMATIC PROCESS OF SINKING

preparatory to mining. Exploration shows the ore deposits to have greater dimensions than supposed at any former time in the mine's history.

West of the American Mine, in a large swamp, George J. Maas, of Negaunee, and several associates, have been operating diamond drills for 2 years. A bed of hard ore and an immense deposit of soft hematite of commercial value has been found. There is also a large area of low-grade ore, which it is planned to concentrate.

The property which is to be developed by the North Range Mining Co. will be supplied with a shaft to be sunk at least 700 feet before the ore is sought by connecting drifts.

The Maas interests control a large acreage in the vicinity and intend doing work on other lands than those embraced within the corporation now being organized. It will take a year to get the shaft down to the high-grade deposits, which will be mined first.

The steel corporation is exploring west of the Maas holdings. It has two diamond drills in commission and intends to put down a series of deep holes.

From Ferdinand Schlesinger's Newport Mine, on the Gogebic range, more than 1,000,000 tons of ore will be shipped this year. The Newport is the largest producer on the Gogebic, as well as one of the most extensive underground mines in the Lake Superior iron region. Not infrequently, upward of 600 six-ton skips are hoisted to surface through a single shaft in the course of a 10-hour shift and that from a depth of 2,000 feet.

* See MINES AND MINERALS, Vol. XXX, page 217.

EVOLUTION OF HOISTING

Written for Mines and Minerals, by E. B. Wilson

(*Continued from November*)

In the November issue of Mines and Minerals, where balancing hoisting engines by different methods was discussed, the flat-rope reel was mentioned. It must be remembered that the greatest load comes on a hoisting engine when the cage, rope, and load are being moved from the bottom landing of the shaft, and to minimize this load the reel drum is made small in diameter.

Engines for Flat Hoisting Ropes. Koepe System, Poore System, Whiting System

The power which the engine must exert to move the load from this place is influenced by the diameter of the reel. Therefore, as the rope winds up and increases the diameter, the leverage becomes greater; at the same time the weight of

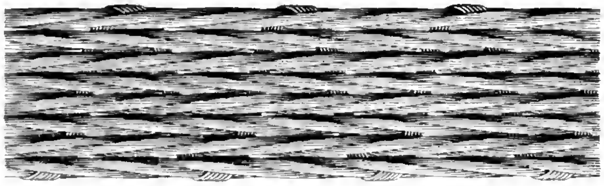


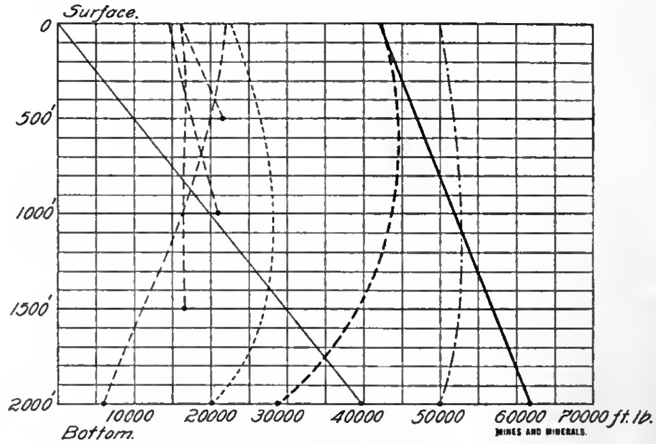
Fig. 26

the rope becomes less. So the two factors in a measure balance and the engine runs comparatively steady.

Flat ropes consist of several strands of round rope sewed together with wire, as shown in Fig. 26. Ropes of this discription are made of iron, steel, crucible-steel, and plough-steel wire, the same as other wire ropes. They possess the advantage of being less rigid than round ropes of the same strength, and may be wound on smaller drums without so great injury. Flat wire ropes weigh more than round wire ropes of the same strength and add that much more weight for the engine to lift. They will last equally as long under favorable conditions as a round rope of the same strength, with this distinction, however, that the round rope will need no repairs, while the flat rope will probably need to be resewed several times before being discarded. One of the advantages to be derived from the use of flat wire rope is that the drum may be lined or centered with the head-frame sheave. This centering avoids the extra strain put on the pulley at that point and does away with the extra power the engine would require to overcome the friction due to the rope angling at the sheave. Owing to this favorable feature the hoist and shaft house might be under the same roof were not such an arrangement bad mining practice, that in case of fire involves the risk of smothering all miners below the surface unless there are two mine exits

Diagram of Static Moments for the Following Conditions:

Ore.......................................4,000 pounds
Cars, cage, and rope, from head gear to cage..........4,500 pounds
Round rope..................................$1\frac{1}{8}$ inch diameter
Flat Rope.......................................$4\frac{1}{2}$ in.$\times\frac{5}{8}$ in.
Straight drum........................9 feet $10\frac{1}{2}$ inches diameter
Conical drum.......................8 feet diameter at small end
Conical drum...............11 feet 9 inches diameter at large end..
Reel center ....4 feet diameter



Straight drum, hoisting singly, loaded
Conical drum, hoisting singly, loaded
Flat-rope reel, hoisting singly, loaded
Flat-rope reel, hoisting singly, empty to change levels
Straight drum, hoisting in balance, loaded from different levels
Flat-rope reel, hoisting in balance, loaded, from different levels

Fig. 27

Note.—The worst disaster in the history of anthracite mining occurred at Avondale, Pa. It was due to the shaft house, placed over the only exit from the mine, taking fire.

It is probable that the principal objection to the use of flat ropes comes from the necessity of making repairs; in more recent years, however, men have become skilled in binding together the round ropes that make the flat one, and this in a measure overcomes the objection.

In the United States, reels vary from 3 to 11 feet in diameter, and as the rope winds on itself, each turn increases the diameter of the reel by twice the thickness of the rope, so that at the finish the diameter of the reels may be from 8 to 20 feet, according to the length of the rope, its thickness, and the original diameter of the reel. In England, reels are made up to 16 feet in diameter, and increase up to 20 feet or more, depending on the length and thickness of the rope.

Large reels are naturally less wearing on the rope, although, in any case, as the rope winds on itself that factor has considerable to do with the life of the rope. In this country the common practice is to make the diameter of the reel as small as the rope will

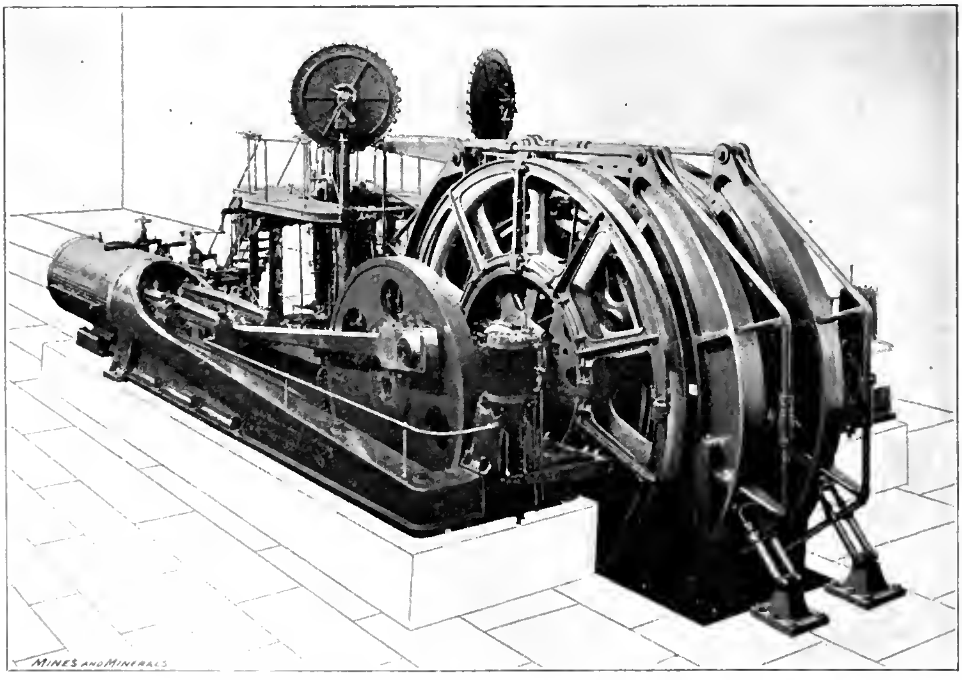


Fig. 28. Double Engine for Hoisting 3,000 Feet With Flat Rope

permit and so reduce the starting load. Assume the load to be that given for the diagram, Fig. 27, with the reel 4 feet in diameter and the rope 4½ in. × ⅜ in.

From Table 1, furnished by wire rope makers, such a rope would weigh about 50 per cent. more than a round rope of equal strength.

The diagram, Fig. 27, shows that the static moment for the whole lift of 2,000 feet varies from 28,000 pounds to 45,000

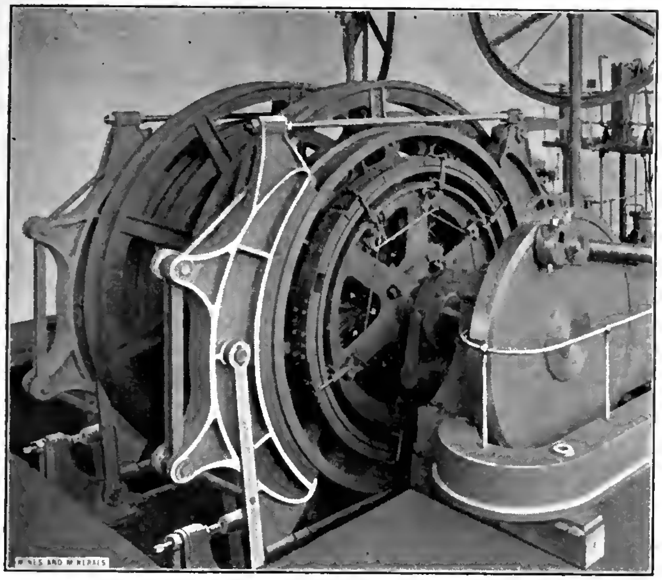


FIG. 29. HOIST WITH TWO LOOSE REELS AND FRICTION CLUTCH

pounds, and terminates at about 42,500 pounds. It will be noticed from the same diagram that these static moments are less than in the case of the cone and straight drums, and that the engine may have, therefore, smaller cylinders, in fact 2 inches less than the cone-drum engine and 4 inches less than

TABLE 1. COMPARATIVE WEIGHTS AND STRENGTHS OF CAST-STEEL ROPE

Size in Inches		Weight Per Foot-Pound		Approximate Breaking Stress, Tons of 2,000 Pounds	
Round	Flat	Round	Flat	Round	Flat
¾	2 × ⅜	.96	1.35	21	20.00
	2½ × ⅜		1.70		25.00
⅞	3 × ⅜	1.25	2.05	29	30.00
	3½ × ⅜		2.40		35.00
1	4 × ⅜	1.70	2.75	38	40.00
1⅛	5 × ⅜	2.15	3.45	47	50.00
1¼	6 × ⅜	2.50	4.15	56	60.00
1	3 × ½	1.70	2.40	38	37.50
	3½ × ½		2.85		43.75
1⅛	4 × ½	2.15	3.30	47	50.00
	5 × ½		4.20		62.50
1⅜	6 × ½	3.17	5.10	69	75.00
1½	7 × ½	4.71	6.00	81	87.50
1¾	8 × ½	5.00	6.90	109	100.00

the straight-drum engine. While this looks as if there would be an economy in steam, the reverse is true, owing to the flat-rope engine making 90.8 revolutions, while the cone and straight-drum engines make but 64.4 revolutions. If it be assumed that the load is raised in 1 minute, the average horsepower of the cone and straight-drum engines would be 704, and that of the flat-rope engine would be 781.* It is evident from this that a waste of power occurs when hoisting out of balance with a flat rope and reel. However, the diagram shows that when hoisting in balance the horsepower needed for the flat rope is greatly reduced, in fact is 267 for a perfect balance.

Hoisting engines with flat ropes and reels, built so as to run either balanced or unbalanced, generally have two loose reels that are driven by clutches. Sederholm, in the article mentioned, says: "The general belief is that when running in balance the effect is good; however, except when hoisting from the most advantageous level, say the 1,500-foot, the static moments vary with every level, and with badly proportioned hoists the effect may be a negative quantity." Suppose for example a hoist is needed for the conditions assumed. In this case the descending cage with empty car materially assists in lifting the ascending car. If the relative positions of the reels are such that one cage is at the bottom when the other is at the top, then the line starting from the bottom of the diagram represents the static moments. If the reels are shifted from the 1,500-, 1,000-, and 500-foot levels the balancing effect is quite different. When hoisting from the 1,500-foot level the balancing effect is nearly perfect and the static moments practically constant. The diagram also shows that when hoisting from the shaft bottom, less effort is required to start the load, but that the effort increases all the way to the surface, so that the relative moments are as 1 to 3.67, and in addition a hoisting engine of this kind is expected to lift the loaded cage unbalanced, which operation requires an effort 7.4 times the minimum effort. Under such conditions good economy is out of the question; nevertheless a reel proportioned on these lines will give the best average results when hoisting must be done from different levels of a shaft. The double-reel hoist, shown in Fig. 28, was built for the Anaconda Mining Co., of Butte, Mont. It has first-motion engines with cylinders 30 in. × 72 in., constructed to hoist four-decked cages from a depth of 3,000 feet. Another reel hoist used at the Copper Queen Mine, Bisbee, Ariz., is shown in Fig. 29. This hoist has two loose reels which are thrown in and out of gear by the friction clutch shown on the crank-shaft.

The reel and flat rope are appreciated by metal miners owing to the hoister taking up comparatively little room in the engine house. The reel is much lighter than a drum, and where the machinery must be packed some distance over a rough country or into almost inaccessible mountain districts, this is also appreciated. The greatest depth to which flat ropes have been used in the United States is probably 3,000 feet, at the Anaconda copper mines. It is also probable that owing to the great weight of the rope, which will necessitate an increase in the size of the engine beyond 3,000 feet in depth, it will make

FIG. 30. WATT HOISTING ENGINE AND REEL FOR FLAT ROPE, USED IN CUBA

it more economical to use some other kind of hoister. Edgar G. Tuttle, E. M., who furnished Fig. 30, states that there is an old Watt engine with reel and 1,000 feet of flat rope wound on it at the Cobre copper mines in Cuba. These mines were put out of working order by Cuban insurgents about the year 1868.

Flat ropes are not a new invention, in fact one of the earliest hoisting engines, shown in Fig. 16 of this article, has a reel† and flat rope, probably hemp.

* E. T. Sederholm, Vol. XXIV, page 579, MINES AND MINERALS.

† MINES AND MINERALS, November, 1910.

When a rope winds off and on a drum it makes an angle with the groove in the head-sheave that creates considerable friction. To overcome this the engine is placed at considerable distance back from the head-frame, which position has a number of disadvantages. In order to overcome this angling, the inertia due to heavy drums driven at a high rate of speed, and to decrease the load on the engine, a system of hoisting was invented by Herr Koepe.

The system known as the Koepe, has been extensively adopted on the continent of Europe; to a limited extent in Great Britain; but not at all in the United States, except in a modified form. To replace the drum and obtain sufficient frictional resistance a large grooved wheel is placed on the engine crank-shaft. The rope passes half around this wheel and over two sheaves on the head-frame, thence to the top of the cages—one being at the bottom of the shaft and the other at the top landing. A balance rope is fastened underneath the platform of each cage. Ordinarily there is sufficient frictional resistance between the hoisting rope and the wheels, if the plant is designed properly, to hoist the loads, but in order to be on the safe side and furnish a good rope bearing, the driving wheel has been made of such size that the advantages in point of lightness have in a measure been lost. Owing also to the chances of the rope slipping on the driving wheel, the load must be started from the bottom gradually and not at full speed as with a drum.

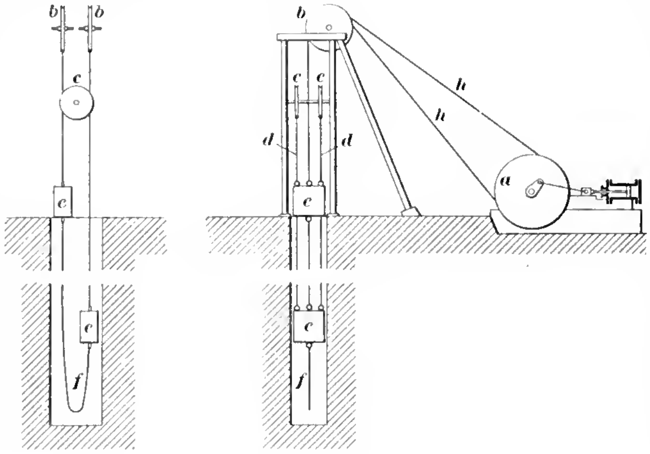


Fig. 31. Modification of the Koepe System

To overcome these objections, and possibly some others, flat ropes have been tried on the pulleys, as they offer more bearing surface; and double drums placed tandem, supplied with round ropes have also been introduced.

With flat ropes there is no overcoiling as in the case of reels, and as the stitching wires are not broken to such extent, less repairs are needed, besides the ropes last longer.

As flat ropes would be in balance in the Koepe system, their weight would not be objectionable after the load was in motion. However, the total weight of the mass to be set in motion may be so great as to favor the use of round ropes. Taken as a whole, the Koepe system of winding with flat-rope reels does not appear to offer advantages over the round-rope system with sheave wheels placed tandem to a double-grooved driving pulley. In the latter case the sheave wheel is mounted on a movable carriage, so weighted as to take up any stretch in the rope. One end of the rope is attached to one of the cages and passes over one of the head-sheaves to the underside of the grooved driving wheel and over one of the grooves on the idler sheave on the balance car, then back to the driving wheel where it passes under and over to the second head-sheave and hence down to the other cage. Fig. 32 shows the Koepe system of winding where two head-sheaves are placed side by side, Fig. 33 shows a method where the head-sheaves are arranged tandem. In the first case the head-sheaves are not in direct line with the driving pulley, while in the latter case they are in such a direct line that there is no side friction due to lateral movement between the moving rope and driving pulley.

One modified application to the system is shown in Fig. 31, in which *a* is the engine pulley, *b* the head-sheaves, *c* the pulley wheels for safety ropes, *d* the safety ropes, *e* the cages, *f* the balance rope, and *h* the hoisting rope. The object of placing the head-sheaves tandem and the pulley wheels *c* at right angles to the head-sheaves when placed side by side as in Fig. 31, is to keep the cages in line with the head-sheaves and prevent the hoisting and tail-ropes interfering. One objection to the Koepe system is that should the rope break both cages would fall, and the safety ropes *d* are connected with each cage to prevent this sort of accident. Where such additions are made the system loses its simplicity, the balance rope must be made heavier, and additional power is required to set the masses in motion.

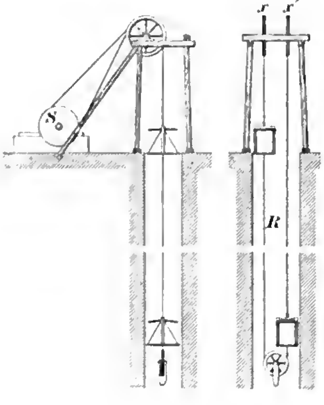


Fig. 32. Koepe System

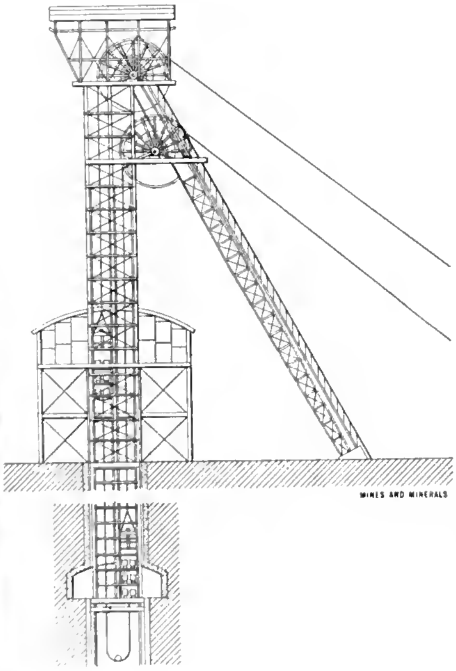


Fig. 33. Koepe System with Tandem Drums

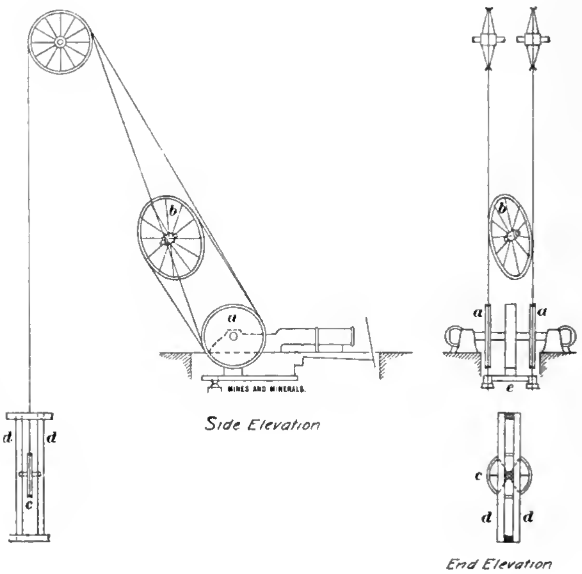


Fig. 34. The Poore System

The Poore system, adopted for a short time at the Pettibone shaft, near Wilkes-Barre, and at the Neilson shaft, near Shamokin, Pa., is a modification of the Koepe system, as shown in Fig. 34.

This system used two driving pulleys *a*, instead of one, on the engine shaft; a transfer wheel *b* so placed and mounted on a movable truck as to carry the rope from one driving wheel to the other in tangential lines; a tail-rope guide sheave *c* hanging in the bight of the rope and held in position by bearings which travel on guides *d*.

The Neilson shaft, at which the Poore system was installed, obtained most of its coal from the 900-foot level instead of the lowest, or 1,278-foot level,* and to hasten operations all loads were hoisted on one cage.

"Out of this developed the only defect of moment in the system, and this in fact was due to the engine always starting the load in the same direction, so that there was a tendency of the rope wheels to slip ahead before the inertia of the rope and loaded cage was overcome. This resulted in the indicator showing the cage as having reached the landing before it had done so. The amount of slip was quite variable with no apparent reason for the change in rate."

Ihlseng† states "that in a shaft 1,260 feet deep a saving of 11 per cent. in power is effected by the Koepe system." "With sheaves 8½ feet in diameter on an 8-inch axle; main rope 1.75 pounds per foot; tail-rope 1.55 pounds per foot; cage and car 2,100 pounds; load 2,750 pounds; depth of shaft 2,500 feet; and velocity of hoisting 2,500 feet per minute, 250 horsepower is saved over that required to operate the same loads by another system."

In 1862, S. B. Whiting, Chief Engineer of the Reading Coal and Iron Co., invented a system of hoisting applicable to deep shafts, but owing to the prejudice against adopting an unknown plan, it was many years before the system was acknowledged. When it became almost imperative to adopt some system which would economize in power at deep shafts, the Whiting system was resurrected and plants installed in three copper mines of Michigan, and two gold mines in South Africa. There is no fear of the rope slipping in the Whiting system and it is possible to arrange matters so that hoisting may be done from several levels, although an auxiliary hoist should be used for upper levels.

There is a very large hoisting plant of this kind at the Red Jacket Mine of the Calumet & Hecla Mining Co., at Keweenaw Point, Mich. This hoist has two hoisting drums 19 feet 1½ inches in diameter, capable of using a rope 2¼ inches in diameter. It is designed to hoist 10 tons from a depth of 4,900 feet. The cages and cars are in balance and a tail-rope of smaller size than the hoisting rope is in use. The plant installed at the Rand Mining Co., Ltd., South Africa, is shown in Fig. 35. The driving rods of this duplex, tandem-compound, Corliss engine are connected to the crank of the first drum, while the first and second drum cranks are joined with connecting-rods as shown in the illustration. The coupling of the engine to the drums is similar to the coupling of a locomotive piston to the driving wheels. The general arrangement of the Whiting system is shown in plan and elevation in Fig. 36.

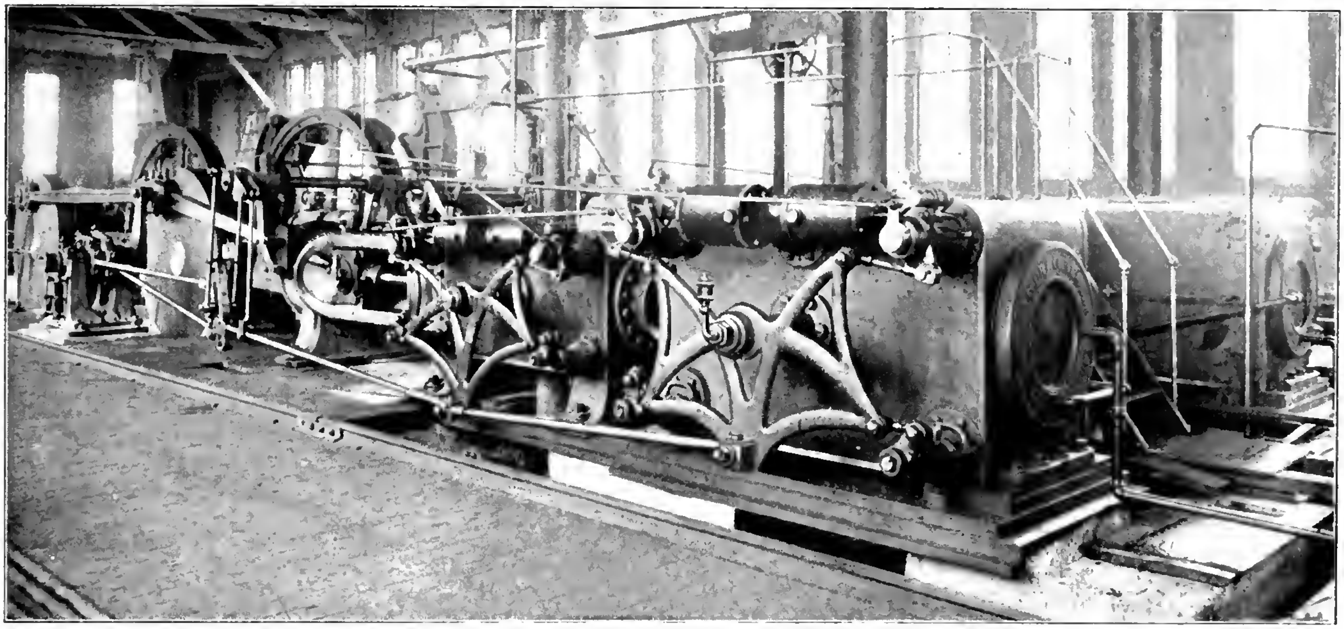


Fig. 35. Whiting Hoisting Engine at Rand Mine, Ltd.

Judging from this experience, the tendency to slip places the Poore system in the same class as the Koepe system, concerning which the following disadvantages are enumerated: In case the winding rope breaks, both cages fall down the shaft; difficulty is experienced in socketing the hoisting rope; extra strain comes on the rope fastenings, due to the weight of the balance rope; the danger of one rope fastening failing, with consequent accident to both cages, and this danger is increased by the lack of cage rests at the top landing. However, should cage rests be used at the surface the rope loses that amount of friction due to the weight of the cage and its balance rope, and may cause trouble when the latches are released. Koepe's system is not suitable for shallow quick winding, for if full steam be quickly put on the engine at the start, the driving wheel slips around like a locomotive wheel at starting; further, once the full speed of the engine is attained it cannot be stopped quickly without a tendency to slip. At the Neilson shaft the speed of hoisting is said to have reached 30 feet per second. However, the system is better adapted for hoisting heavy weights at slow speed from the bottom of deep shafts than where other conditions prevail.

The rope it will be noticed starts from the top of cage *A*, goes over the head-sheave *S*, down to the guide wheel *T*, from underneath which it goes to the under side of drum *M*, where it is given one-half turn to drum *F*. The rope is given three half turns over these drums, and passing off from underneath *F*, goes to the tension car *C*, then half around this, back to and under another guide pulley up to the second sheave on the head-frame and down to cage *B*.

* Arthur H. Storrs in Vol. 24, page 609, Mines and Minerals.

† Manual of Mining, page 147.

The drums M F would be on parallel shafts were it not that the drum F is given a slight inclination so that the rope shall run direct from the groove of one drum to that of the other without binding or friction. Special arrangements are made so that the connecting-rods to this drum shall run in line with the cranks of drum M.

The object of the tension car C, shown also in Fig. 37, is to permit hoisting from different levels. The first dotted circle represents one level, the second circle the present situation of the car another level, and the third circle another level. It is obvious that one cage goes down the shaft when the other is lifted, and that by changing the position of the tension car, one cage is raised or lowered without affecting the position of the other cage, because the length of rope on one cage has not been changed. Where there are a number of levels to be worked simultaneously, it is necessary to change the position of tension car with some haste, for which reason it is mounted on a track and moved by a strong engine of the geared type. When the car is in position it is clamped to the track. In using the Whiting system a tail-rope is useful both for balancing and safety, as it is a kind of guarantee that the cage or skip will move down the shaft whenever it is desired to lower it from the top landing. During sinking operations tail-ropes are impracticable and auxiliary hoists should be used to hoist at least to the lower level.

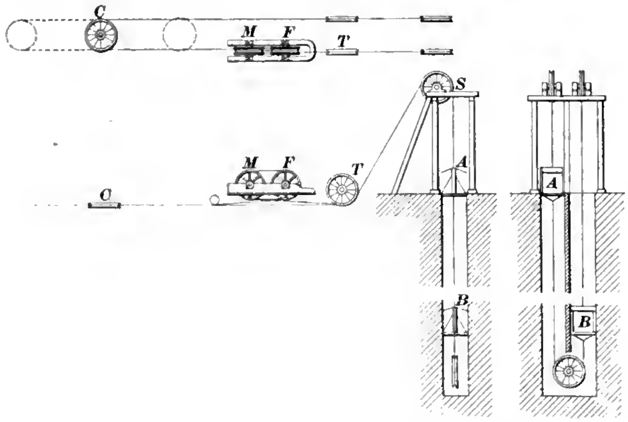


FIG. 36. PLAN AND ELEVATION OF WHITING SYSTEM

In either the Koepe or the Whiting systems, hoisting depends upon the frictional resistance, and this depends on the tension of the rope, the coefficient of friction, and arc of contact.

Let T_l = the tension on the loaded rope;

T_u = the tension on the unloaded rope;

f = coefficient of friction;

θ = arc of contact;

e = .434, the modulus of the common logarithm referred to 2.718, the base of the Naperian logarithm.

Then the relation of the various factors to each other may be expressed by the formula $\frac{T_l}{T_u} = e f \theta$, or $\log \frac{T_l}{T_u} = .434 f \theta$, if θ is in circular measure = degrees × .01745; or $\log \frac{T_l}{T_u} = .007578 f \theta$ if θ is in degrees. Take, for example, a load of 8,000 pounds to be raised from a depth of 5,000 feet, that needs a steel rope $1\frac{1}{2}$ inches in diameter. The rope will weigh 18,250 pounds and the skip it is assumed weighs 5,600, hence, $T_l = 31{,}850$ and $T_u = 23{,}850$. $\frac{T_l}{T_u} = \frac{31{,}850}{23{,}850} = 1.33$, and log 1.33 = .123852. Assuming the arc of contact to be 180 degrees, although this contact could only occur when the sheave were directly over the hoisting drum, 180° × .01745 = 3.14, and 3.14 × .434 = 1.36276. The coefficient required to prevent slipping will be, therefore,

$$f = \frac{.123852}{1.36276} = .09.$$

The coefficient of friction for a wire rope running on a cast-iron pulley may be taken as .1, and for a wire rope running on a grooved pulley lined with wood or leather, as .2. In order to obviate slipping $\frac{T_l}{T_2}$ must be less than the constants given, and where this condition is not fulfilled either the ratio $\frac{T_l}{T_2}$ must be

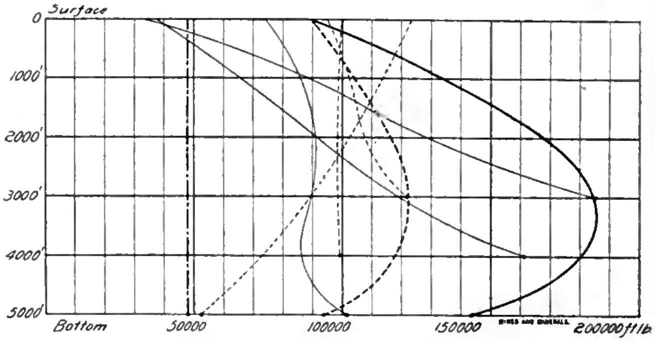
FIG. 37. TENSION CAR, WHITING SYSTEM

reduced by increasing the weights on one or both ropes hanging in the shaft, or the factors f or θ must be increased, usually the latter, as where in the Whiting system the rope is passed several times over the two pulleys. It may be noted here that flat ropes, owing to their greater weight, increase the friction on the pulley. In the Whiting case grooved pulleys are used and $\theta = \Phi n$ in which n = the number of grooves in the pulleys when both are driven and Φ the arc of contact of one groove, or 180 degrees.

If* it were attempted to hoist in balance on straight drums without a tail-rope, the engine would have to overcome both the weight of the ore and 18,250 pounds of rope, and the drums would be 18 feet in diameter in order to keep the length within reasonable bounds. The advantage of the Whiting system compared with conical drums is shown in Fig. 38.

DIAGRAM OF STATIC MOMENTS FOR THE FOLLOWING CONDITIONS

Ore .... 8,000 pounds
Skip, and rope from head gear to skip .... 5,600 pounds
Plain rope .... $1\frac{1}{2}$ inch diameter
Taper rope .... $1\frac{1}{4}$ inch diameter at top
Taper rope .... 1 inch diameter at bottom
Conical drum .... 12 feet diameter at small end
Conical drum .... 32 feet diameter at large end
Whiting drums .... 12 feet



Conical drum, plain rope, hoisting singly, empty, to change levels
Conical drum, taper rope, hoisting singly, empty, to change levels
Conical drum, plain rope, hoisting in balance, loaded, from different levels
Conical drum, taper rope, hoisting in balance, loaded, from different levels
Whiting drums, plain rope, hoisting in balance, with tail rope, loaded

FIG. 38

Deep hoisting and electric hoisters will conclude this article.

(*To be continued*)

*E. T. Sederholm, Vol. XXIV, page 581, MINES AND MINERALS.

HYDROMETALLURGY OF COBALT ORES

Written for Mines and Minerals, by E. B. W.

Different Methods of Treatment As Illustrated by Flow Sheets of the Principal Mills

Probably the town of Cobalt shows to better advantage the benefits to be derived from the metal-mining industry than any other town in Canada east. What was a wilderness in the spring of 1903 is now a thriving place with most of the advantages found in cities. Indirectly this is due to the construction of a railway, and in this respect Cobalt was again fortunate, for usually a mining town must prove its worth prior to its receiving railway assistance. When grading for the construction of the Temiskaming and Northern Ontario Railway, silver, associated with cobalt, nickel, and arsenic, was discovered. The first application for a mining claim in this district was made in August, 1903, by J. H. McKinley and Earnest F. Darragh, who at that time were subcontractors on the Temiskaming railway.

The second application for a mining claim was made by Fred LaRose and was dated September 29, 1903. The third discovery was made by Tom Herbert, who on October 21, 1903, located the first vein of ore on the property now owned by the Nipissing Mining Co. In November, 1903, Neil King discovered what is known now as the O'Brien property. The principal factor in attracting outside capital to the Cobalt district was W. G. Trethewey's discovery of the Trethewey and Coniagas claims in May, 1904. Shortly afterwards Alexander Longwell located the Buffalo Mine. These continued rich finds gave credence to the number and stability of the deposits. Until the spring of 1905 little attention was given the Cobalt district and then the Tretheway shipment of approximately $208,000 from an 8-inch vein caused people to take notice, and in August, 1906, the real boom commenced, caused by the Nipissing declaring a 3-per-cent. quarterly dividend on a capitalization of $6,000,000. Up to January 15, 1910, this company has declared dividends equaling 64 per cent. on its capital stock. The Little Silver vein, located by Mr. Herbert, in 1903, is shown in Fig. 1.

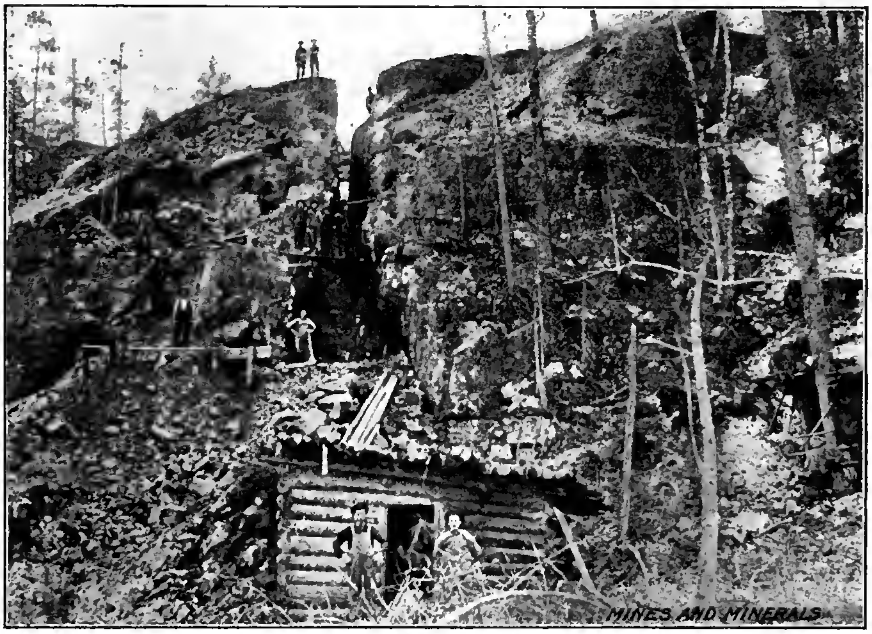


Fig. 1. The Little Silver Vein, Cobalt

The town of Cobalt is in Colman Township, in the province of Ontario, Canada, 10 miles from the Quebec line; 330 miles north of Toronto; and about 103 miles from North Bay Junction on the Canadian Pacific Railway.

For 200 years previous to the construction of the Temiskaming railway, white men have been traveling this country to and from Hudson Bay both by land and water. Lake Temiskaming, which is about 90 miles long, and from 1 to 5 miles wide, is but a short distance from Cobalt. It forms in part the division line between Quebec and Ontario, and Coleman Township, is on its western shore about half-way up the lake. Since the discovery of silver at Cobalt, the country for many miles around it has been prospected with an intelligence that has extended the known mineral area as far south as the Gilles Limit, and to Hudson Bay on the north. While all the ground has not been thoroughly covered, a number of silver, gold, lead, and copper deposits have been discovered.

The native silver in the Cobalt veins occurs in flat, thin sheets in vein rock, in nuggets, and occasionally in slabs up to 1 inch thick. In order to break the ore with as small waste as possible and at the same time work to advantage, an excavation about 4 feet wide is made on one side of the vein, and the ore broken down with light charges of powder placed in shallow holes. The best ore is assorted by hand, sacked inside the mine, and when sufficient has accumulated at the ore house on the surface, is shipped to some smelter. The remainder of the ore is hoisted in bulk, concentrated by hand cobbing, or by machinery and the concentrates shipped to the smelters. That ore which is third class and which may carry from 25 to 100 ounces of silver per ton is given hydrometallurgical treatment which reduces the bulk of ore to such an extent it carries up to 4,000 ounces of silver per ton. In the course of concentration an average of 12 per cent. of the silver is lost unless the cyanide process is introduced to treat the tailing. Cobalt ores, because of their complexity, are not easy to smelt economically, because unless carefully mixed they form matte, spiess, and accretions that must be retreated. In early days the ore was reduced to metal by spreading from 1,000 to 1,500 pounds over a lead bath. This volatilized some of the impurities although a large quantity of spiess was formed that had to be re-treated. At present the ore is roasted to volatilize the arsenic, antimony, and sulphur, after which it is smelted in the lead blast furnace.

To roast the ore so as to eliminate more than 10 per cent. of arsenic is not economical, as various complications and extra expense will follow, although the smelters as a rule penalize shippers for ore that carries over 5 per cent. arsenic. There are excellent reasons why they attach this penalty: first, they must oxidize the ore, which costs money, and, secondly, the arsenic in volatilizing in the furnace carries away part of the silver and other metals.

Those Cobalt ores which are too low grade to ship are hoisted from the mines and sorted on shaking tables which are sprayed with water to clean the ore. The coarsest of the ore is picked out by hand. In some cases the ore is screened and washed in a trommel and the fine material which settles out of the wash water is shipped to the smelter, as it carries from 200 to 500 ounces in silver. In concentrating by means of water there is a loss of from 10 per cent. to 20 per cent., making it advisable to keep the high-grade ore separated from the lower grades. The screenings from the picking tables, which are over 100 ounces of silver per ton, are shipped generally to the smelter, although recent practice has indicated the advisability of concentrating ore running as high as 125 ounces. All ore, however, which is considered too low in silver to ship, that is from 75 ounces down, is saved, and either a concentrating mill is constructed after enough such ore is accumulated to

warrant its construction, or else it is sent to a custom concentrating mill. Where there is a mill in connection with the mine, the ore is hoisted direct to the mill bins or crusher floor after cobbing in the mines. Some favor cobbing in the mines, but it is probably better to cob in daylight.

There are two distinct kinds of low-grade ore. One in which the silver mineral is disseminated through the rock, and

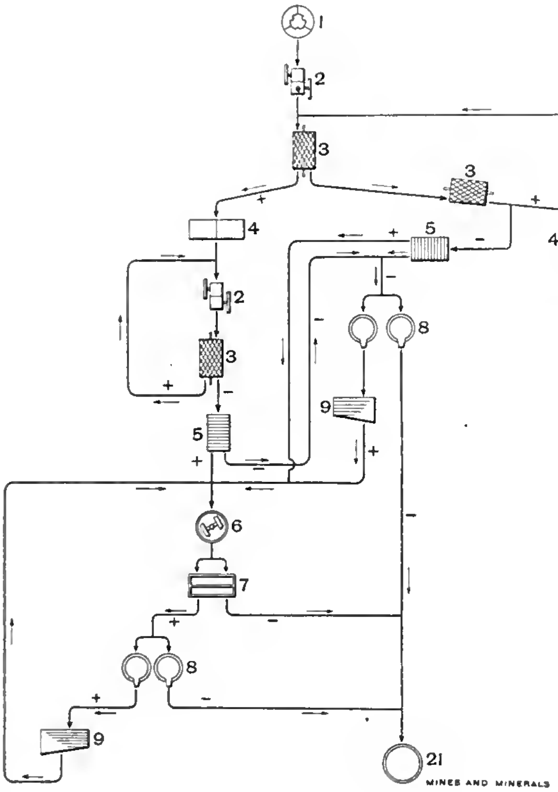


FIG. 2. BUFFALO MILL

the other in which the ore is mixed with vein material and wall rock. The silver in the wall rock is so irregular it makes a uniform low-grade ore impossible, a condition which adds somewhat to the difficulties of water concentration. The rock which contains the disseminated silver mineral demands fine crushing, while the mixed vein matter and wall rock are treated by coarse crushing and jigging. The calcite of the vein and the diabase walls adjacent to the vein usually contain wires of smaltite and native silver.

The object of concentrating low-grade ores is to reduce their bulk and separate the valuable from the worthless mineral. By this means it is possible to obtain a valuable ore for shipment to the smelter. There are 13 hydrometallurgical mills in the Cobalt district and the methods of treatment at the different mills are shown by the flow sheets appended, the legend for which is as follows: 1, crusher; 2, rolls; 3, trommel; 4, jigs; 5, impact screen; 6, Chilian mill; 7, settling box; 8, classifiers; 9, concentrating tables of various makes; 10, Hardinge conical mill; 11, stamp batteries; 12, pulp thickeners; 13, canvas tables; 14, dewatering trommel; 15, Frue vanners; 16, tube mill; 17, Spitzkasten; 18, shaking screen; 19, waste; 20, Richards jigs; 21, cyanide tank; 22, agitators; 23, precipitation tank; 24, filter press; 25, Moore filter.

Fig. 2 is a flow sheet of the Buffalo mill, which originally was of 60 tons daily capacity, was erected in September, 1907. A cyanide plant was installed later that went into operation in August, 1909. The average recovery with this concentrating mill was 86.4 per cent. of the silver in the ore, and with the cyanide process in operation, 91 per cent. The average value of the ore being treated is 45 ounces of silver per ton, and the ratio of concentration is 45 tons of ore to make one of concentrates. About 125 tons of ore are treated daily. From the flow sheet it will be noticed that this mill has three pairs of rolls 16 by 36 inches. The middling from the seven concentrating tables is reground with a Chilian mill, while the tailing from the classifiers is cyanided.

Fig. 3 is the flow sheet of the Cobalt Central mill, which treats about 100 tons of ore per day, and produces 75,000 ounces of silver in the form of concentrates in the course of a month. The recovery is said to be 88 per cent. with a concentration ratio of 50 to 1. The average value of the concentrates is 2,000 to 3,000 ounces per ton. It is stated that it is the intention of the company to install a cyanide plant. By reference to the flow sheet it will be seen that this mill has three pairs of rolls, one coarse jig and two fine jigs; two trommels, one with ½-inch mesh and the other with ⅛-inch mesh. All material passes through a classifier or Hardinge conical mill before going to the tables. The overflow from the last of the classifiers, which in this case are Callow settling tanks, goes to slime tables. In the Cobalt district concentration runs as high as 75 tons of material to one of concentrates and possibly higher, for which reason no side discharges are used on the plunger jigs. Once or twice, possibly oftener, in each shift the jig is skimmed by hand. The smaltite which is softer than the rock wears away rapidly and jig bed builds up slowly. Smaltite also breaks with sharp corners and causes trouble in fine plunger jigs owing to its clogging the screens, in fact it was so bad at one mill, the mill-

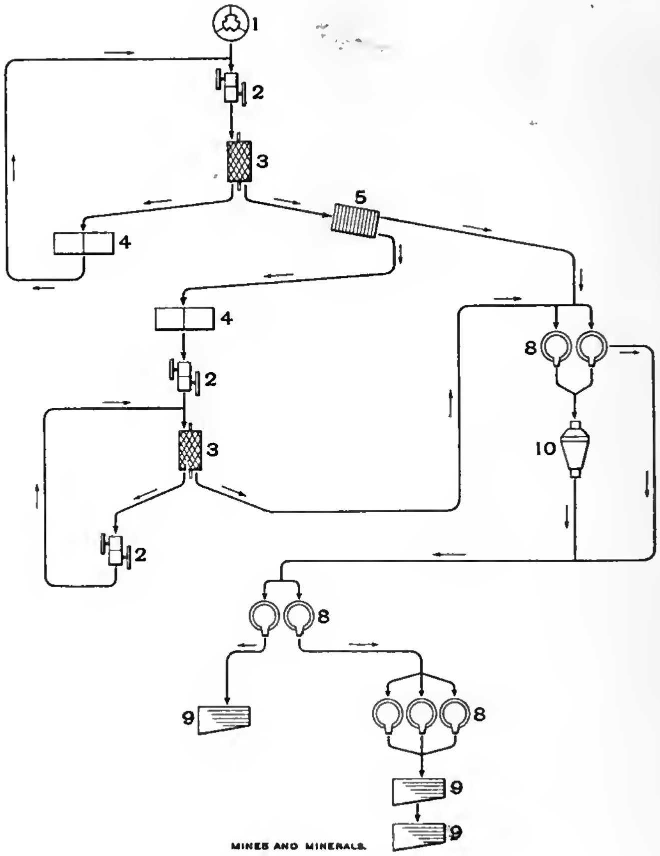


FIG. 3. COBALT CENTRAL MILL

man gave up trying to jig through ¼-inch or 12-mesh screen and sent the product directly to the regrinding machines, and hence to the tables. Pulsator jigs are liked in the Cobalt district, as the screen does not blind and the capacity of the jig is large for the screen surface, and permits the use of an automatic side discharge which results in less loss from smaltite tatrition.

Fig. 4 is a flow sheet of the Colonial mill which belongs to the Colonial Silver mines. This mill is of comparatively recent construction and not so much information is to be had concerning it. According to advices, this company has a large tonnage of low-grade ore and it was for the purpose of concentrating this ore that the mill was constructed. As will be observed, the fine material from the roll goes to classifiers, while the oversize from the jigs goes to the stamp mill and then to classifiers. The middling from the concentrating tables is reground in a tube mill. There are five concentrating tables and 10 stamps in this mill. In the Cobalt district there are three types of recrushing apparatus used for jig tailing. The stamp is most generally adopted, with the tube mill growing in favor as a recrusher. The Chilian mill is used but little and is not to be recommended for regrinding this particular ore. The middlings after leaving the concentrating tables are sent to tube mills. In some cases the material from the tube mills flows to classifiers and then to slime tables. In other cases it is cyanided.

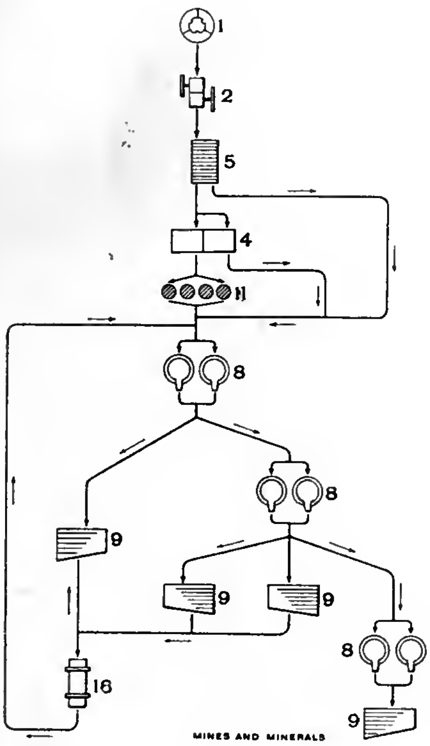


FIG. 4. FLOW SHEET COLONIAL MILL

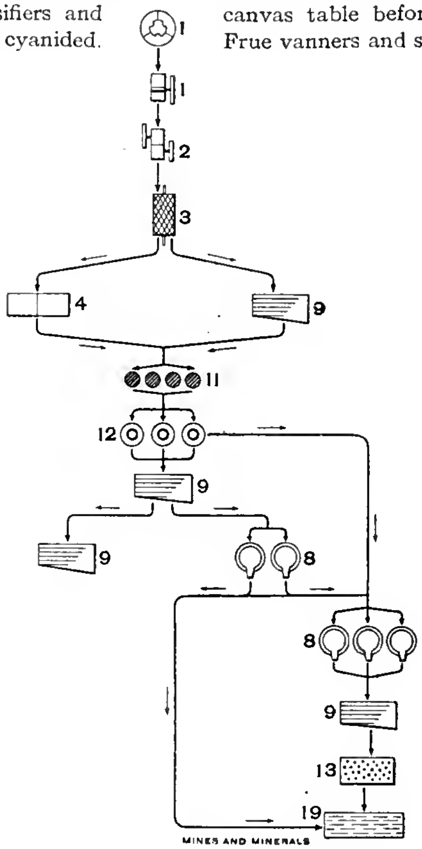


FIG. 5. CONIAGAS MILL

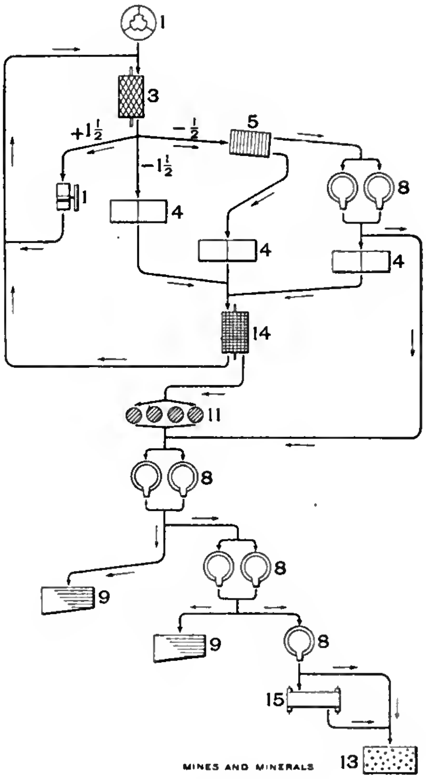


FIG. 6. KING EDWARD MILL

In a number of mills, after leaving the slime tables, the tailing flows over canvas tables before going to waste. In some of the mills the stamps are of California type and weigh 1,250 pounds; in other mills they are of the Nissen type with circular mortar and weigh 1,500 pounds.

Fig. 5 shows the Coniagas concentrating mill; the property of the Coniagas Mine Co., which also owns the issued stock of the Coniagas Reduction Co., Ltd. The smelters of this company, as previously stated, are at Thorold, where they handle the product of the mine and are producing refined silver, refined white arsenic, and expect to produce refined cobalt oxide and nickel oxide. The Coniagas mill has 30 stamps with a capacity of 80 to 90 tons of ore per day. Thirty additional stamps have probably been installed at this writing, which it is supposed will increase the capacity to 160 tons per day. The ore sent to the mill averages between 80 and 100 ounces of silver to the ton and a reduction of about 40 to 1 is made, and an extraction of 80 to 85 per cent. At this mill there is a 10″×16″ Blake crusher which delivers 4-inch diameter material to a gyratory crusher that further reduces the material to 1½-inch diameter. The rolls are 30 in.×30 in. and are said to crush to ¾-inch. The middling from the six jigs and one table goes to the stamps, and after the pulp has been thickened it goes to seven tables. The middling goes to classifiers which separate the slime from the sand. After settling the slime goes to slime tables; the tailing from these to canvas tables, and then to waste. While this mill is putting in canvas tables for slime treatment, it would seem that the more logical treatment would be cyaniding, for the reason that in the latter case the end product is bullion, where the table concentrate is low-grade and entails high smelting cost for treatment.

Fig. 6 is the King Edward Mill, which is one of the best arranged so far mentioned. The discard from the coarse jig is the only material returned for a second treatment. The remainder goes to the 10-stamp mill, and, after classifying, to its proper concentrator. The tailing from the Frue vanner flows to a canvas table before going to waste. The concentrates from Frue vanners and slime tables average about 900 ounces of silver to the ton, and so far without cyaniding it is probable that the average loss from milling is at least 12 per cent. of the silver in the ore.

Fig. 7 is a flow sheet of the McKinley-Darragh mill, which has a capacity of 120 tons per day, with a ratio of concentration of 30 to 1. The recovery is 82.71 per cent. of the silver. One of the two tube mills is 5 feet in diameter and 20 feet long; the other is an 8-foot Hardinge conical mill. There are 30 stamps, 13 slime tables and canvas tables before the tailing is wasted. The fine pulp from the first classifiers goes to four Frue vanners, but the recoveries are generally low and the concentrate of low-grade in these vanners.

Jig concentrates at Cobalt commonly carry from 1,000 to 4,000 ounces of silver per ton. The table concentrates from 1,000 to 1,800 ounces per ton. In some instances the tables concentrate 120 into 1, making 1,500 ounces of concentrate from a feed that carries less than 10 ounces of silver per ton.

No. 8 is the Nipissing reduction mill. This is a custom mill built on the Nipissing ground and has been concentrating

the low-grade ore in the Kendall dump. At this mill is introduced a shaking screen which makes two separations. Anything over 20 mesh goes to the jigs, all the discard from them going to Hardinge conical mill. All 20 mesh and under goes to classifiers, then to tables, the tailing going to waste and the middling to the Hardinge mill. By the arrangement of this mill no unnecessary material is rehandled. The capacity of the mill is said to be 80 tons per day. At the shafts of the Nipissing mines bumping tables separate the ore into three classes. The first class averages 2,000 ounces of silver to the ton; the screening averages about 100 ounces of silver to the ton; the discard averages from 15 to 20 ounces of silver to the ton. By the aid of recently improved tables the screenings are subdivided. The high-grade screenings that run over 100 ounces per ton are sacked and sent direct to the smelter. The discards and the low-grade screenings go to the custom mill for treatment. The is dried and sacked for shipment. The average assay of the ore treated is about 25 ounces of silver to the ton. The concentrates average about 1,000 ounces of silver to the ton, or a ratio reduction of 44 to 1. The mill averages an extraction of about 88 per cent. The tariff charged by this custom concentrator is about as follows:

On ore yielding less than 20 ounces of silver per ton crushed, 10 ounces of the silver are retained and the balance turned over to the mining company with 50 per cent. of other metals that can be sold. Ore running from 20 to 35 ounces of silver per ton pays 50 per cent. tariff; 35 to 50 ounces of silver pays 55 per cent. tariff; 50 to 70 ounces of silver pays 60 per cent. tariff; 70 to 90 ounces of silver pays 65 per cent. tariff; 90 to 110 ounces of silver pays 70 per cent. tariff; 110 to 125 ounces of silver pays 75 per cent. tariff.

Ore is also treated on the tonnage basis of $4 per ton

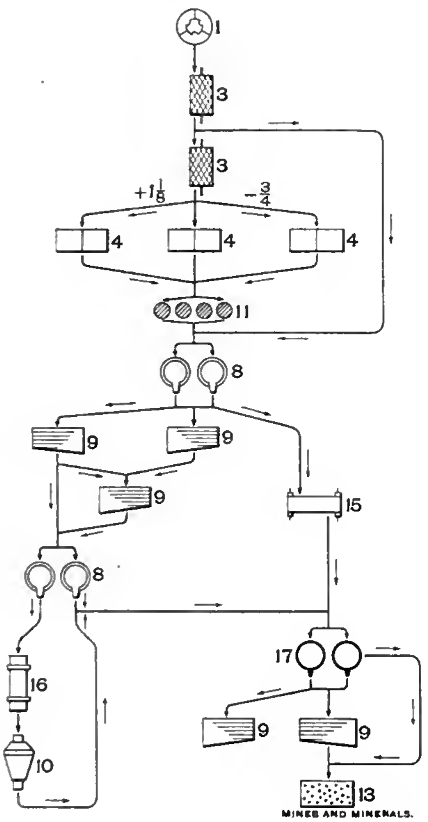


Fig. 7. McKinley-Darragh Mill

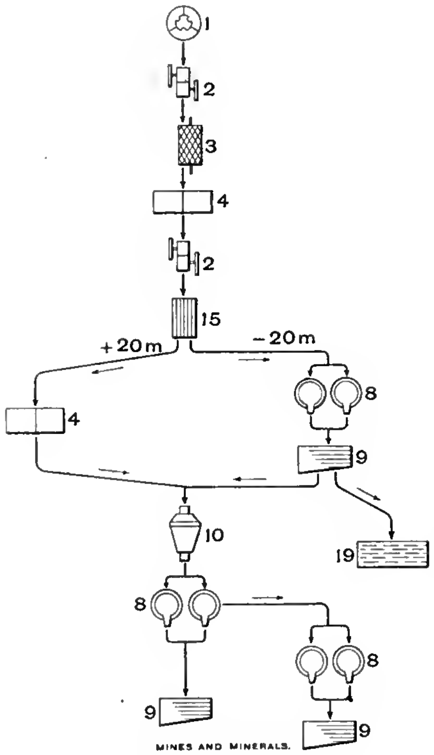


Fig. 8. Nipissing Customs Mill

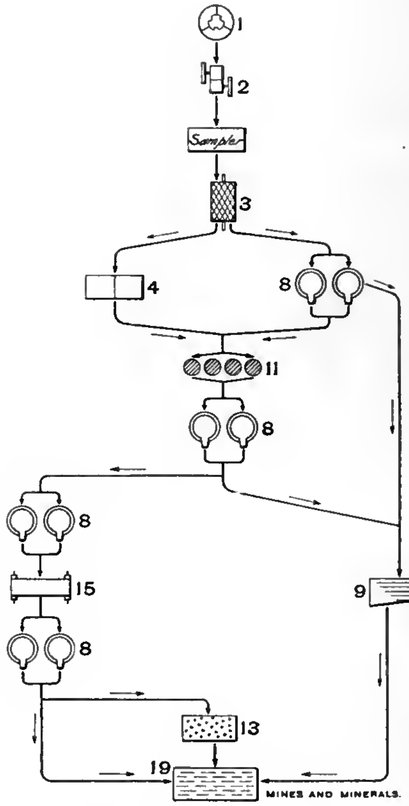


Fig. 9. Northern Customs Mill

actual cost of mining is a small part of the expense in connection with Cobalt operations. The cost of marketing ore alone amounts to 6.3 cents and the total cost of production amounts to 20.7 cents per ounce of silver.

The flow sheet shown in Fig. 9 is that of the Northern Customs Concentrating Co. The capacity of this mill is said to be 100 tons per day. The ore after passing the crusher in rolls is elevated to a Vezin sampler, where 10 per cent. is cut for a sample. The remainder goes through a trommel which screens out the fines, sending them to a hydraulic classifier, and the coarse or oversize to a 24″×40″ jig, where the metallic particles are sent to ore bins for sacking and shipment. The jig concentrate includes small pieces of high-grade ore and heavy silver. The tailing from the jigs and spigot product from the classifier go to the stamps which crush to 30-mesh. The pulp from the stamps is elevated to a Richards classifier and separated into two sizes; the spigot or coarse sand goes to the tables, the overflow to the vanners. The tables and vanners extract the loose particles of metallic silver and the heavy metallic minerals with more or less rock attached. This product delivered at the mill and the concentrate delivered to the owners is sacked, dried, and loaded on the cars. As an example, one milling rock contains 25 ounces of silver and one contains 100 ounces of silver, assume that 100 tons of each is to be milled. One hundred tons of 25-ounce silver would, if there were no loss, furnish 2,500 ounces of silver, but there is a loss of 12 per cent. in concentration or 300 ounces of silver, making 2,200 ounces of silver in $2\frac{1}{4}$ tons of concentrates. With silver at 50 cents an ounce this would be $1,100. The cost of concentrating 100 tons at $4 a ton would be $400, thus leaving balance in favor of the miner of $700, or $7 a ton.

In the second case 100 tons of 100-ounce silver would mean 10,000 ounces of silver. As there is a loss of 12 per cent. in concentrating there would be 8,800 ounces of silver available. This at 50 cents an ounce would amount to about $4,400. Less the cost of concentrating $400, thus netting the operator $4,000. The concentrates are now shipped to the smelter. Assuming it would pay for 94 per cent. of the silver contents, less a treatment charge of $10 per ton, plus $\frac{1}{2}$ cent on each ounce of silver contained. No smelter can afford to treat ore running

as low as 25 ounces, and the cost of freight on a ton of 25 ounces of ore would make the shipment prohibitory. On the other hand, by concentrating the miner obtains a fair return for his trouble. Different custom mills have different schedules but they are not far apart.

Fig. 10 shows the flow sheet of the O'Brien mill which is said to have a capacity of 125 tons per day. At this mill Richards jigs are introduced to work on material under ¾ inch in diameter. Also at this mill cyanide solution is introduced in part of the stamp-mill battery. The stamps are generally run in batteries of five, having separate cam-shafts, and part of them are run on clear water and send their pulp to tables, classifiers, slimers, and vanners. The pulp from the stamps, crushing in cyanide solution, is delivered to two Hardinge conical mills, which recrush the slime for the usual cyanide slime process, consisting of Dorr pulp thickener, Pachuca agitators, Moore filters, zinc precipitation tanks, and a filter press for the precipitate.

Fig. 11 is a flow sheet of the Nova Scotia mill, where there are 20 stamps which crush in cyanide solution, delivering the pulp to tables, then to classifiers and the spigot to tube mills. The overflow from the classifiers is given in cyanide treatment and the precipitates after filter pressing are refined into bullion. The Nova Scotia mill and the O'Brien mill seem to be the only two straight cyanide mills so far in the Cobalt district.

Fig. 12 is a flow sheet of the Silver Cliff concentrator. At this mill straight concentration will be practiced when it is finished. It is designed to handle 100 tons of ore per day. There are no stamps at this mill but there are two sets of 14″×42″ rolls and one 6-foot Chilian mill for a recrusher of the material coming from the jigs.

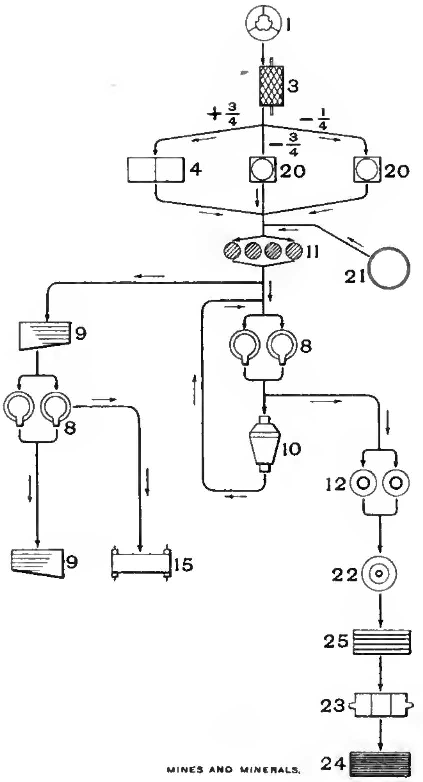


FIG. 10. O'BRIEN MILL

The flow sheet shown in Fig. 13 is that of the Temiskaming mill, designed for a capacity of 100 tons a day with 30 stamps. The mill is to work on ore carrying 50 ounces of silver to the ton and the management thinks that $15 per ton profit is not too much to be expected. It is stated that a Chilian mill will be used at this plant as a recrusher previous to cyaniding the tailing from slime tables.

Fig. 14 is that of the Trethewey mill. This mill has 30 stamps, the sand from which is to go to six tables, the middling to be reground in a tube mill, from which it goes to six slime tables. The capacity of this mill is reckoned at 100 tons with 30 stamps. No arrangements have been made for cyaniding. While the cost of power at Cobalt has been as high as from $150 to $180 per horsepower a year, the Cobalt electric plant on the Montreal River, 10 miles south of Cobalt, is now selling power for $50 per horsepower year. Compressed air is also delivered at Cobalt from the hydraulic air compressor at Ragged Chutes in the Montreal River. This is sold to the mines at about 25 cents per thousand feet, reckoned at atmospheric pressure.

The future of Cobalt, for a number of years to come, is firmly established by the quantity of low-grade ore and the reduction which is being made in the cost of treatment. New veins are being continually found in the district, and at some of the mines for every ton of ore mined 3 tons have been blocked out. By this means dividends are assured for some time to come.

TUNGSTEN ORE IN WASHINGTON

**By August Wolf*

H. E. Schleiff, M. E., of Berlin, Germany, a recognized authority on tungsten, vanadium, and uranium, who has just completed a thorough investigation of the deposits and development work in the three camps in Stevens County, Wash,. located from 40 to 75 miles north of Spokane, declares that within its confines is the largest and most promising tungsten field in the world. He said:

The tungsten district in Stevens County has a chance to overshadow Boulder County, Colo., in the next few years. Boulder County is the country's chief producer, having an output of from 3,000 to 4,000 tons of concentrates a year. There is sufficient ore in the Stevens County deposits to harden steel for all the war guns, all the battleships, and all commercial uses throughout the century, and the profits that will accrue from its production may equal the millions in gold taken from the foremost fields in the country.

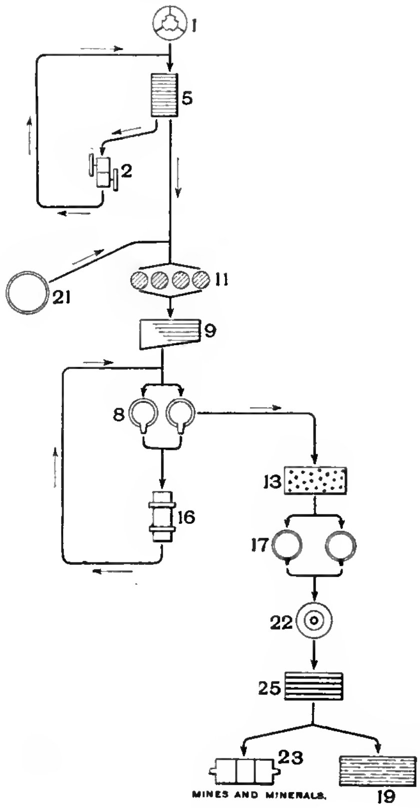


FIG. 11. NOVA SCOTIA MILL

Three camps are under development. One is the Germania in Cedar cañon, 30 miles north of Springdale; the Tungsten King, 8 miles north of Deer Park; and the Blue Grouse, a mile south of the Tungsten King and 8 miles from Loon Lake. Several tons of concentrates have been sent to the market. Prospecting is proceeding at many points in the field, embracing an area of 10 by 40 miles, and corporations are being formed for development on a broad scale.

The Germania, the oldest corporation in the district, has been operating the last 3 years. It has a property of large area on which the tungsten showing is large and promises greater. The country rocks throughout the district are feldspar and hornblende, containing small fragments of mica. The feldspar near Cedar cañon occurs in large pieces of the rock itself, differing in this characteristic from the granite structure in the Loon Lake–Deer Park district, where the feldspar is smaller and the granite contains more mica.

* 225 Hutton Building, Spokane, Wash.

Tunnels driven on one of the big veins in the Germania group have exposed a stoping area of 3,000 tons. Other veins of promise have been exposed by open cuts and short tunnels. Breaks in the main tunnel have caused some inconvenience in the process of development. The characteristic course of the cut-off is toward the foot-wall, pushing the vein 25 feet to the side, after which its regular course is resumed.

The crude ore in the lower levels contains from 3 to 4 per cent. tungstic acid and a little pyrite, while in the upper levels the percentage is from 4 to 6 per cent., with much pyrite. An examination of the dump and slimes suggested that more tungsten is being turned out of the mill than is being saved. The ore loses a great deal of the tungsten because of the imperfect treatment. The company intends to erect a 100-ton mill erect a mill next year and purposes making it the best on the continent.

The concentration of tungsten ore is accomplished with less difficulty in Stevens County than in Colorado. The tungsten ore in Boulder County occurs in fine seams in the quartz, or is impregnated in fine particles of quartz in the vein of the granite, and requires a fine grinding to extract the tungsten. In Stevens County the tungsten is of greater solidity and of higher percentage. It occurs in chunks of various sizes, sporadic in the vein of the granite, and, therefore, is not subject to fine grinding. Crushing the ore to a mesh of $\frac{3}{16}$ inch will produce less slime and make possible a higher saving.

The markets in the United States, Canada, England, Germany, France and other countries demand tungsten running

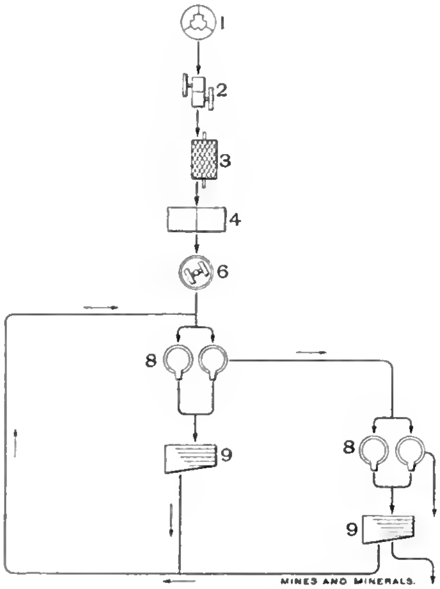


Fig. 12. Silver Cliff Mill

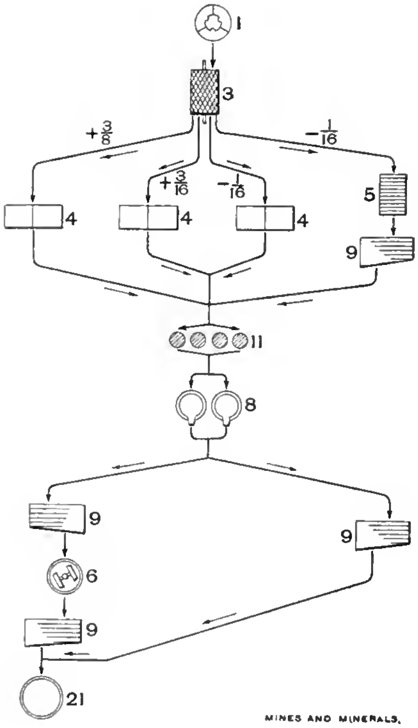


Fig. 13. Temiskaming Mill

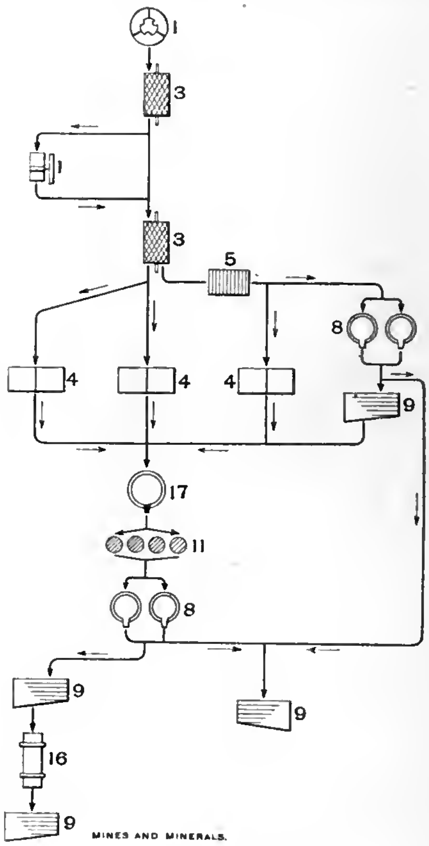


Fig. 14. Trethewey Mill

next year to replace the present 25-ton plant, with the expectation of saving 95 per cent. of the tungstic acid, which is easily possible with special machinery.

The Tungsten King obtains its ore through tunnels driven on several veins. It runs from 6 to 8 per cent., but about 50 per cent. of it is lost by poor treatment. No provision is made to save the slimes, which are bound to occur by grinding any kind of ore, especially tungsten. This part of the product must be saved by machinery constructed especially for that purpose.

A tunnel is being driven on the Blue Grouse property to reach the lead, which crosses the contact between granite and quartzite. The Blue Grouse occupies the granite side of the contact while the Tungsten King has the quartzite side. Enrichment appears to take place on the contact, the tungsten mineral metasomatically replacing the hornblende by the granite.

The Blue Grouse property is in the hands of W. A. Brockway, manager and superintendent, who is one of the few men in the country familiar with tungsten. He gained his knowledge of the metal in Boulder County, Colo. It is said that he will 60 per cent. tungstic acid (WO_3). The desire of many millmen to raise the concentrate to a percentage of from 70 to 72 is absurd. A high percentage of concentrate can be obtained only at the expense of the middle product, which, by the lower percentage of concentrate, can be saved. The experience with tungsten has been that the best results are obtained in an effort to make a 62-per-cent. product.

Because of the great ease and facility of transmission, the telephone has in the past couple of years been making rapid strides into favor among railroad managers. It performs the work formerly done by the telegraph in a more rapid and efficient manner, and the adoption of the telephone standard by a great many of the largest railroads in the country has caused a complete change in dispatching methods.

The smokeless-powder factory of the Mexican Government having been completed, will shortly be inaugurated. Its annual capacity is 110,000 pounds of powder of the best quality.

TIN SLUICING IN TASMANIA

*By Edward Edwards**

The tin fields of Northeastern Tasmania are in the ancient and present river beds tributary to the Ringarooma River. Much of the granite rock which predominates in the Ringarooma watershed is tin bearing, and from this rock the stanniferous alluvial deposits have been derived. The deposits have a length of 13 miles, a width of over a mile and in places at the northern end, have a depth of 230 feet.

Methods of Prospecting and Breaking Down the Ground. Arrangement of Sluices

Figs. 1 and 2 show the granite country rock, how it was eroded to form the ancient river channel, the alluvial drift that filled the channel and was later covered over by a volcanic overflow of basalt, likewise the sections of the present rivers relative to the Briseis and North Home mines. Tin is scattered as cassiterite SnO_2 throughout the alluvial drift, although as a rule, its great weight (specific gravity 6.5–7.1) compared with the other minerals in the granite has caused it to sink to bed rock wherever possible and become more or less concentrated.

The present river beds are worked by dredging; while the ancient river beds are worked by dredging or hydraulicking, and it is this latter class of deposits this paper describes.

For successful hydraulic mining there are five essentials; and neglect to investigate any one may cause failure:

1. The deposit must be capable of being broken down and sluiced by water.
2. There must be a sufficient quantity of water available for breaking down, transporting material, and for all other purposes.
3. The water must be delivered under sufficient pressure to the face of the drift, and wherever hydraulic elevators are needed.
4. There must be dumping ground available for the tailing debris; or a river having sufficient current to remove it out of the way.
5. The overburden must not be of such quality or quantity as to render the cost of removal prohibitive.

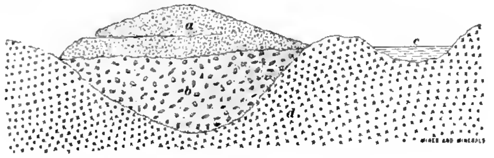


Fig. 1
a, Basalt; *b*, Drift in Ancient River Bed; *c*, Present River Bed; *d*, Granite

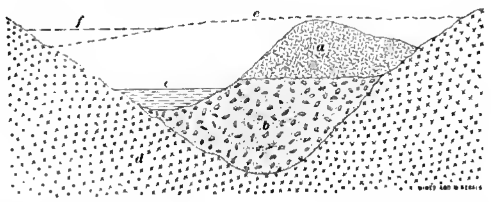


Fig. 2
a, Basalt; *b*, Drift in Ancient River Bed; *c*, Present River Bed; *d*, Granite; *e*, Original Surface Basalt; *f*, Original Ringarooma River Bed

Very few hydraulic mines are worked without elevating a portion of the drift material by hydraulic elevators, therefore the height of the lift must be determined. For estimating the quantity of drift and overburden the method of cross-sections is adopted as shown in Fig. 3 by sinking bore holes to bed rock at intervals of 100 feet. The difficulties found in this practice were the deviation of the bore holes from the vertical and the irregularity of the bed rock. To insure the drill had reached bed rock, and not a boulder, it would be sunk 10 feet after rock was reached in the drift. The method of cross-sections is universally adopted in estimating the drift in ancient river beds, and in one case the estimate was within 4 per cent. of the actual quantity.

In the case of shallow deposits of greater superficial extent, a plan is drawn showing the position of each bore hole or shaft

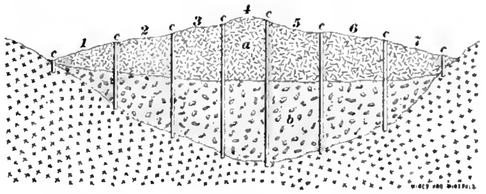


Fig. 3
a, Basalt; *b*, Drift in Old River Bed; *c*, Bore Holes; *1*, *2*, *3*, etc. Values Shown in Table 1

and its depth. Where these depths are fairly uniform over a particular area the average is multiplied by the area represented, giving the volume of that portion; the sum of all such volumes

Table 1. Log of Drill Holes

	No. 1	No. 2	No. 3	No. 4	No. 5	No. 6	No. 7	
R. L. surface........	314.0	326.4	337.2	344.0	342.7	335.1	332.3	316.0
R. L. top of drift....		307.4	303.7	303.0	303.7	305.1	307.3	
R. L. bottom........	312.0	293.4	276.7	261.0	258.7	268.1	282.3	316.0
Depth basalt........	2	19	33.5	41	39	30	25	
Depth drift.........		14	27	42	45	37	25	
Value..............		1.42	1.06	2.16	2.80	3.50	2.00	
Bore No............	16	17	18	19	20	21	22	

Columns 1, 2, 3, 4, 5, 6, and 7 refer to the depth of the bore holes and material passed through corresponding to the same numbers in Fig. 3.

and the total quantity of drift. In cases where the bottom is uneven either of the following methods are used:

1. Contour lines are drawn at regular intervals of depth; the areas enclosed are then measured by a planimeter, and the volume between every two adjacent contours is computed by multiplying the mean area by the contour interval. The use of the prismoidal formula in place of the mean-area method is a refinement generally not justifiable in this class of work on account of the comparatively few points actually determined, and, consequently, the large amount of assumption in interpolating the remaining points and contour lines.

2. The whole area is divided up by lines connecting adjacent drill holes or shafts. The interval between drill holes would depend on the funds available for prospecting purposes, the nature of the ground and the distribution of values therein. An interval of two chains (132 feet) may be taken for a preliminary test, and should the result warrant closer drill holes, a one-chain or half chain interval may be used by drilling the ground between the first series of holes. The area having been divided into small squares, and depths recorded, the volumes of all these are then computed and figured together with the value per cubic yard, and placed on the plan in respective divisions. This serves as a rough assay plan and is of more use to the management during the progress of mining operations than is a contour plan, which has little practical value. The method adopted in any case can only be decided by a careful study of the material on the spot, but generally one of the above methods, or possibly a combination of them, will be found, with perhaps certain modifications, to suit local conditions.

Perhaps the most important and most difficult problem is estimating the value of the drift. The chief methods employed are: Face sampling, tunnels, bore holes, shafts, combination of any two or more of the above, and bulk test under working conditions.

*Abstracted by E. B. W., with permission of the author, from "Notes on Tin Sluicing in Tasmania," published in Bulletin of Australasian Institute of Mining Engineers, April, 1910.

The object in each case is to treat a known quantity of drift and weigh the tin recovered. If the sample is small it is panned, or if large it is worked in a small sluice. The whole of the sample obtained should be treated, for experience has proved that small samples taken from a larger one nearly always underestimate the tin ore. The chief troubles in estimating the quantity of tin are the difficulty of obtaining representative samples owing to the irregular distribution of the ore in the drift, and secondly, in determining accurately the true volume occupied by the whole sample *in situ*. The best method of arriving at the true size of the sample is to measure all the dirt removed in suitable boxes of known capacity. In the case of tunnels the cars can be used for measuring provided they are over filled and the excess swept off by a straight edge drawn across the top of the car. Allowance must be made for the increase in bulk of the dirt after being broken down. The factor for this is best obtained by carefully measuring a regular portion of the excavation and the volume of dirt it yields. A corresponding allowance is made on the volume of the whole sample. In sampling a face, a cylindrical vessel such as a nail can of known capacity is forced into the face at points where the value is to be determined. It is then withdrawn and any loose sand is poured off. The contents of the can still retain the original volume they occupied in place, and this is carefully determined. The dirt is now tipped out and remeasured in the loose state; the increase in bulk is generally expressed at so much per cent. on the original. The mean of all the separate determinations is the figure applied to larger samples or bulk tests. In very fine wash the increase is from 20 to 25 per cent.; in coarse gravel from 30 to 40 per cent. These face samples are panned separately for tin ore. Where the depth of a deposit has been proved by drilling, the drift material from each bore hole is used as a sample for obtaining the ore contents. Its bulk should not be calculated from the dimensions of the hole, because the rubbing of the drill rods causes an enlargement in the diameter and a corresponding increase in the amount of dirt extracted. This increase may be a large proportion of the total sample. For example, a 4-inch drill hole worn to 5 inches in diameter would yield 56 per cent. more dirt than the calculated amount. The only satisfactory method, therefore, is to measure the dirt in a box or similar apparatus as described. Frequently shafts are sunk to determine the quantity of ore in shallow ground. In this case all the dirt excavated is sluiced if possible; but if not, samples are taken at regular intervals in depth and panned. The results obtained in this way have approximated closely the subsequent working, but the large sample is the more reliable.

The difficulty with prospecting by means of shafts is the necessity of keeping the water baled out, and in very wet ground this may prevent the shaft being carried down to bed rock. The bulk test under working conditions requires a plentiful supply of water, also facilities for the disposal of tailing.

A scheme of operations comprising all the above methods was employed on the Briseis-Ringarooma workings. Tunnels were driven about 350 feet apart and the first two connected by a branch drive. A sample was taken each 4 feet of the drift, and bore holes were put in 100 feet apart. In this manner 5,600 feet of driving and 2,600 feet of boring were done up to the end of 1908.

Face sampling, while it has the advantages of cheapness and simplicity, unless very carefully done does not furnish an accurate value of the ground. In fact in nearly every instance this method has given excessively high results.

Tunnels and drifts when sampled as described, give reliable information for their particular level, but it must be remembered that certain horizons are generally richer than others and the samples must be taken at all depths from top down to bed rock.

Bore holes are generally sunk vertically. Therefore in ground containing rich horizontal seams of tin the hole penetrates all layers and theoretically a more accurate sample should result. In practice, however, it is found that material falls in from the sides of the bore hole, also water in the bore hole washes fine tin down to the bottom, whence it may or may not be recovered. The sample from a bore hole is rather small. Shafts have all the advantages of bore holes and the following additional points in their favor: They give a larger sample and therefore a more accurate value; the size of the excavation allows of a thorough inspection of the ground at all depths, thus giving more exact information on the nature of the ground; if desired, fresh samples can be taken at any depth by sampling the side of the shaft, provided, of course, that the shaft is not in running or wet ground. Generally shafts cost more than bore holes, and the expense of sinking a sufficient number of them to thoroughly test a deposit may be prohibitive, and for this reason drilling is the usual practice. In the writer's opinion a few shafts should always be sunk in conjunction with drilling, both as a check and to give more accurate information as to the nature of the drill hole.

In several cases boring gave a very close approximation to the actual conditions, but in other cases the results were high, and in one or two other instances low. However honestly and faithfully the work be done, assays gotten by boring must always be accepted with caution.

As showing the advantage of large over small samples, the following figures are significant:

Case I. All the dirt taken from a drift was treated in a sluice. As an alternative test a smaller sample was taken by a shovelful from each truck. The result shows that the small sample gave only 53 per cent. of the bulk test.

Case II. Drill holes were sunk in drifts and shafts were then sunk in the same locality. The bore holes gave an assay of 2.16 pounds of tin per cubic yard, while the shafts gave 4.1 pounds; that is, the bore holes gave only 53 per cent. of the value obtained by shafts in the same spot.

Case III. Twelve blind shafts were sunk below a tunnel. Bore holes were put down on the same sites. Without exception the result of the bore holes was less than that yielded by the shafts.

Case IV. On a gold-dredging proposition the ground was bored and then dredged. Bore holes gave only 77 per cent. of the dredge returns. In this case, however, the ground was worth only 1½ grains per cubic yard and the bore holes gave 1.15 grains, a discrepancy of about ⅓ of a grain per cubic yard. The result of these tests seems to show that bore holes have a tendency to underestimate the value, also that small samples taken from the larger ones in the way described, cannot be relied upon.

Before leaving this portion of the subject, a method used with some success in gold dredging in Australia may be worth mentioning on account of its application on a fairly large scale though in a modified form in Victoria. Shafts are sunk to bed rock. The samples taken are panned, the number of colors of gold and total weight being recorded in each case. A table is then drawn up showing the average number of colors per pan at the various shafts. A number of bulk tests, 50 to 100 cubic yards, are treated from the localities of the shafts and the actual value entered opposite the previous result.

In Australia it was found that the two values bore a fairly definite relation to one another. In one case in Victoria the ground at the poorest bore hole gave the highest dredge return, the bore hole having evidently bottomed on a blank in good average ground.

Before the ancient river beds can be mined the basalt cover must be removed. At some mines this overburden is so thin it may be hydraulicked with the drift, but the companies operating on the Cascade River have an overburden sometimes 150 feet thick with the core hard columnar basalt. The Briseis company removed 4,500,000 cubic yards of overburden and drift, using for two consecutive years 30,000,000 gallons of water per day. The overburden being too high to be safely

worked as one face, was worked in benches, which was also advantageous because the debris from the top bench could be wasted in a higher place than that from the bottom bench and drift faces. The different benches were kept sufficiently ahead of the drift faces to permit all to be worked simultaneously. Water for the overburden workings is brought from the Ringarooma and Cascade rivers in 20-inch pipes, a branch going to

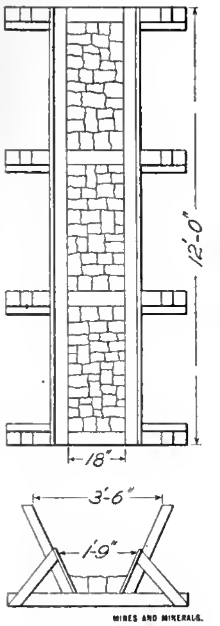


FIG. 4. OVERBURDEN TAIL SLUICE

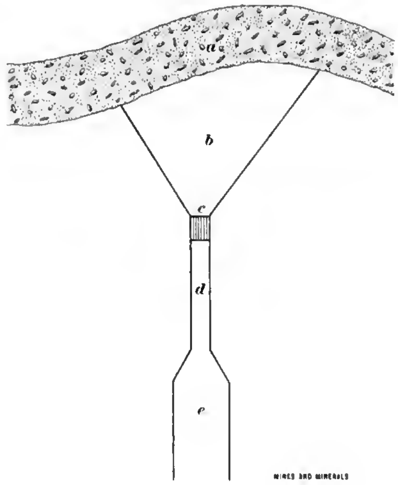


FIG. 5
a, Face; *b*, Head-Sluice; *c*, Grizzly; *d*, Feed Sluice; *e*, Sluice

each nozzle. The universal ball-and-socket nozzle, with interchangeable tips, varying from 3 to 7¾ inches in diameter, is used, in order that the "nozzle man" may readily swing the stream of water to any part of the face and regulate the quantity of water by the size of the tip. The method of working is known locally as "falling" and "washing out."

During the falling operation a 4½-inch tip is used on the nozzle and the water directed against the bottom of the face, which is gradually undermined.

When undermined sufficiently, the ground above the face will be seen to crack several feet from the edge and small pieces fall as a warning. A short time after this the undermined portion comes down and lies in a broken heap at the bottom of the face. When the ground is too hard for this practice, 3-inch holes are put down from the top and other holes put in the face. The holes are then charged with gelignite and blasting powder, the charge varying, but sometimes 10 kegs of powder are placed in a hole and discharged. This shatters the basalt so that water easily brings it down. The operation known as "washing out" is then begun and a larger quantity of water is brought into use. It is kept confined in a head sluice by wing boards, as shown in Fig. 8, where the softest portion of the overburden forms with the water a thick heavy mud which assists in transporting the larger rounded lumps of less decomposed basalt. Some of the stones removed in this manner weigh nearly 200 pounds.

Two or three men with rakes keep the sluice clear and prevent the water spreading and thereby losing its power. One man is employed on the nozzle and one on the dump. When the finer dirt has been washed out, the water is turned off and used elsewhere. The material remaining includes partly decomposed basalt which is broken with picks and washed out, and what remains is trammed out, the pieces too large to handle being broken by block holing and blasting. The sluice boxes of the special design shown in Fig. 5, are made in 12-foot sections and paved with columnar basalt. They cost about $40 per chain. When a wide face is worked, branch sluices are carried well into the face so there may be no difficulty in removing stones. The sluice is 1 foot 9 inches per chain, or 1 in 37½, and the quantity of water that can be used with safety is 10,500 gallons per minute.

The Tasmanian mining regulations limit the size of stones that may be discharged into a river, to a diameter of 2 inches. To separate the large stones the tail-sluice discharges on a large grizzly, the oversize being retained on the dump, the undersize and water sent to the river. In another case the tail-sluice discharges on to the surface dump and the water spreads so that the heavier material settles and the lighter passes on with the water to the river.

The number of gallons of water required to remove a cubic yard of overburden depends on the following conditions: Kind of material being dealt with; quantity of water used (within limits); pressure of the water; design and grade of the tail-sluices; ease of getting material into sluices; care and attention on the part of nozzle men and rakers.

In one face 321,100 cubic yards were removed with 11,630 sluice-head days, or 27.61 cubic yards per sluice head per day. Taking for example the average duty at 27 cubic yards per sluice head per day, on a sluice having a grade of 1 foot 9 inches per chain, then if the fall be 1 foot in G feet, the ratio of overburden to water in the tail-sluice is practically $\frac{1}{G}$. In such calculations the grade must always be stated, the amount of overburden being proportional to the steepness of the sluice, which in this case is 1¾ feet in 66 feet, or 1 in 37.7, so that 37.7 parts of water would remove 1 part of basalt. Since 1 sluice head $= 150$ gallons or $\frac{150}{6.25} = 24$ cubic feet of water per minute, and $\frac{24 \times 24 \times 60}{27} = 1{,}280$ cubic yards of water per day, theoretically the quantity of rock that should be removed per sluice head of water per day is $1{,}280 \times \frac{1}{37.7} = 34$ cubic yards. Practically, the result of a year's work averages 27 cubic yards, therefore the efficiency is $\frac{27}{34} = .80$. During the year but 2 per cent. of the overburden had to be trammed. As a general

FIG. 6. OVERBURDEN AND DRIFT FACE, BRISEIS TIN MINE

formula let B represent overburden; S sluice heads of water; E the efficiency .80; and $\frac{1}{G}$ the fall of the tail-sluice, then

$$B = S \times 1{,}280 \times \frac{1}{G} \times E = 1{,}280 \frac{S \times .80}{G} = \frac{1{,}024 S}{G}*$$

* For the benefit of those who desire to convert these factors to miner's inches, it is stated that the miner's inch is 1.5 cubic feet water per minute or $1.5 \times 7.48 = 11.22$ United States gallons. The British gallon is larger than the United States gallon and 6.25 British gallons make 1 cubic foot.

The formula, while not mathematically accurate, furnishes a sufficiently close approximation for most purposes of the kind.

The drift containing the cassiterite is worked in benches the same as the overburden. Fig. 6 shows the overburden *a*, the drift *b*, with the latter being hydraulicked; also the rock car and track. The stone dumped from overburden faces is shown piled at *c*; the main tail-sluice from the drift faces at *d*; and the streaming shed *e* where finally concentration is accomplished. Apart from the convenience, this method possesses the advantage, that only that portion of the drift below the level of the tail-sluices need be raised by an hydraulic elevator, which means economy of pressure water, saving in time which is occasionally lost owing to stoppages of elevators for repairs etc.

The work consists of washing the dirt from the bank, separating the stones, and allowing the water and dirt to pass through sluices to the dump. Most of the tin ore remains in the head sluice. Where two benches are close enough they are commonly worked together, the material from both passing through one sluice. The lower bench material is raised by hydraulic elevators to the sluice and, as it carries considerable water, the latter aids in transporting the thick material from the upper bench. Where a bottom face is worked singly, the mixture with elevator water is too dilute for sluices of this type, and one of the following methods may be adopted:

1. Run as an ordinary sluice on the lower level and elevate the tailing.

2. Elevate the feed and, where dirt is scarce, work with a "V" settler at the head of the sluice, returning the overflow for feedwater to the same face.

3. Work on a different principle with high riffles. In the first case, if the elevator stops the sluice is flooded, unless all water is turned off at once. In the second case, there not being much pressure in the feedwater, more skill is required. It is, however, economical where dirt is scarce.

In commencing operations on a drift face, a portion of the floor of the face is boarded off to form a rough triangular shaped enclosure All drift has to run over this area and most of the cassiterite is deposited from the water and sand passing on to the sluice. The cement gravel is broken by explosives and spalled into convenient size for handling. Then, together with any clay and stones left from overburden operations, it is thoroughly washed to remove adhering tin, and finally it is trammed out of the mine and dumped. Any cement gravel, stones, etc. still left are caught on the grizzly placed in the sluice head for the purpose. If these were not removed the heavy material would remain in the tin concentrate, while clay, being of a sticky nature when wet, is liable to carry the tin into the tailing.

The dimensions of the sluice are, width, 8 to 14 feet; length, 132 to 180 feet; laid on a grade of 2 feet 3 inches per chain. The sluice is built in 4 to 6 sections, each 24 to 30 feet long. At the end of each section riffles are provided. They are made of $3'' \times 2''$ and $3'' \times 1\frac{1}{2}''$ hard wood, corresponding in length to the width of the sluice. Timber is used throughout in the construction. Bearers and studs are 3 in. $\times$ 3 in.; chocks and struts 3 in $\times$ 1 in.; boards for bottom and sides 12 in. $\times$ $1\frac{1}{2}$ in.; battens 4 in. $\times$ $\frac{3}{4}$ in. and 5 in. $\times$ 1 in. The sluice should be perfectly level from side to side. It may be supported on trestling or built on the ground according to the requirements of the case. Feed-sluices are built exactly like the sluice, except they are narrower, being only 3 or 4 feet wide. The depth of both the feed-sluice and sluice is generally 3 feet. The change in width is made gradual, the best arrangement being as shown in Fig. 7 at *a*, where the centers of feed-sluice and sluice coincide In the other cases there is a tendency toward an accumulation of tin on one side of the sluice, which causes inconvenience in working, especially during the clean-up. Riffles are used in the feed-sluice for the same purpose as in the sluice. The distribution of the feed across the sluice is regulated by baffle boards placed at the junction of the two sluices. At this place there is generally a drop of a foot or two.

At the commencement of a run only one or two riffles are put in the sluices at each section, just sufficient to provide a sand bottom to hold any tin that may reach the sluice. As the work proceeds tin and drift accumulate in the head-sluice and the grade becomes steeper accordingly. The water then has a greater carrying capacity, and not only is fine tin liable to be carried down the sluice, but also the sand covering the tin in the feed-sluice is carried away, and when tin is exposed to the water it is transported to the sluice. To restore the original grade, a riffle is placed in the sluice at each section, in this way causing the sluice to rise correspondingly with the head-sluice. At the same time the riffles prevent any tin being carried past them, and also hold back sufficient sand to cover whatever tin is in the sluice. Work continues, the material in the head-sluice rising slowly all the time, and the sluice being riffled up at the same rate until it is nearly full; it is then run down, the tin is sunk under the drift, a couple of inches of sand being left to protect it from the water. Sluicing is recommenced in the face and operations are repeated as before. The feedwater and sand, on entering the sluice, spread out in a thin layer and flow through the sluice with a velocity just sufficient to carry the sand in suspension without washing away any particles of tin which remain in the sluice, and which eventually settle on to the working bottom of sand. This sand bottom must be kept loose and rough in order that metallic particles may not be rolled along and washed away in the tailing. To keep it in good order the sluice man constantly forks it, causing the tin to sink below the surface where it is then safe. This loosening of the sand makes the working bottom just as loose as the bed on a jig, but there is no pulsion or suction, the effect being more like that of quicksand.

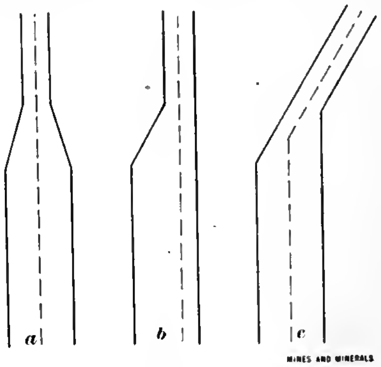


Fig. 7

Apart from the drift carried in suspension by the water, a certain amount is moved slowly along the bottom of the stream, that is, along the surface of the working bottom. This fact renders it important that the tin should be sunk below this field of corrosion which should consist of tailing only. A layer of sand 1 inch thick would probably be sufficient to protect the tin, but a 2- or 3-inch layer is the usual practice, so that if the water should from any cause form a channel in the sluice and scour out the covering drift to a depth of an inch or so, the tin would not be exposed to the water. This channeling can be avoided with reasonable care. Any slight tendency toward it can be remedied by judicious use of the sluice man's shovel, but if the trouble appears to be permanent, a readjustment of the baffle boards at the head of the sluice will generally right matters. The same trouble may be to a certain extent caused through a badly-constructed sluice or by bent or misshapen riffles. Once a good set of conditions is established as regards quantity of water, grade of head-sluice, feed-sluice, and sluice, quantity of drift or feed, etc., the aim is to keep as near to them as possible. If this can be done, the head-sluice and sluice rises at a uniform rate and the sluice is riffled up as required. The quantity of water used depends upon the width of the sluice; 1 to $1\frac{1}{2}$ sluice heads of water per foot of width is the proper quantity, according to coarseness of tin.

It is often difficult to keep the feed regular, especially when working in a cement gravel or stony face. A diminution in the quantity of drift being carried by water results in corrosion of the surface of the bed, and in time the water would cut down to the bare tin, and a loss of metal would result. If a sufficient

feed cannot be maintained, the riffles must be put in at each section of the sluice and form a series of dams which will fill with water and hold back any tin which may be escaping. The feedwater now coming from the face flows over the surface of these dams, the still water held by the riffles protecting the tin in the same way as did the sand covering. Any drift brought through with the feedwater settles as soon as it reaches dead water, and when sufficient has been collected to renew the sand covering, the upper riffles may be withdrawn one by one. The reverse trouble, that is, thicker feed, may possibly cause a loss of tin through its failure to separate out, but in any case it causes excessive deposition of sand in the feed-sluice and sluice, which necessitates the latter being run down more often than really necessary, requiring extra labor and loss of sluicing time. This running down process is a concentration of the sluice contents into a thinner but richer bed so as to reduce the level in the sluice and restore a suitable grade.

The operation is performed by from 3 to 5 men, according to the width of the sluice. The feed being turned off, clean water is admitted at the head of the sluice, and the men working in a line shovel the dirt and throw it over. The water carries light particles and the level is thus gradually lowered. A riffle is taken out wherever necessary and the process continued until the tin is fairly blackened. This is then covered by a layer of sand, and sluicing is then commenced in the ordinary way.

After passing through the sluice the debris drops into a tail-sluice constructed on a flatter grade and is carried away to the dump. The quantity of water used in sluicing is insufficient for this purpose on account of the lessened grade, and an additional supply, called tail-water, is necessary. The quantity depends on the relative proportion of sand and water in the sluicing mixture, and on the relative grades and widths of the sluice and tail-sluice. The steeper the grade of the tail-sluice the less water is required, but on the other hand the lower will be the level of its outlet, and the capacity of the dump is correspondingly reduced. The selection of the tail-sluice grade must be decided in each particular case according to local conditions. The considerations involved are: Dumping space, volume required; dumping areas available; quantity of water, if any, available for use as tail-water; cost of elevating or stacking tailing, if necessary, and amount requiring elevating.

The water supply from Briseis Mines Co.'s various flumes is distributed over a dozen or more overburden and drift faces, also to the water turbines of the tailing-pumping plant and to the electric-lighting Pelton wheels. It is necessary to record the amount supplied to each particular machine or nozzle in order that the working costs of each face and the cost of power may be obtained, also to check the efficiencies, and finally to warn the management of any leaks along the sluice or waste at the mine.

In calculating water measurements the discharge of nozzles and hydraulic elevators having circular orifices is taken simply as the product of the jet area a, the mean velocity v, and the coefficient of discharge c (=.94). The water used would be expressed by the equation

$$w = c\,a\,v = .94 \times \frac{\pi d^2}{4 \times 144} \times \sqrt{64h}$$

In the last expression d is the diameter of the nozzle in inches, and as $v = \sqrt{2gh} = \sqrt{64h}$; h represents the pressure head in feet at the jet. The value of w converted into sluice heads becomes

$$w = .94 \times \frac{\pi d^2}{4 \times 144} \times \sqrt{64h} \times \frac{60}{24} = .1025 d^2 \sqrt{h} \text{ sluice heads.}$$

Gauge cocks are provided at each nozzle so that the pressure in feet of water may be read in every change of shift or nozzle.

The method employed for monthly or quarterly measurements of drift and overburden removed is that of horizontal contours. Points are carefully selected on the face, and with the theodolite set up at a convenient known point, the bearing, angle of elevation, and distance of each point on the face are read. For the distances, stadia measurements are taken. With this information the true horizontal distances on reduced levels are found and plotted on a plan of the workings. Points for the contour lines are interpolated between those taken in the actual survey and a system of contour lines is drawn in 5-foot intervals.*.

The corresponding lines for the previous period are left on the plan, and the area removed at each level is measured by a planimeter. The areas and depths being known, the rest is simply a matter of multiplication. After an extended period of regular measurements of water used and drift removed by it, also of the grades on which the sluice was run at different times, an investigation was made into the relationship existing between these quantities. It was found that the nature of the material being dealt with had such an important bearing on the result that a coefficient had to be introduced to make a formula at all possible. Calling this k, and g the average grade of sluice throughout the run, d the cubic yards of drift removed, w the cubic yards of water used, then $d = k w \sin g$. One sluice head being equal to 1,280 cubic yards of water per day, the amount removed daily becomes $d = 1{,}280\; ks \sin g$, s being the average number of sluice heads in use. The value of k varies so widely that the formula can only be used, and then with caution, after some experience of the ground being worked. For very clean loose drift free from cement gravel and granite, the value of k approaches very nearly to 1. In decomposed basalt capable of being removed almost entirely by water, k=.75 to .80. In cement gravel faces experience is the only guide, for so long as the drift is reasonably clean a fairly good approximation may be expected.

The following example is from actual practice. Where the drift contained a little clay but otherwise was clean the value assigned to k was .95. The water used during the period was equivalent to 900 sluice-head days, and the average grade of the sluice was 2 feet 9 inches per chain, or 1 in 24. Substituting these values in the formula, $d = 1{,}280 \times .95 \times 900 \times \frac{1}{24} = 45{,}600$ cubic yards. The amount removed was estimated by survey to be 46,100 cubic yards, a difference of only about 1 per cent., which is probably within lines of accuracy of the data used.

The work of cleaning up consists in recovering from the head-sluice, feed-sluice, and sluice, the tin and gold which has become concentrated there. The first operation is to "run down" the sluice. This concentrates the tin which is skimmed off and thrown out of the sluice on to a platform. During skimming, the water is turned off, and when the sluice has been practically cleaned out, the feed-sluice is "run in," clean water is admitted at the head of the sluice and men assist by shoveling the tin with the current. The sluice being empty, the grade of the feed-sluice becomes much steeper, and the sand and tin travel much faster than during ordinary sluicing. Riffles must be put in the sluice to hold the tin. When sufficient material is in the sluice it is "run down" again and skimmed as before, and the operation continued until the feed-sluice is cleaned out. Lastly, the material in the head-sluice is "run in" in the same way, clean water with practically no pressure being drawn from the nozzle, through pipes if necessary. In order to hasten the

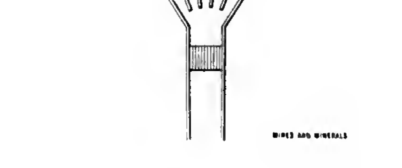


FIG. 8.
a, b, etc., Ring Boards; 1, 2, 3, etc., Head Sluice

* See Tinker's Stadia Chart, MINES AND MINERALS, Vol. XXX, page 483.

operation the water is kept confined between two rows of wing-boards, *a*, *b*, Fig. 8, and, when the sand inside area 1 has been washed away, the boards *a* are shifted to a parallel position *c* alongside the last. The area 2 is then treated in the same manner as 1, and then boards *b* are moved to *d*. This is continued until the whole of the head-sluice has been treated and nearly all the sand removed. The tin mostly has remained behind, and it is skimmed and wheeled in barrows to the feed-sluice, whence it is carried by water to the sluice. Here it goes through the running down process, and when tin is skimmed off and thrown out for next treatment, work is recommenced on the face.

The tin on the plat, although often very clean, still contains sufficient sand and other impurities to make further treatment necessary before smelting. This operation, well known as streaming, is conducted in special streaming sheds. The tin is conducted there through small wooden launders 12 inches wide in which a current of water flows continuously. At some convenient place in this tin launder the gold-saving appliances are provided.

On reaching the sheds, the tin undergoes a preliminary cleaning in a small sluice outside the shed. This is 4 feet wide, 24 feet long, and in two sections. The concentrates from this are ready for the final operation.

Fig. 9. No. 2 Sluice, Two Drift Faces and Derry Overburden Face, Briseis Tin Mine

Streaming is conducted in a small sluice 2 feet wide, 24 feet long, divided into two sections, head and tail, and laid down on a grade of 8 inches per 12-foot section (1 in 18). Water is admitted at the head of the streaming sluice by a door in the launder. About a hundredweight of tin is streamed at a time. The tin is put in the race and water admitted, then the streamer, who must be a very skilful man, shovels the tin very rapidly up the sluice against the water current. After a minute or so the water is stopped, and the tin at the head of the streaming sluice is thrown out and allowed to drain before being bagged. The rest of the material accumulates at the tail of the section against the riffles. As this may contain a little fine tin, it is retreated, and tailings are then stacked outside. They contain, besides heavy sand, topazes, sapphires, titaniferous iron, and tourmaline.

After the streamed tin has drained it is placed in bags holding 1 hundredweight. Moisture determinations are made daily, 112 grams weight of the drained tin being dried and reweighed. The loss in weight is the number of grams of moisture in 112, which is equivalent to pounds per hundredweight, and in weighing the ore into hundredweight bags a corresponding allowance is made.

Reviewing the operations, it will be seen that the concentration is done in four stages. Taking a case where the ground averages 5 pounds per cubic yard, this is less than .1 per cent.

First concentration. The head-sluice would be, say, 1 to 3 per cent.

Second concentration. The sluice concentrate, 60 or 70 per cent. down to 30 at tail.

Third concentration. The preliminary streaming, over 70 per cent.

Fourth concentration. The final streamed tin, 75 per cent.

Returning to the gold sluice already mentioned, this is about 3 feet wide and 10 or 12 feet long, set on a steep grade, 1 in 4 or 1 in 6. It is made of wood, with a coarse screen of perforated metal plate at the head to separate any heavy foreign matter. The gold is saved on coconut matting, of which there are five or six sections. Most of the tin passes on to the streaming shed, but a little remains on the matting with the gold. No mercury is used in the sluice. The gold is practically all recovered from the first and second mattings, which are taken out and washed in troughs. The remaining mattings are moved up toward the head of the gold sluice, and those just cleaned are put in at the tail end for the next clean-up. The gold concentrate, after being freed from small stones and other miscellaneous articles, is treated in an ordinary prospector's dish. Mercury is now added, and the whole thoroughly mixed; some boiling water is poured in to keep the mercury lively and prevent the mass from "caking." The whole contents must be thoroughly mixed until amalgamation is complete. This can be seen by inspection of the concentrates. When no particles of free gold can be seen, and only tin and amalgam are visible, it is safe to give about five minutes further mixing and then pan off the amalgam. The tailings are panned off again to recover any amalgam lost in the first treatment, and about 1 or 2 per cent. may be recovered here. The amalgam is squeezed in chamois cloths, retorted and smelted. The tailings which are chiefly black tin, are run once again through the gold sluice, and thence to the streaming shed.

Drilling Costs in Decomposed Basalt and Drift

Depth	Total Cost	Cost Per Foot
70 feet	£3 10d.	1s.
168 feet	£14	1s. 8d.
255 feet	£27 5d.	2s. 1¾d.
440 feet	£52 10d.	2s. 4½d.

Cost, including all operations to the sacking of concentrate, 5½d. to 6½d. per cubic yard.

During one annual period the cost of each portion of the work was as follows:

	Per Cubic Yard Pence
Sluicing	.77
Hydraulicking and labor in face	1.89
Extensions and maintenance pipes, elevators, and sluices in face	.29
Cleaning up, streaming, weighing (13s. 3d. per ton)	.36
Blacksmithing, blasting, etc.	.25
Electric light	.20
Maintenance main head races	.47
Stores, timber, and explosives	.29
Maintenance main columns, salaries, office expenses, and general	1.08
Total	5.60

Duralium is a new alloy containing aluminum. It is said to be slightly heavier than aluminum and as strong as steel. It can be rolled, drawn, stamped, extended, or forged at suitable temperatures. It is less corrosive than other high aluminum alloys under the usual corrosion tests, and it is only one-third the weight of brass. It is expected that the new metal will find a steadily-increasing demand for aviation uses, while it is likely to prove an important factor in the construction of motor cars, owing to the difficulty of securing aluminum castings sufficiently reliable not to break under the strain of sudden jars and shocks caused by quick stoppages, changes of speed, and jolts upon bad roads.

CHITINA VALLEY COPPER DEPOSITS

By E. Jacobs, Victoria, B.C.

The completion of the Copper River & Northwestern Railroad from Cordova, Alaska, to the mouth of the Chitina River, opens a mining region hitherto almost inaccessible. It is expected that by the end of the year the road will have reached Kennicott, 50 miles eastward from the confluence of the Chitina with Copper River, and 194 miles from Cordova. At Kennicott the rails will be alongside the ore bunkers of the Bonanza Mine, owned by the Morgan-Guggenheim Alaska Syndicate, which is building the railroad. At present the Bonanza is about the only property in the big Chitina basin ready to ship ore regularly, the conditions in the case of this mine being exceptional.

Alaska Copper Deposits To Be Made Available By a Railroad. Peculiarities of the Country.

The Bonanza vein is exposed on both sides of a high sharp ridge from which erosion has broken a vast quantity of rock mixed with vein matter. This great natural dump is sufficiently high in copper to be worth handling in mass, so it is possible to begin regular production with comparatively little development.

On the limestone-greenstone contact, which produced the Bonanza ore body, there are in all about 30 groups of mineral claims, all located on exposures of copper ore. There are, as well, other groups elsewhere in the district. The main contact, above mentioned, has been found to occur along a distance of more than 100 miles, and the easterly end of it has not yet been reached.

FIG. 1. LOOKING BACK AT BONANZA MINE

No other property on this contact has been developed beyond the advanced prospect stage, and owing to the great cost and difficulty of getting supplies, comparatively little work has been done on other properties, and even prospecting in the district has been slight. There is little doubt that many parts of the contact have not been even superficially examined, as comparatively few prospectors have been in the district, and most of those who have prospected have been in search of placer gold, which has been found at the head of the basin, chiefly in Dan and Chititu Creeks, estimated to have together yielded between $300,000 and $400,000.

A small colored geological map of the Chitina basin, recently issued by the United States Geological Survey, shows significant conditions from the miner's point of view, while the evidence of prospectors and engineers warrants high hopes for a large, varied, and widely-distributed production of minerals there. The chief areas are of limestone and greenstone, with dikes of andesite and porphyry that are yet to be carefully examined. The entire geological area is varied, and the fact that it is now accessible not only to prospectors, but as well to tourists, must give it speedy and wide-spread confidence.

Apart from its importance as a prospective mining territory, the Chitina basin is of interest as containing untried agricultural resources; also for its majestic beauty and favorable summer climate. It is well wooded and watered. The present gateway is the new town of Chitina, situated at the junction of the Chitina and Copper rivers, which place the railway has lately reached over a steel bridge across the Copper River.

Exploration in the Chitina Valley.—*Exploration in the Chitina River valley was begun in 1885, when Lieutenant (now Major) Henry T. Allen, in the course of a remarkable journey from Prince William Sound to St. Michael, during which he traversed portions of the valleys of Copper, Tanana, Koyukuk, and Yukon rivers, ascended Chitina River on the ice from Taral to Nikolai house near the mouth of Dan Creek. Lieutenant Allen made a reconnaissance route map of the region through which he went and in his report mentioned the occurrence of copper in the vicinity of Chitistone River, or Nizina River, as it is now called.

In 1891 Lieutenant Frederick Schwatka, U. S. A., and C. Willard Hayes, of the Geological Survey, entered Chitina River valley by way of Skolai Pass. They descended Nizina River nearly to Dan Creek on foot, and after building a canvas boat continued their journey down Nizina, Chitina, and Copper Rivers to the coast. They also made a reconnaissance route map of their traverse. In 1889 a party in charge of Oscar Rohn explored the region south and east of Mount Wrangell. This party was a detachment from the military exploration in charge of Captain W. R. Abercrombie.

In 1900 a geologic and topographic reconnaissance of the Chitina and Hanagita valleys and of the lower Copper River was made under the direction of the Geological Survey. A topographic map was made and geologic investigations were carried on. This work, although hastily done, owing to the difficulties of travel and the necessity of covering the greatest possible area in the time available, has nevertheless been of very great value to prospectors of the region.

In 1902, Walter C. Mendenhall, geologist, investigated the geology and mineral resources of the Central Copper River region, and visited the copper prospects of Kotsina River and Elliott Creek, its tributary.

A list of the publications resulting from the work of these expeditions is here given:

Allen, Lieutenant Henry T. Report of an Expedition to the Copper, Tanana, and Koyukuk Rivers, in the Territory of Alaska, in the Year 1885: Washington, Government Printing Office, 1887.

Hayes, C. Willard. An Expedition Through the Yukon District: National Geographic Magazine, Vol. 4, 1892, pages 117-162.

Rohn, Oscar. A Reconnaissance of the Chitina River and the Skolai Mountains: Twenty-first Annual Report United States Geological Survey, Part 2, 1900, pages 393-440.

Schrader, Frank Charles, and Spencer, Arthur Coe. The Geology and Mineral Resources of a Portion of the Copper River District, Alaska: Special publication of the United States Geological Survey, 1901.

Mendenhall, Walter C., and Schrader, Frank C. The Mineral Resources of the Mount Wrangell District, Alaska: Professional Paper United States Geological Survey, No. 15, 1903.

Mendendall, Walter C. Geology of the Central Copper River Region, Alaska: Professional Paper United States Geological Survey, No. 41, 1905.

Rivers and Mountains of District.—Copper River is joined about 100 miles from its mouth by its large eastern branch, Chitina River. From there its general course is nearly west-

* Bulletin 374, United States Geological Survey.

northwest. The two branches of Copper River, the Chitina and the upper Copper, nearly surround the Wrangell Mountains. The area included between these two streams and the heads of White, Chisana (or Shusana), and Nabesna Rivers is generally referred to as the Copper River region. It is divided into a northern and a southern district by the Wrangell and Skolai Mountains, the ice-covered ridges of which form an almost impassable barrier between them. United States Geological Survey Bulletin 374, from which this information has been taken, deals with the southern district only.

The Wrangell Mountains are a somewhat detached or partly isolated mass, bounded on the north, west, and south by valleys, the trend of which is west-northwest and east-southeast. To the southeast they merge into the Skolai Mountains, and these in turn unite with the St. Elias range. Their summits rise to altitudes ranging from 8,000 to more than 16,000 feet, and reach their greatest elevation in such peaks as Regal Mountain, 13,400 feet; Mount Blackburn, 16,140 feet; and Mount Wrangell, 14,005 feet; all of which are visible from Chitina Valley. Mount Drum, 12,000 feet, and Mount Sanford, 16,200 feet, are conspicuous in upper Copper River valley, but are hidden on the south by the other peaks mentioned.

The lofty summits of these mountains and the ridges that join them are the gathering places for snows that feed the numerous glaciers creeping down their sides and out to the valleys at their feet. These snow fields and glaciers are the sources of nearly all the large tributaries of Chitina River which are, from the north, Nizina, Lakina, Gilahina, and Kuskulana rivers and from the south, Tana, Chakina, and Tebay rivers, draining Hanagita valley.

Fig. 2. Kennicott, Alaska

The Chitina valley floor is a broad, gravel-covered, lake-dotted, flat land expanse with a maximum width of at least 10 miles, with the surface broken here and there by low, round-topped hills, and deep cañons of streams which cross it. Chitina River, in the lower 50 or 60 miles of its course, has cut a deep, broad channel in the valley floor, and for the most part of that distance it flows close to the foot of the mountain slopes on the south. The flood plain in places, particularly along the river's lower course reaches a width of 1 mile, and is bounded on one side or on both sides by banks—in some places of gravel, in others hard rock—which gradually decrease in height down stream, but which have an average height of between 100 and 200 feet. Over this gravel flood plain the river flows in numerous branching subchannels, the positions of which are constantly changing, and are particularly unstable at the time of spring floods, so that those who follow them one year may find them entirely different the next. The current is swift, rarely less than 6 or 7 miles per hour.

About 45 miles above its mouth Chitina River forks, the southern branch retaining the name Chitina, and the northern being known as Nizina River. The latter is almost as large as the former and drains nearly half the copper-bearing region.

Occurrences of Mineral.—General information, and in some instances particulars, relative to occurrences of mineral on various streams and in several basins tributary to the Chitina valley may be found in Bulletin 374, pages 71-100. Space restrictions prevent the reprinting here of even a condensed account of the chief claims and the development work done on them. Those dealt with in the bulletin are situated, respectively, on the following rivers or their tributary creeks: Kuskulana River, with its tributary Nugget and Strelna creeks; Lakina River; Kennicott River, into which flow Hidden, Glacier, Fourth of July, Bonanza, Jumbo, McCarthy, and Nikolai creeks; and Chitistono (upper Nizina River, with Glacier, Dan, and Chititu creeks). The greater part of this information relates to copper claims; the Nizina gold placers, however, are also described, and an account is given of placer mining on Chititu and Dan creeks; mention is made, as well, of the occurrence of realgar (sulphide of arsenic) and coal.

The Bonanza Mine, shown on the hill to the left in Fig. 1, is described as being on the most valuable known copper deposit of Chitina Valley. It is situated at the head of Bonanza Creek, about 1½ miles east of Kennicott glacier and 7 miles north of the southern extremity of that glacier. No other property in the Chitina valley region gives similar promise of ore production in the near future. Bonanza Creek is about 3 miles long and heads on the west side of the high mountain ridge running north and south between Kennicott glacier and McCarthy Creek. Its general course is southwest. The main camp and office of the company owning the mine are, however, situated at Kennicott, shown in Fig. 2, the mouth of National Creek, nearly 4 miles by trail from the mine. Fig. 2 shows wire tram terminal on hill; post office in center; shops and mess house below.

South of National Creek the high north-south ridge between the glacier and McCarthy Creek is made up of Triassic shales and limestones, intruded by large masses of light-gray quartz porphyry. These Triassic rocks are separated by a great fault from the greenstone and overlying Chitistone limestone on the north. The strike of the limestone is northwest and southeast, and its dip averages between 25 and 35 degrees northeast. It therefore cuts diagonally across the main ridge and appears at the eastern edge of the glacier, nearly 9 miles north of the head of Kennicott River. The limestone here has a thickness of more than 1,000 feet. Still farther northeast the shales conformably overlying the limestone reappear, but they do not occur within the area of the copper-bearing rocks. Bonanza Creek and the other creeks where copper claims have been located lie wholly within the greenstone-limestone area.

The Bonanza Mine is situated on the west side of Bonanza Creek, on a spur running down to the southwest from the main ridge. The spur divides Bonanza Creek from a small southwestward flowing tributary heading just west of the mine, and is crossed by the greenstone-limestone boundary about ½ mile southwest of the main ridge. On the axis of the ridge this boundary has an elevation of approximately 6,000 feet above sea level, or 3,800 feet above the mouth of National Creek, where the ore bins are built. To the southwest the spur is greenstone; to the northeast it is limestone, rising to an elevation of more than 1,000 feet greater than that of the contact.

The copper ores are chalcocite* and azurite. The chalcocite is in masses of solid ore up to 5 or 6 feet in thickness, in large irregularly shaped bodies, and in stockworks in the limestone. Two principal veins of chalcocite are seen on the surface. They stand almost perpendicularly, 12 to 15 feet

* Copper sulphide, Cu_2S, 79 per cent. copper.

apart, and strike north 41 degrees east, forming the comb of the sharp ridge, but crossing it at a slight angle, as the ridge at this place has a more nearly north-south direction than the veins. The veins do not extend down into the impure lower part of the limestone, but end abruptly and flatten out on reaching it. In places the precipitous northwest face of the ridge is plastered over with masses of solid chalcocite for a distance of 50 or 60 feet vertically below the top.

Azurite* appears on the surface of the chalcocite and also as a lining of small vugs, but it is present chiefly as thin veins that form a network in the limestone and probably are due to the alteration of original chalcocite veins, for some of the azurite has an inner core of chalcocite. Azurite is more conspicuous than chalcocite in the northern 150 feet of the ore body, but chalcocite forms the great mass of the remainder. The ore bodies formed along the northeast-southwest faults of the northern part of the deposit are not the direct continuation of the large chalcocite veins at the south, but lie in nearly parallel veins which cut the ridge at a regular angle, their strike being about north 60 to 70 degrees east. The very rich ore can be traced on the surface for a distance of about 250 feet. It ends abruptly on the south in a nearly vertical limestone wall, but on the north gives place to the lower grade ores, consisting of small veins of azurite and chalcocite, with scattered masses of chalcocite, some of them weighing several tons. This lower-grade ore shows on the surface for a distance of at least 150 feet northeast from the high-grade ores, and small scattered azurite veins extend still farther in that direction. The ore, as it shows on the surface, therefore, extends northeast and southwest along the strike for a distance of 400 feet. The thickness, however, is more indefinite, but the very rich ore, with its included limestone, as seen at the surface, has a width of approximately 25 feet, although the thickness of ore sufficiently rich to be mined may be greater.

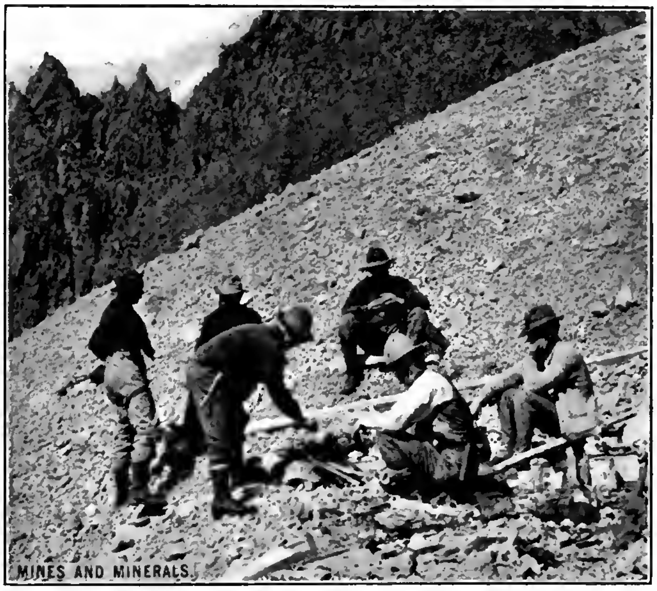


Fig. 3. Geologists on Trail Half a Mile Above Kennicott

A little chalcocite and less bornite is found in some of the shearing planes in the greenstone, but it does not extend far into the greenstone. The quantity is small and inconspicuous and might readily pass unobserved. A small amount of epidote is associated with it in places. The main shear zone in the greenstone cuts an older set of quartz-epidote veins, whose direction is about north-northwest. These quartz-epidote veins do not intersect the limestone. They reach a maximum thickness of 1 foot and carry small amounts of chalcocite, bornite, and native copper.

Two cross-cuts have been driven in the ore body in a direction north 33 degrees west. They are not, therefore, exactly perpendicular to it. The longer of these cross-cuts starts on the east side of the ridge and 75 feet below its top. It is 180 feet in length and extends through to the west side of the ridge. The richest ore, consisting of large masses of chalcocite with some included limestone, is encountered at a distance of 90 feet from the mouth of the tunnel and continues for 21 feet 6 inches, as measured in the roof. There are smaller bodies of chalcocite, however, for a distance of 10 or 15 feet on either side of the main ore body. About 150 feet from the entrance to the tunnel a winze was sunk 33 feet in the ore, and from the bottom a drift zigzags northward approximately 100 feet.

* Blue copper carbonate

About 120 feet southwest of this tunnel is a parallel tunnel driven from the west side of the ridge and 50 feet lower than the little saddle above it on the north. This tunnel starts in a face of solid chalcocite and extends S 33° E for 50 feet. The ore, which is chalcocite, with a small amount of azurite, is exposed for 34 feet along the tunnel, but is interrupted by horses of limestone. The remainder of the tunnel shows limestone cut by small azurite veins and in places containing a small amount of chalcocite

The two main parallel surface veins afford only an imperfect idea of the deposit. Those two veins represent a total replacement of limestone along minor zones where shearing was most intense. The two tunnels show that not only is the limestone replaced along the main shear zone, but that mineralized waters followed minor fracture planes also and thus yielded the low-lying ore bodies and great irregular masses seen underground. Between and around the large masses of chalcocite the limestone was shattered and filled with many small veins of ore, forming a stockwork that is most noticeable in the winze tunnel and on the surface northeast of the main ore body. As a rule the brittle chalcocite is very little fractured. The limestone, on the other hand, is greatly shattered and is filled with thin veins of calcite, which are older than the ore deposition. Open cavities in the fractured limestone have been filled with ice, and both the country rock and the talus on either side of this ridge are frozen all summer except for a few feet at the surface. The talus slopes below the ore body contain a large quantity of chalcocite resulting from weathering of the veins above, and are a valuable source of copper.

It is a suggestive fact that, although the main shear zone of the Bonanza Mine extends from the limestone through the thin shale bed into the greenstone below, the large chalcocite bodies, so far as can be determined on the surface, end abruptly at the top of the impure shaly beds forming the lower 50 or 60 feet of the limestone. Copper minerals are associated with the shear zone in the greenstone, but only in small amount. Apparently the impure thin-bedded part of the limestone was a less favorable place for deposition than the purer massive beds above. This fact has a practical bearing on the quantity of ore present, for it is evident that if the same condition continues underground it limits the downward extension of chalcocite in the limestone. The continuation of the ore body to the northeast will probably be limited chiefly by the continuation of the shear zone in that direction. The exact conditions which determined the deposition of the Bonanza ore body are not known; possibly it was the presence of a shear zone favorable to circulation, but the occurrence of such a zone, together with that of the Jumbo and Erie chalcocite bodies to the northwest indicates that favorable conditions for deposition have been established in more than one place and offers encouragement for seeking other chalcocite bodies at the base of the Chitistone formation.

From the description that has been given it will be seen that there is little on the surface or in the tunnels by which to determine that the ore body has a greater extension from southwest to northeast than about 400 or, at most, 450 feet, or that it extends down into the basal beds of the Chitistone limestone.

It is evident, however, that the Bonanza is an exceedingly rich and unusual body of copper ore.*

The reason usually given for the comparatively small amount of underground development work yet done in the Bonanza Mine is the high cost of getting in supplies. However, as progress is made toward the completion of the railway the cost of transportation is lessened accordingly. There is one feature of the situation that is probably unique; namely, that notwithstanding there is much coal in Alaska within 50 miles of the railway, none of it is at present available for use in the locomotives, so that foreign coal has to be taken by sea to Cordova, costing, it is stated, $12 per ton delivered there. There seems, therefore, to be force in the contention that existing conditions, which do not permit the development and utilization of coal in Alaska, are retarding the full development of that territory.

Estimates have been published of the quantity of ore available at the Bonanza Mine. These are, necessarily, roughly approximate. They place the quantity at 200,000 to 300,000 tons of ore ranging up as high as 50 per cent. copper. A considerable quantity is contained in the talus slope, and this is easily accessible for shipment.

Transportation from the mine to the terminus of the railway will be over a Trenton Iron Works aerial tramway completed last year; this has been constructed in two sections of about 7,000 and 8,000 feet in length, respectively, the upper section being the shorter. The difference in altitude between upper and lower terminals of the tramway is about 4,000 feet.

Pending the opening of coal mines in Alaska, and the subsequent erection of smelting works for the reduction of copper ores, ore from the Bonanza Mine will, it has been announced, be shipped to Tacoma, Puget Sound, Wash. Conditionally that the Copper River & Northwestern Railway shall be completed to Kennicott by the end of the current year, as hoped it will be, production and shipment will be commenced next January. The realization of this expectation will be matter for much congratulation as marking another important advance in the development of the mining industry of Alaska.

* The ore body described is not sufficiently developed to warrant a railroad; and looks to be too rich to be large.

ORE MINING NOTES

The Iowa Tiger Mine, in the Eastern San Juan, Colo., gold belt, was considered a silver-lead milling proposition until it was noticed that by putting more run of mine into the ore the concentrates increased from 1 to 2 ounces in gold. Recently, when prospecting in the hanging wall, minute stringers of what appeared to be pyrite were found to be gold. This ore is known to cover an area of 60 ft. × 50 ft., but as the stope is filled with ore below, and not opened above, the extent is unknown. It seems to be a repetition of the Camp Bird experience, where galena was supposed to be the rich ore, until the late Thomas Walsh found the gold in the hanging wall.

During the two years in which the 300-foot level in the British Mine, at Broken Hill, New South Wales, has been idle, it has been converted into a miniature Jenolan cave. This has been brought about by the percolation of water through the filling in the worked-out stopes becoming charged with sulphate of zinc, which forms stalactites and stalagmites, the former being in some instances 4½ feet long. An analysis shows that for the most part the stalactites are composed of goslarite, which is hydrosulphate of zinc.—*Australian Mining Standard.*

In mining silver-lead ores in the Yellow Pine district, Nev., large quantities of light-colored material were thrown on the dump piles as worthless. This was eventually identified as zinc ore and investigation showed that the mines contained 10 tons of zinc ore to 1 of lead.—*N. B. Gregory.*

At Leadville, Colo., oxidized zinc ores were found in drifts made over 20 years ago. It is presumed that mining engineers did not recognize this material, although the editor is of the opinion that they not only recognized but avoided it to prevent their ores being penalized by the smelters. The Leadville mineral zone is not finished with her surprises and will probably find besides gold, silver, lead, iron, manganese, and zinc, some other metal mineral.

The last fiscal year the Granby Smelting and Refining Co. took over the Cliff and Consolidated St. Elmo mines. It produced 22,750,111 pounds of refined copper at an average cost of 10.34 cents per pound, which was sold at an average price of 12.91 per pound. The tonnage of ore mined and smelted was a little in excess of 1,000,000 tons. The company's holding in the Crows Nest Coal Co. paid 10 per cent. interest. With the smelter running at full capacity it is expected to produce between 25,000,000 and 30,000,000 pounds of copper per year. The company has mined 6,250,000 tons of ore to date and has 6,500,000 tons in sight and expects to develop more tonnage each year than is mined, acquiring new properties if necessary, to this end.

The land law of the Republic of Panama governing the acquisition of public lands (tierras baldias) is of interest. By a recent decision American citizens cannot take up public lands unless they are able to show that foreigners are able to do the same in the respective states of the United States in which the applicants reside, or, in case of a company, in the state in which the company is incorporated. The authorities here have decided to accept as competent proof the certificate of the Secretary of State of the various states of the United States to that effect.

One gold-mining company in the Klondike is building a 6-mile ditch capable of carrying 20,000 miners' inches of water to supply a power house for furnishing electrical power for dredges it intends building in the near future. It also intends furnishing power to other dredging companies. Part of the machinery for this plant has arrived at Dawson and the balance is on the way. The company contemplates increasing the size of the ditch and adding to the power house in a short time. The machinery and supplies for this power house were principally purchased in the United States.

Conservationists are at present interested in the plan of leasing government mineral lands on royalty. As far as coal land is concerned this plan might not be so bad, but there are so many considerations independent of royalty that are sure to creep in, that the plan should not be attempted hastily. Conservation of the lead ores of Missouri was undertaken by the United States Government in 1833 and proved so unsatisfactory it was abandoned.

OBITUARY

Edward F. Payne

On October 17, Edward F. Payne, one of the largest coal operators in the anthracite field, died at his residence in Wilkes-Barre, Pa. He was born in Schuylkill County, Pa., 64 years ago; was educated in New Jersey, and soon after leaving school embarked in the coal business at the East Boston Mine, Luzerne Borough, near Wilkes-Barre. For some time he held the position of outside and inside foreman until finally he was made general superintendent of the colliery. At the time of his death Mr. Payne was director of the Morris Run Coal Co., Blue Creek Coal Co., the West Side Colliery, and also a director in the Miners' Savings Bank, at Wilkes-Barre. He leaves two daughters and one son, Edward Payne, Jr., a senior at Princeton college.

Copper mining in Alaska was less active in 1909, when seven properties were productive, as compared with nine properties in 1908. The output for 1909 was valued at $536,211; that for 1908 at $605,267.

THE RUBY

Written for Mines and Minerals, by Morris R. Ward

Anthrax—"live coal," was the beautiful and appropriate title given by the ancient Greeks to the ruby, which by them, as well as other ancient peoples, was held as the very type of all that was rarest and most valued in the natural world.

Composition, Color, and Characteristics. Methods of Mining. Manufacture of Artificial Rubies.

Mineralogically, the ruby belongs to the corundums, of which the sapphire, topaz, Oriental emerald, and Oriental amethyst are also members. Corundum is a sesquioxide of aluminum, and the various tints are due to the presence of different coloring matter. In the ruby, oxide of chromium in greater or less degree is responsible for the ruddy color so sought after.

A ruby of perfect color is a very rare and beautiful gem, and its value is commensurate with its rarity. Indeed, a flawless, "pigeon's-blood" ruby weighing 5 carats would be worth ten times as much as a first-water diamond of the same weight, while perfect rubies of 6, 8, or more carats are of such unusual occurrence as to bring very high prices. By the term "pigeon's blood" when applied to the color of the ruby, is meant that shade which is by international taste accorded the post of honor, though many of the gems have a darker tint which is quite as beautiful. A drop of blood from a freshly-killed pigeon, when deposited on a sheet of white paper, gives the shade which such a ruby should have.

While rubies, as they increase in size, are almost always filled with serious defects, flawless specimens of large size have been found, though such discoveries are few and far between. A few years ago, a remarkable ruby, perfect in color, was found in Burma by a Shan. This gem weighed 80 carats, was over an inch in length, and in form resembled the celebrated Shah diamond of 86 carats. Taking 20 carats to the ounce, this ruby would weight about 4 ounces, though owing to the wide divergence of a carat in different countries, the ounce weight given is merely approximate. The value of such a stone would be enormous, combining, as it does, perfection in color with size. For the sake of comparison, we may say that a Burma ruby of but one-twentieth the weight of the above (or 4 carats) brought $20,000. And the value of a perfect ruby does not increase directly as its size, but many fold.

Numerous other large rubies of beautiful color are known to exist, most of them probably belonging to imperial courts. In the Russian treasury is said to be the largest ruby in Europe and perhaps the largest perfect one in the world. This stone is described as weighing 100 carats, of a beautiful color, and free from serious flaws. It formed one of the numerous other objects of value which the Chinese brought over to Russia in exchange for pelts.

Another beautiful specimen belongs to the crown jewels of France. This ruby was long in the rough state, owing to a serious imperfection, but an expert diamond cutter turned this flaw to account by forming the ruby to resemble a dragon with outstretched wings. According to *Le Diamant*, of Paris, this is the most beautiful Oriental ruby known.

In 1875, the appearance in Europe of two magnificent rubies from Burma aroused great interest. These stones had been sold by the Burmese Government, owing to certain financial troubles, and extravagant care was taken in conveying them to the ship, a strong military guard being placed about the vehicle in which the rubies were carried. After being recut, they weighed $32\frac{5}{16}$ and $39\frac{9}{16}$ carats, and were sold for $50,000 and $100,000, respectively. Competent judges pronounced these gems the finest of their size ever seen, and of beautiful color.

Besides the above specific instances of large rubies, there are any number of fine stones to be found in private collections, not to speak of the hidden and well-guarded stones of various Oriental monarchs, one of whom (the Shah of Persia), is said to possess a ruby of 175 carats; and stories are told of rubies the size of hen's eggs, and of the immense fortunes represented by the hidden gems of old Indian rulers, who, as "Lords of the Rubies," claimed all stones worth over $50 and inflicted terrible tortures on all who were suspected of holding back desirable stones. We find old travelers like Marco Polo, John Mandeville, and others (aided, we fear, by generous imaginations and credulity) writing of rubies so large as to form pillars in palaces, bowls made of that gem, and specimens which extend beyond the palm of the hand, when so held; and so on. Such tales, while they have their place in literature, are not to be trusted. It is not at all impossible, however, that the deposed king of India, Theebaw, has hidden away great numbers of fine, beautiful rubies, the accumulation of years, which may some day be unearthed.

In the United States are a number of rubies worth over $10,000, and one of $9\frac{3}{8}$ carats is valued at $33,000.

Notwithstanding the many tales of the fortune-making rubies, the mining of these stones is by no means a romantic occupation. The industry has its difficulties and keen disappointments precisely as occur in the search for coal, though the difficulties are intensified because of the great rarity of the gem.

Burma, in India, is conceded to be the principal source of the true pigeon's-blood ruby. There the mining is in the hands of the Burma Ruby Mines, Ltd., an English corporation, which secured the right of working the district soon after Burma became ceded to Great Britain in 1886. Previous to that date the location of the famous mines was unknown, and all attempts to learn about them resulted in failure, for the rulers exercised a most strict surveillance over foreigners, and saw to it that any such were kept in ignorance of the exact location, etc. It is known, however, that the methods used were very crude and tedious; the natives merely dug out the "byon," or ruby-bearing earth, hoisted it by crude derricks to the top of the pit (these "pits" were from 50 to 60 feet deep), and there left it to dry in the sun before searching for gems. Today, essentially the same methods are used by the native miners, in whose hands such labor is still left, and who work under terms much more advantageous to themselves than were permitted under the old regime. In defense of the antiquated methods used it may be said that they bring results, and do away with the necessity of costly machinery.

Besides the true gems, large quantities of spinel and balas rubies are found. These spinels, though often (and quite excusably) mistaken for the genuine stone, are really minerals of very inferior qualities and entirely different composition, being in this latter respect aluminate of magnesium. The spinel is much less rare than the ruby, and not so hard. Besides red, they exhibit a great variety of colors, some being bright cherry; a rare variety is a deep violet, others have a cinnamon shade; and a white spinel comes from Brazil mixed with diamonds. Many of the large historic rubies in royal regalias (including that in the Maltese cross of the English crown), have been pronounced spinels by modern mineralogists, and are, consequently, of little value. The balas ruby is merely a spinel of proper quality, having a rose or deep pink color. The best balas are found principally in Ceylon, though both they and spinels are generally found wherever the ruby occurs.

From Ceylon also comes a true ruby of a rich rose color, but not many of the desired pigeon's-blood shade are found, though in all other respects they are beautiful and attractive stones, suffering, of course, from the flaws which occur in all rubies. The Ceylon stones often belie the name, for in color they run from very light red almost to a pinkish; being otherwise very brilliant.

Similar to the Ceylon stone in many respects is the ruby of Siam. This country, though principally noted for its beautiful sapphires, often produces rubies that rival the best Burma

stones, though their color is more frequently a deep red (often called "ox blood"). It is believed that any really good stones which may be discovered in Siam are sent overland to Burma, and sold as Burma rubies.

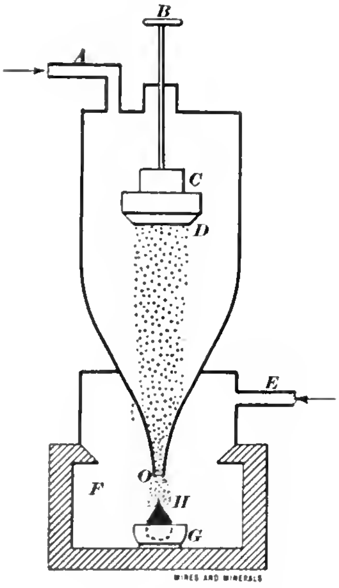
The United States has brought forth rubies as well as sapphires. The principal districts are in North Carolina and Montana. In the former state gems of 3 and 4 carats were found, free from inclusions and of good color; indeed, the quality of these stones often equals the pigeon's-blood rubies of Burma, though their occurrence is rare.

Concerning the discovery of corundum gems in Montana, Mr. E. W. Streeter, a prominent gem expert, has said: "Excepting the South African diamond field, I consider the sapphire and ruby mines of Montana to be the most important (gem) discovery of modern times." Our own expert, however, Mr. George F. Kunz, after a critical examination of these fields, found no true red rubies. So Montana must, for the time being, content itself with producing sapphires of exceptional beauty and purity.

The detection of artificial rubies, as they are made today, is an exceedingly difficult task, for the reason that the best artificial ruby is made of precisely the same materials of which the natural ruby is composed—pure alum, and oxide of chromium or manganese, which latter form the coloring matter.

The furnace used in making these "synthetic rubies" is illustrated in the accompanying diagram. It is a very ingenious application of the principles of the oxyhydrogen blowpipe, and by its means a temperature as high as 2,000° C. (over 3,000° F.) may be obtained. The process of manufacture is as follows:

A certain amount of powdered alum is placed within the box *C*, along with a proportionate measure of oxide of chromium to produce the red color of the ruby. By means of a small hammer operated by an electromagnet, the tapper *B* is touched at various times. This causes the box to vibrate, and the powdered mixture is thus sifted through a fine gauze sieve *D*. Thence it passes down to the orifice *O* within the firebox *F*. At that point the combining of the oxygen gas (admitted through the pipe *A*) with the hydrogen, which enters at *E*, on ignition causes a heat so intense that the powdered alum and oxide of chromium are instantly melted and fused into crystal globules, which fall into the platinum crucible *G*. In this manner a pear-shaped gem *H* (called a "brut") is built up, the size depending upon the amount of material used. Some of these synthetic rubies are 80 carats in weight.



Furnace for Making Rubies

After this "brut" has been allowed to cool (if touched when warm it would fly to pieces owing to the great strain of its molecules), it is sent to the jeweler's, by whom it is cut into one or more gems which in color, refraction, hardness, durability, and general beauty are identical with the natural ruby. Indeed, so absolute is this identity that the pawnbrokers of Paris and other great cities refuse to take rubies on pawn, for they cannot distinguish the artificial from the genuine.

Rubies can be made in this way at an expense of about 40 cents a carat. In the little laboratory of M. Pasquier in Paris (who introduced this method), 100 carats a day may be produced; and 8 to 10 carats an hour is a fair average in all such factories.

Such a discovery as this will have a serious and lasting effect upon the legitimate mining of stone. According to Mr. R. K. Duncan, in his "Chemistry of Commerce": "The ruby mines on their present basis of profitable working, are absolutely doomed." We can only hope that those who control this process will exercise good judgment in regulating the manufacture of their beautiful product, so that the market may not become unstable or overstocked.

Already the use of these synthetic gems is very widespread, and the natural ruby bids fair to become the exception rather than the rule. This fact, however, is not to be greatly deplored, for the artificial product is not an imitation; it is an equal of the genuine, even excelling it at times in beauty of color and crystallization, and in freedom from flaws. If this were not so, the synthetic ruby could never have become the success which it is today.

New Inventions

Complete specifications and drawings for any of the following patents can be obtained from the Commissioner of Patents, Washington, D. C., at the following rates.

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
At entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED OCTOBER 4 TO OCTOBER 25, 1910, INCLUSIVE

No. 971,614. Car haul, Paul W. Holstein and George C. Horst, Columbus, Ohio.

No. 971,779. Mine-timbering machine, Joseph C. Pardue, Willowton, W. Va.

No. 972,017. Ore concentrator, Ira F. Monell, Boulder, Colo.

No. 972,109. Magnetic ore separator, Anders Gustaf Holmberg, Langgrufvan, Sweden.

No. 971,638. Air or steam rock drill, John Redington, Cobalt, Ontario, Canada.

No. 972,352. Treatment of coke, Benjamin Ely and Arthur Rollason, Pye Bridge, England.

No. 972,463. Ore concentrator, Eldor H. Moe, Salt Lake City, Utah.

No. 972,149. Method of treating ores, Charles E. Baker, Chicago, Ill.

No. 972,726. Power rock drill, Sterling Stanley Stevenson, Saltillo, Mexico.

No. 972,509. Curve for double-rope tramways, Sebern A. Cooney, Trenton, N. J.

No. 973,234. Miner's safety lamp, Josef Szombathy, St. Louis, Mo.

No. 973,180. Ore-concentrating jig, Melvin Doubledee, Joplin, Mo.

No. 973,281. Ore jigger, Camden E. Knowles, Webb City, Mo.

No. 973,467. Apparatus for separating minerals from their ores, Samuel K. Behrend, Denver, Colo.

No. 974,075. Ore concentrator, William O. King, Mountain Home, Idaho.

No. 973,984. Magnetic ore separator, Charles A. Sellon, Halleck, Cal.

No. 973,732. Process of reduction of ores containing sulphur and iron, Arnold Wiens, Bitterfeld, Germany.

To manufacture air-nitrate fertilizers, the Southern Power Co., near Charlotte, N. C., has acquired the right to use the "Geneva process" of abstracting nitrogen from the air and also rights to use a newer process. The first plant to be built by the Charlotte company will be used to a great extent for experimental purposes, and the second, which will require 24,000 horsepower to operate, will be used for the manufacture of nitrate fertilizers on a large scale.

Mines and Minerals

Vol. XXXI—No. 6. SCRANTON, PA.—JANUARY, 1911—DENVER, COLO. Price, 25 Cents

THE TOOELE SMELTER*

Method of Handling the Ore. Utilization of Waste Heat. Smelting Processes. Power Plant

The Tooele smelter is situated about 4 miles east of Tooele City, Tooele County, Utah, and by rail is 41 miles from Salt Lake City. The smelter, which belongs to the International Smelting Co., is connected by its own railroad called the Tooele Valley Railroad, with the San Pedro, Los Angeles & Salt Lake Railroad, the junction being about 7 miles from the smelter site. The smelter is on a hillside, which makes it possible, to a large extent, for the level of delivery of the product of one building to be on the level of the charge floor of the next succeeding one.

The ore treated comes principally from the mines of the Utah Consolidated Mining Co. in Bingham cañon. These mines are connected with the smelter by an aerial tramway, 4 miles long, having a capacity of 125 tons per hour.† Custom ores from other mines are received over the San Pedro and the Tooele Valley railroads.

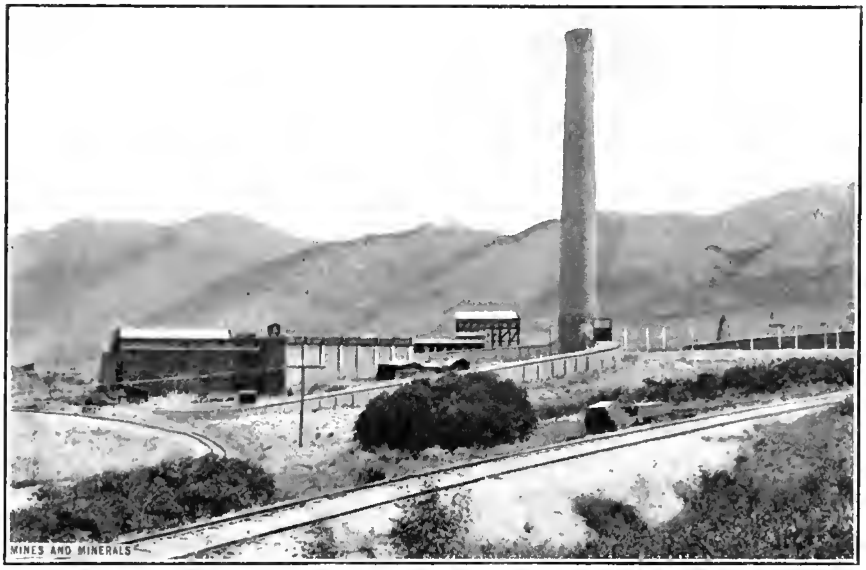


Fig. 1. Tooele Smelter

The ore bins are constructed of steel, are fireproof throughout, and have a capacity for 10,000 tons of ore and coal. The bins are so arranged that they may be served either by the 50-ton electric larry cars, which deliver the ore from the tramway terminal, or by standard railroad cars.

The building in which the crushing and sampling is carried on is of steel and concrete. It is shown in the center of Fig. 3 and is 40 ft. × 84 ft. in plan, and five stories high. The mill is in two sections, and each has a crushing and sampling department. The ore from the receiving bins is automatically fed upon belt conveyers, as shown in Fig. 4, which convey it to shaking grizzlies that discharge the coarse ore into the crushers. The ore from the crushers is lifted to the top of the building in bucket elevators. In the sampling department the ore is cut four times by Brunton automatic samplers, taking one-fifth of the amount each cut and discarding four-fifths, so that from each ton of ore crushed a sample weighing 3.2 pounds is obtained.

Fig. 2. Electric Ore Cars

The sampling department of each section of the mill contains, besides a 12″ × 24″ Blake crusher, which is common to the crushing and sampling department, the following machinery and apparatus: One 0-A sampler, one 20″ × 10″ crusher, one 1-A sampler, one 48″ × 12″ rolls, one 2-A sampler, one 26″ × 15″ rolls, one 4-A sampler.

The discard from the sampling machines drops into the screens of the crushing department, from which it is conveyed to the roaster bins, shown to the right of the sampling building in Fig. 3. The final sample is cut down by a quartering shovel, the sample placed on a steam dryer to expel moisture, then ground to pass through a 100-mesh sieve. The 100-mesh product is put up in three sample packages, one of which goes to the smelter laboratory for analysis, one to the owner of the ore, and one is filed away for use in case of a dispute.

The crushing department of each section of the mill contains one 12″ × 24″ Blake crusher, two 15″ × 9″ crushers, two 48″ × 14″ screens, two 48″ × 12″ rolls.

From the crushing plant the sampled ore is carried by belt conveyers to the McDougall roaster steel receiving bins, which have a capacity of 5,700 tons. If it is desired, the coarse ore after being sampled may be conveyed to the blast-furnace steel receiving bins, shown to the left in Fig. 3, which have a capacity of 3,500 tons.

A blast furnace has not yet been constructed, but provision has been made for it.

From the roaster bins the ore is automatically fed upon a conveyer system which conveys it up to, and discharges it directly into, the McDougall furnace charge hoppers. The ore in transit from the bins to the roaster plant passes over a Blake-Denison automatic and continuous weighing and recording machine, which accurately weighs all the ore delivered to the roaster plant.

*Abstract from paper read by C. H. Redpath and C. M. McGregory, Consulting Engineers.

†Mines and Minerals, October, 1910, page 150.

The roaster plant consists of two buildings, each 64 ft. ×162 ft. Each building contains 16 McDougall calcining furnaces, with six hearths 16 feet in diameter, and 18 feet high. The furnaces have revolving water-cooled shafts and arms, driven by a suitable gearing from the bottom. The rabbles are set to move the material from the circumference to center and vice versa on alternate hearths, until it finally drops into

Blast-Furnace Bins *Sampling and Crushing Plant* *Roasting Ore Bins*

FIG. 3. ORE BINS AND SAMPLER, TOOLE SMELTER

the calcine hoppers, immediately over the tracks of the electric cars shown in Fig. 2, for transportation to the charge floor, shown in Fig. 5, of the reverberatory building. No fuel is used other than the sulphur in the crushed ore, the burning of which furnishes sufficient heat to do the calcining. The gases are taken through flues into the large brick and steel dust chamber, which is 120 ft.×140 ft.×40 ft. high. This dust chamber is so arranged that the bottom forms a series of hoppers. The entire contents of the dust chamber can be drawn into flue-dust cars of the electric tramming system, and conveyed directly to the reverberatory furnaces for smelting. Each of the 32 McDougall furnaces has an approximate capacity of 45 tons in 24 hours.

The reverberatory plant consists of three buildings joined together; the reverberatory charge building, containing the ore and coal hoppers over the furnaces, 66 ft.×280 ft.; the reverberatory furnace building, covering the larger portion of the furnaces, 82 ft.×280 ft., and the boiler house, 36 ft.×326 ft. The latter, at the present time, contains four 746-horsepower waste-heat boilers and three 350-horsepower hand-fired boilers. The boilers are all of the Stirling water-tube type. The furnace plant contains five coal-fired reverberatories, the hearth dimensions of which are 19 feet in width, 102 feet in length, with a grate area of 7 ft.×16 ft. These furnaces have a maximum capacity of 300 tons of roasted ore in 24 hours, on natural draft. The fuel used is Diamondville coal, shipped from the mines in Wyoming, owned by the Washoe Copper Co. The coal is dumped from large railway coal cars, or from cars of the electric tramming system, into hoppers that have five points of discharge, directly over the firebox. The flame after leaving the furnace passes through a 746-horsepower Stirling boiler, which reduces the temperature of the gases going to the main flue to about 600° F. By this means approximately 600 boiler horsepower are obtained from each furnace from the waste heat. The ashes from the furnace firebox fall into the hopper cars of an electric tram system, and are hauled away to the ash dump. The slag is skimmed from the reverberatories twice in 8 hours. It is allowed to accumulate until its depth is from 3 to 4 inches above the skimming plate in the front of the furnaces, then it is skimmed into slag cars having a capacity of 225 cubic feet, and hauled to the slag dump over the electric tramming system. The matte is tapped from the side of the furnace through a copper tap-hole plate and run through cast-iron launders directly into the converter building.

One of the waste-heat boilers has been equipped with a superheater, placed at the rear of the boiler. In case it proves satisfactory the other waste-heat boilers will probably be equipped in the same manner.

The main converter aisle of this plant is 65 ft.×408 ft., and the casting shed is 52 ft.×255 ft. The aisle contains five stands for converters, and the lining department. The converters are of the horizontal-barrel type, shown in Fig. 6, 96 inches in diameter, 150 inches in length, and are electrically operated. The main isle is served by one 60-ton electric traveling crane; and the casting department by a 30-ton electric traveling crane. The end sections of the launders from the reverberatories are pivoted, so that the matte will flow directly into the converter opposite a reverberatory furnace, or the launder may be turned so that the matte will flow into a ladle, and be transferred to any of the other converters in the building. The slag is poured from the converters into unlined cast-steel ladles, and transferred to the reverberatory furnaces by means of overhead cranes. There are two of these cranes of 12½ tons capacity each. The blister copper from the converters is poured into a ladle, lifted by the crane, and transferred by car to the casting department. In the casting department the ladle is lifted by a crane and its contents emptied into anode molds, which completes the operation. The anodes, which are virtually the metal called pig copper, contain 99 per cent. copper and the gold and silver. They are shipped East where they are electrolytically refined and the gold and silver recovered.

The converters are lined in the main building, but the lining material is prepared in a 24 ft.×40 ft. building, adjacent to the ore-crushing plant and receiving bins. The lining material contains a high percentage of silica, also some gold and silver. The material is crushed in the crushing plant and conveyed to a 7-foot dry pan and pug mill in the clay and silica mixing and grinding plant. The product of the pug mill drops directly into a bin over the electric tramming system, from which it is transported to the converter lining plant. The lining plant at the converter contains a 7-foot grinding and mixing pan. From the grinding pan the material is transferred to the converter, where it is tamped around a cast-iron form by a special Ingersoll-

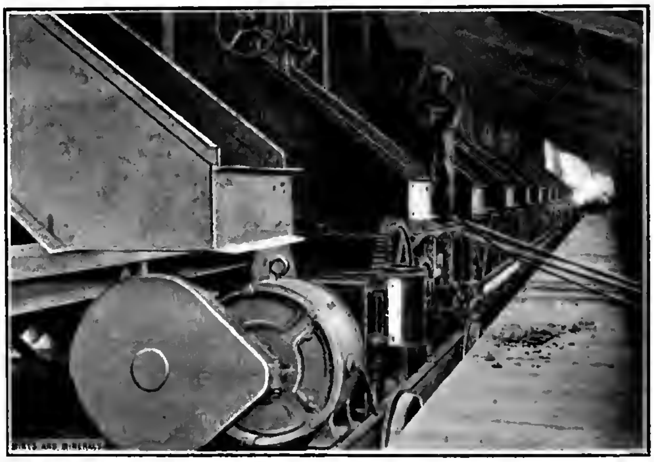


FIG. 4. FEEDERS TO BELT CONVEYER

Sergeant tamping machine, 5-inch diameter by 20-inch stroke. The tamping machine is supported by a revolving jib crane, the vertical motion of which is controlled by an electric hoist. After lining, the converter is removed to its stand, where it is dried.

The smelter power plant building is of brick and steel, the main aisle of which is 52 ft.×240 ft. It has a lean-to 30 ft. ×90 ft. for the steam auxiliaries, and one on the other side

30 ft. × 60 ft. for the switchboard and transformer. The building contains the various power engines, blowing engines, compressors, and auxiliaries and is equipped with an overhead electric traveling crane.

The building contains one 16″ × 32″ × 36″ Corliss engine, direct-connected to a 250-kilowatt, 500-volt, direct-current generator; one 15″×30″×36″ Corliss engine, direct-connected to a 250-kilowatt, 500-volt, direct-current generator; two 18″ × 26″ and 40″ × 24″ vertical triple expansion engines, each direct-connected to one 750 K. V. A., 2,200-volt generator; one 15″ × 30″× 36″ × 42″ blowing engine; one 26″× 52″× 48″ blowing engine; one 13½″ × 26″× 26″× 15″× 36″ steam-driven air compressor; one 26¼″ × 15¼″ × 18″ motor-driven air compressor; one 200-kilowatt, motor-driven, direct-current generator; one 50-kilowatt, motor-driven, exciter generator; one engine-driven, 50-kilowatt, exciter generator; two feed-pumps, having a total capacity of 678 gallons per minute; one fire-pump of 750 gallons per minute.

Besides the machinery listed, the plant contains the necessary condensers, vacuum pumps, feedwater heaters, traps, etc. The steam for these engines is furnished by the waste heat and hand-fired boilers adjacent to the reverberatory building.

The engine-driven generators, in addition to furnishing power for the various motors about the plant, will also furnish power for driving air compressors and for hoisting and pumping purposes at the mines of the Utah Consolidated Mining Co.

A cooling tower of the natural draft type has been constructed for cooling the water from the power plant condensers, and for cooling the cooling water for the McDougall roasting furnaces. The tower is 49 feet high, and at the base is 20 feet wide by 140 feet in length.

The equipment of the electric tramming system consists of three 7½-ton and two 18-ton electric locomotives, also 50 cars of various kinds for handling ore, coal, calcine, flue dust, ashes, and slag. There are about 10 miles of industrial tracks in the plant.

The main flue, shown in Fig. 1, which conveys the gases from the reverberatory furnaces and converters, is 20 feet wide, 18 feet high, and 1,360 feet long. One section of the converter flue is 8 feet wide, 12 feet deep, and 248 feet long; the other section has a cross-sectional area of 132 feet, and is 181 feet long. Both sections are provided with hopper bottoms and chutes for removing the flue dust. The roasting plant flue is

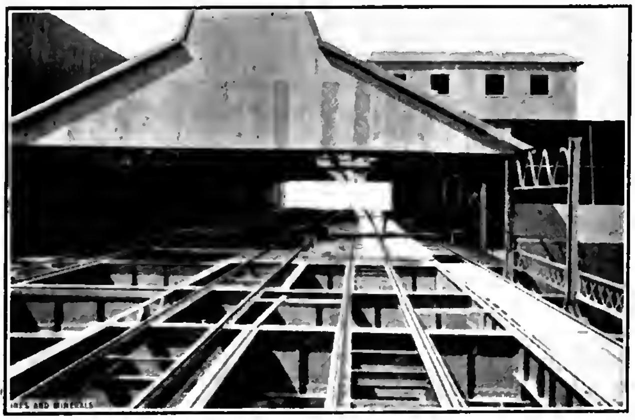


Fig. 5. Reverberatory Charging Tracks

16 ft.×16 ft. and is 255 feet in length. The main and roasting plant flues are rectangular in section, and of brick and I-beam construction, while the converter flues are constructed entirely of steel.

The stack is 25 feet inside diameter at the top by 350 feet in height above the base.

The offices and shops are in steel frame buildings, the outsides of which are covered with corrugated iron. The office rooms are lathed with expanded metal and plastered. The main building is 74 ft. × 348 ft.; at one end are the offices; and at the other the machine, carpenter, and electrical shops, Between the shops and office are the warehouse, change rooms. and laboratory. The blacksmith and boiler shop is 180 feet away from the machine shop. It measures in plan 74 ft.×117 ft. Adjacent to the blacksmith and boiler shop is the locomotive

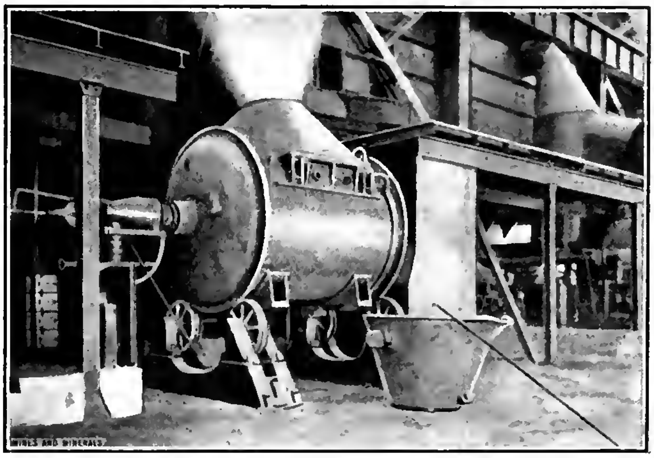


Fig. 6. Converter

shed, which is 42 ft.×89 ft. The locomotive shed shelters the locomotives and spare electric tramming equipment.

The water for various purposes in the smelting plant is obtained from a dam in Pine cañon. The water flows by gravity from the cañon through a 12-inch pipe, approximately 5,000 feet long, to a standpipe of 50,000 gallons capacity at the plant. From the standpipe the water is distributed to the various departments.

Two tanks having a capacity of 43,900 gallons each, are located at an elevation such that ample pressure is afforded for fire purposes at any part of the plant. Water is pumped into these tanks from the general water supply by the fire pump at the power plant.

COST OF CORE BORING

Mr. Moffett, of the Plainfield Mining Co., with pyrite operations at Occoquan, Va., gives the following information which will be of interest to those who drill prospect holes. The tools used were the Terry Core Drill Co.'s 3¾-inch size, which removed a 2⅞-inch core. The men operated the drill 9 hours per day through slate. The advance, which varied from 13 to 36 feet, but averaged 18 feet per day, cost less than $1 per foot, including $4 per day for hauling fuel and water, $7 per day for drill runner and helper working 9 hours daily, and $2 per day for miscellaneous expenses. This record compares favorably with the percussive drill boring of the American oil well rig. The log is as follows:

Date	Material	Total for Day Feet	Total to Date Feet
July 19	Earth 4 feet, slate	23	23
July 20	Slate	36	59
July 21	Slate	15	74
July 22	Slate	13	87
July 23	Slate	13	100
July 25	Slate	24	124
July 26	Slate	15	139
July 27	Slate	22	161
July 28	Slate	14	175
July 29	Slate	18	193
July 30	Slate	15	208
Aug. 1	Slate streaked with pyrites	13	221
Aug. 2	Slate streaked with pyrites	14	235

THE OLD DOMINION PUMPING SYSTEM

Written for Mines and Minerals, by R. L. Herrick.

Having experienced several partial shut-downs of its mine because of sudden flooding in the lower levels, the Old Dominion Co. has recently completed a pumping plant that should easily prevent another such experience, and the capacity of the new plant added to that of the old makes it one of the largest in the country. The new plant is concentrated upon the twelfth level of the A shaft, whereas the old one was distributed over three levels; namely, the tenth, twelfth, and fourteenth, thus entailing as many different sets of pumpmen. The new plant now handles the ordinary flow of the mine, which at this writing amounts to about

Equipment of Pumps. Arrangement of Sump. Methods of Supporting Column Pipes

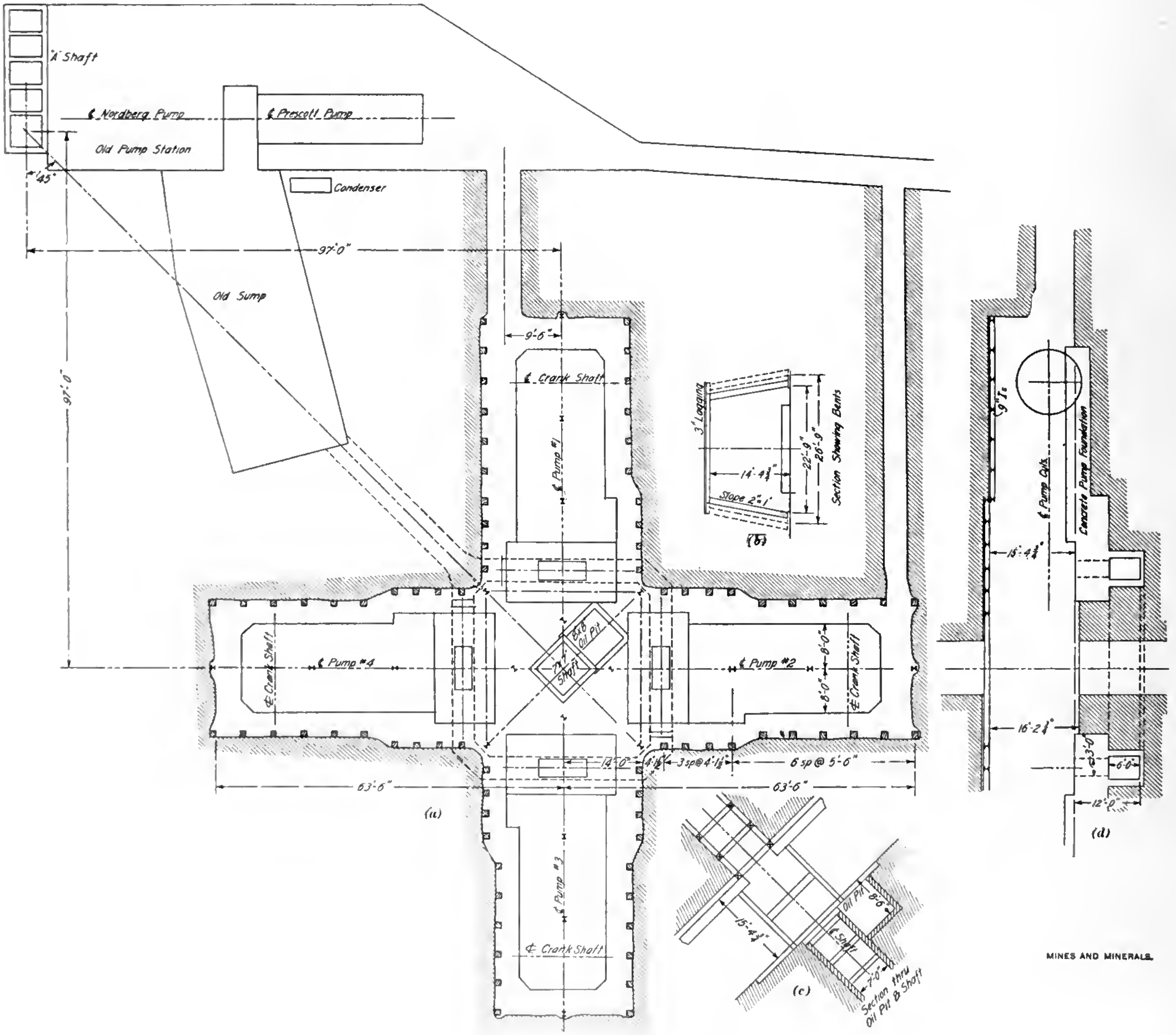


FIG. 1. PLAN OF PUMP STATION, 1,200-FOOT LEVEL "A" SHAFT

2,800,000 gallons daily, while if a sudden flooding is threatened in the future, the old pumps can be started at a few moments' notice, and the combined plants handle nearly 10,000,000 gallons daily, which capacity seems adequate to prevent another flooding.

The total normal drainage capacity of the old plant was therefore 2,750 gallons per minute, or 3,960,000 gallons per day, which by speeding the pumps was at times increased to nearly 5,000,000 gallons per day.

The old pumping plant consisted of the units shown in Table 1.

The pumps on the 1,000-foot level threw to the level of the top shaft landing 30 feet above its collar, from which the water flowed by gravity to supply the concentrator and smelter. The pumps on the 1,200-foot level, however, threw to the level of the drainage tunnel, 205 feet below the collar, where the water discharged into a concrete flume and ultimately into Pinal Creek.

The new system consists of the units shown in Table 2.

From this it is seen that the new plant has a capacity of 4,200 gallons per minute, or more than 6,000,000 gallons per day. This capacity is based upon the normal speed of 50 strokes per minute for the Nordberg pumps and may be increased by speeding them at more than 50 strokes. The combined capacity of the old plant units on the tenth and twelfth levels and the new plant thus easily exceeds 10,000,000 gallons per day.

Of the present flow of 2,800,000 gallons, 1,296,000 gallons are used by the company's concentrator and smelter, and this amount is pumped by one of the smaller pumps to the top shaft landing. The remainder is pumped to the drainage tunnel level

and it is of interest to note, 1,000,000 gallons will be used by the Miami Mining Co.'s concentrator.

The new pump station is located upon the twelfth level of the A shaft, a short distance back of the old station, the arrangement being shown by Fig. 1. The illustration clearly shows the novel cross-shaped plan of the station. The four arms of the cross each have an area 25 ft. × 50 ft. × 15 ft. high outside the timbering, while the central rectangle is 32 ft. × 32 ft. The 7′ × 7′ shaft noted in the center of this rectangle, as best shown by view *C*, comes up from the 1,400-foot level below and was especially driven to carry the pump columns from the electric pumps below, as well as to promote ventilation in the station.

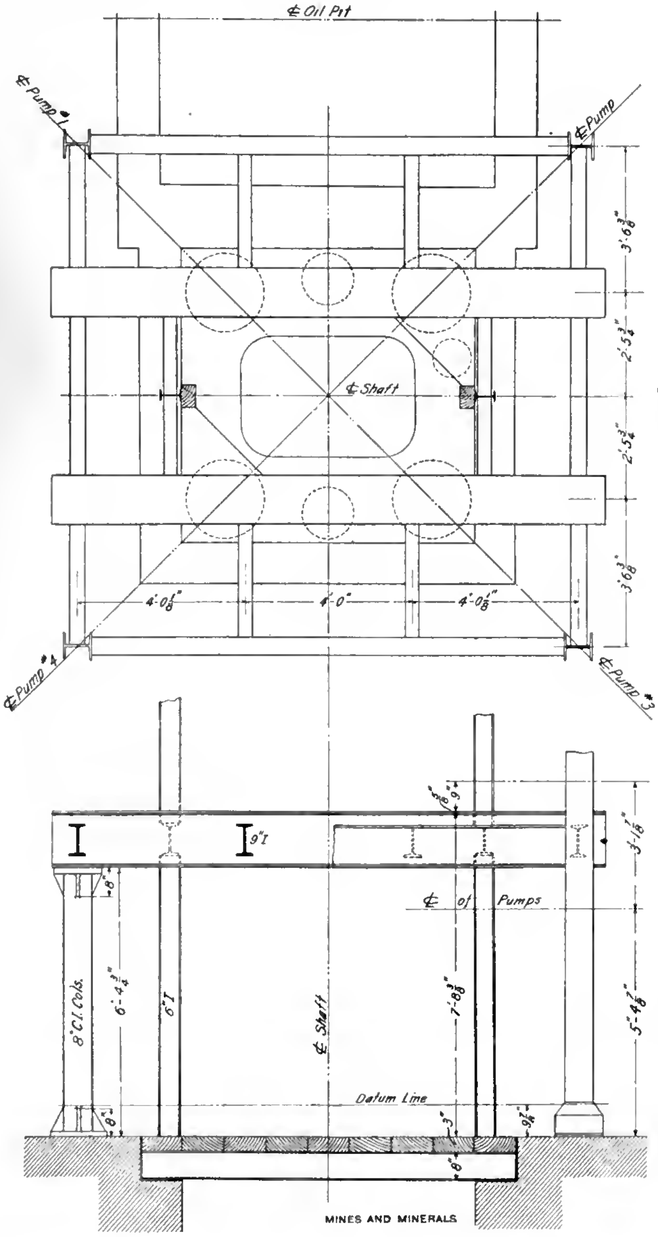


Fig. 2. Pipe Column Support

Above the 1,200 pump station this shaft continues up about 100 feet to where a horizontal cross-cut, 137 feet in length, connects with the pipe compartment of the A shaft. The steam pipes from the plant above come down to this cross-cut and follow thence to the pump station below. The pump water columns are all conducted up the A shaft to the drainage tunnel where they discharge behind a concrete dam into the tunnel flume. As shown in Fig. 3, the columns leading from the two smaller pumps are equipped with valves at the level so that they may deliver either on the surface or into the drainage tunnel. At the present time only one pump delivers at the surface.

Table 1

Level	Depth Below Collar Feet	Pumps	Level to Which Water is Thrown	Capacity Gallons Each Per Minute
1,600	1,229	2 electrically driven triplex Goulds pumps............	1,400	200
1,400	1,028	1 Prescott, steam driven.....	1,200	500
1,200	828	1 Prescott, steam driven.....	205	750
		1 Nordberg.................	205	500
1,000	627	1 Prescott, steam driven.....	Surface	750
		1 Nordberg.................	Surface	750
		Total		2,750

Sump Arrangement.—The large sump, shown in Fig. 1, behind the Nordberg and Prescott units of the old plant is utilized by the new plant, so that in case of emergency both new and old plants will pump from the same sump. Fig. 1, view (*a*), shows a drift 4 feet wide by 6 feet high driven entirely around the central rectangle and connected with the main sump near the A shaft. This sump receives the heaviest flow of the mine which makes on the tenth level and is piped down the shaft to the sump. The bottom end of the rectangular sump shown in view (*d*), is 9 feet below the level of the floor station, and the suction pipes of the four pumps draw from the sump through wells 8 feet long by 3 feet wide.

Station Construction. The station walls are supported by 12″ × 12″ timber posts, set on 5-foot centers and given a batter of 2 inches per foot of height, or 30 inches for the total height of 15 feet. Instead of timber caps, however, whose length would be a minimum of 18 feet, as shown in view *b*, Fig. 1, the posts support 9-inch steel I beams, lagged transversely with 3-inch plank. The massive concrete foundations for the pumps are shown in section in view (*d*) Fig. 1.

Each of the four arms of the Maltese cross contains one of the Nordberg pumps, as described in Table 2. Each pump is of the four-cylinder triple-expansion type equipped with Corliss valves, standard on the low-pressure and intermediate cylinders, but having a full stroke cut-off on the high-pressure cylinder. The two larger pumps having a capacity of 1,200 gallons each per minute, speeded at 50 strokes, have the size 14″ and 26″–26″–26″ × $7\frac{5}{16}$″ × 36″ stroke while the two of 900 gallons capacity each have the size 14″ and 26″–26″–26″ × $6\frac{3}{8}$″ × 36″ stroke.

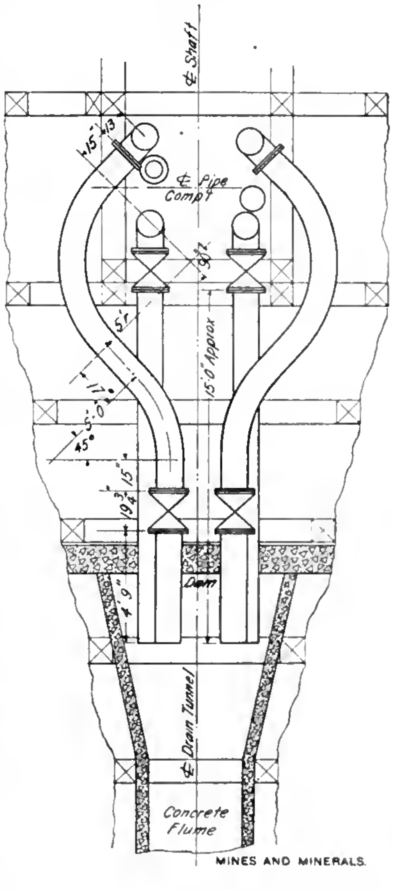


Fig. 3. Plan of Connection of Columns to Drain Tunnel

These pumps work under a pressure of 135 pounds of steam at the throttle. Their flywheels of 12-feet diameter, weighing 9 tons, insure an even running, while the Corliss valves effect a maximum efficiency which results in considerable steam economy. Each pump has two 12-inch diameter suction pipes

drawing from the wells, connecting with the sump, which are situated about the four sides of the central rectangle. The two 8-inch discharge pipes connect with branches of the 12-inch column pipe leading to the surface. Of the four column pipes leading up the shaft, three are of 12-inch diameter and the fourth, which is wood lined, is of 14-inch diameter, the latter being formerly used for pumping the most acid water from the lowest mine levels.

Pipe Arrangements.—It will readily be understood that the support of both the water-column pipes and the steam pipes of such an extensive system requires an arrangement out of the ordinary. View (*c*), Fig. 1, gives a hint of this arrangement, but for details reference must be had to Fig. 2. The sectional view illustrating this support shows that the main weight is borne by four cast-iron cylindrical columns of 6-inch interior diameter and 1 inch of metal thickness. These columns support two heavy 15-inch I-beam girders. Four 9-inch I beams likewise constitute the supporting columns for the four corners of the structure which are further supplemented by two central 6-inch I-beam columns. The rest of the structure consists of 9-inch and 6-inch I-beam girders as shown.

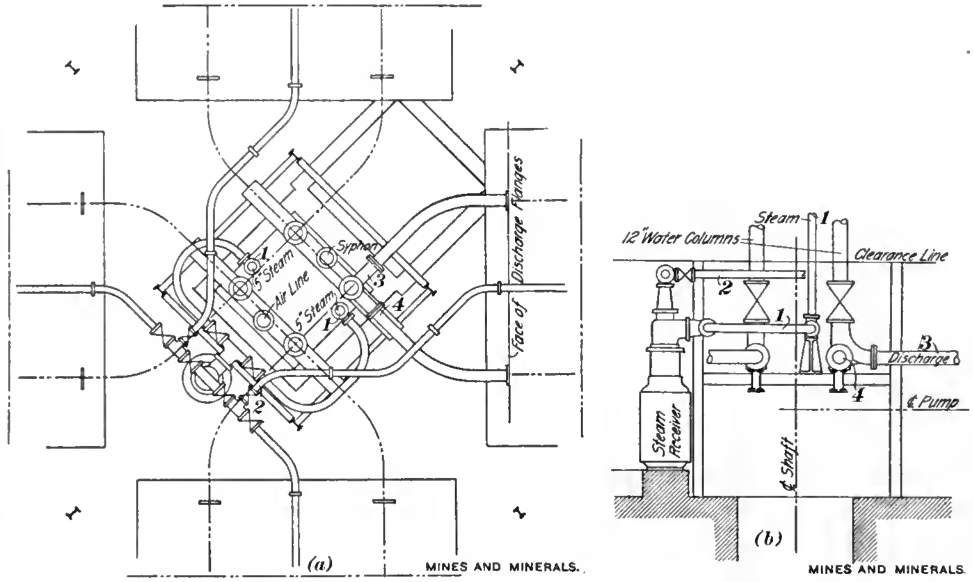


FIG. 4. PLAN OF PIPE ARRANGEMENT, 1200-FOOT LEVEL

This structure is built in the central rectangle of the Maltese-cross-shaped station and supports the weight of both the steam pipes and water columns. By referring to view (*b*), Fig. 4, the method of supporting the pipes from the I-beam girders will now be apparent.

TABLE 2

Level	Depth Below Collar Feet	Pumps	Level to Which Water is Thrown	Capacity in Gallons Per Minute Each
1,600	1,229	2 electrically driven Goulds triplex, size 5½"×7".......	1,400, head 200 feet	250
		1 Goulds triplex, size 10"×16" to be installed in the near future..............	1,400, head 200 feet	500
1,400	1,028	2 electrically driven Goulds triplex, size 12"×16"......	1,200, head 200 feet	750
1,200	828	2 steam driven Nordberg four-cylinder triple-expansion, size 14" and 26"–26"–26" ×7 5/16"×36" stroke........	Drain tunnel 205 feet, head 623 feet	1,200
		2 steam driven Nordberg, same type; size 14" and 26"–26"–26"–26"×6¾"×36" stroke..............	One throw to surface, head 858 feet. Other to drain tunnel, head 623 feet	900

In Fig. 4, view (*a*), it will be noted that two steam lines of 5-inch diameter come down the shaft from the cross-cut and unite in a T leading to the steam-trap receiver, shown in sectional view (*b*). These traps are provided in duplicate and serve to abstract the water from the steam. In the near future the steam will be given 100 degrees of superheat at the boiler plant so that dry steam will probably be delivered at this point. From the receiver the four steam mains radiate to the pumps located in the arms of the cross. The mains from the two 8-inch discharge ends of pump No. 2, numbered 3 and 4 on the illustration, are likewise shown uniting at the 12-inch column pipe that goes up the shaft.

Owing to the arrangement, previously mentioned, of carrying the pipes up 100 feet, thence through the 137-foot cross-cut to the pipe compartment of the A shaft, the column and steam pipes are necessarily given two bends, as shown in Fig. 5 (*a*). By giving these columns a 5-foot radius of bend, however, friction is reduced to a minimum, so that it is said no jarring at the elbow is perceptible. Fig. 5 (*b*) likewise shows how a proper spacing of pipes in the A shaft was effected by giving them ample radii of curvature. Fig. 3 shows the pipe arrangement at the level of the drainage tunnel where the pipes discharge behind the concrete dam into the tunnel flume.

Steam for these pumps is generated at the surface plant, located a short distance from the shaft, which likewise supplies the hoisting engines. This plant consists of five oil-burning water-tube Stirling boilers of 325 nominal horsepower each, and one of 250 horsepower. This plant is modern in every respect, having as additional equipment a Cochran feedwater heater, feedwater pumps in duplicate, as well as a duplicate set of oil pumps.

Summary.—By concentrating the pumping plant which will serve ordinary requirements at the 1,200 level, a considerable steam economy was effected, not to mention the saving in wages of pumpmen. Although the new installation may be assumed to have cost in excess of $100,000, the daily economy as well as provision against flooding has already justified the expense. This daily economy is said to amount to about 25 per cent. of costs.

In conclusion the writer acknowledges his indebtedness to Manager S. H. Dowell and Mechanical Engineer C. H. Weideman for much of the data and the accompanying illustrations of the above article. To Master Mechanic John Langdon, and Mr. Frank Zlatnick, of the Nordberg Co., and to Chief Engineer H L. Norton, thanks are likewise extended for their painstaking assistance.

During 1909 Alaska's auriferous lode mines produced gold valued at $4,107,363, an increase of $749,928 over the product of 1908 and the largest annual lode output yet recorded. The number of producing mines was the same as in 1908, but a score of prospects were in course of development and some of these made small outputs.

JOPLIN DISTRICT ZINC AND LEAD ORES

In excavating for the new union depot at Joplin, Mo., free galena has been encountered and weekly turn-ins are being made by hand-jig operators who pay a royalty to Jenkins & Jennings, the contractors having the excavation work in hand. The ore is found within a few feet of the surface. The site of the new depot is within two blocks of the main business center of the city.

A lower run of ore than any hitherto worked on the tract has been encountered in a drill hole at a depth of 127 to 137 feet by the Waneta Mining Co., operating on the Riseling estate, west of Joplin. A shaft is to be put down to the new formation. On adjoining leases recent drilling has disclosed a still deeper run from 180 to 200 feet. The formations are the typical open-ground variety common in that vicinity.

In the development of the Hackett Mining Co.'s land, west of Joplin, the longwall system of mining is to be tried. This is only one feature of the Hackett development that is radically different from any other mining work in the Joplin district. The company has a large concentrating plant combining features of the western mining plant with those of the prevailing types found in this district. The crushing, the jigging, and the concentrating departments are absolutely independent of each other. Any department may be operated while the other two are shut down. This is effected by means of separate power units and intermediate storage bins. The plant is situated near the center of a 40-acre tract. At regular distances shafts are to be sunk and the ore supply is to be carried to the mill over surface tramways with the exception of the ore that comes from the mill shaft which is hoisted direct to the hopper. The ore occurs in brecciated flint in an 18-foot face beneath a solid cap rock of limestone. It consists of a good quality of zinc blende thinly disseminated through the ground and cemented between the openings in the chert.

Shallow mining operations on the Log Cabin Mining Co.'s tract east of Joplin are resulting in large turn-ins of free galena, while five good prospects hold promise of still further increasing the production from a tract that has been almost unworked for many years and which was looked upon as exhausted. The shallowest ore encountered was at a depth of 25 feet. Other deposits were encountered at 40 to 60 feet. Some of the dirt requires hand jigging.

Each week sees additional mines closed down in the thin sheet-ground districts where operations cannot be conducted at a profit with zinc ore selling for $41 a ton, basis of 60 per cent., and lead ore bringing $49 a ton. The demand for open-ground properties, on the other hand, is increasing. Many mines, similar to those that flourished when zinc ore brought only $18 a ton, are being opened in all parts of the district. The life of these little producers is short as a rule, but the profits are large while the mineral supply holds out. When prices reach

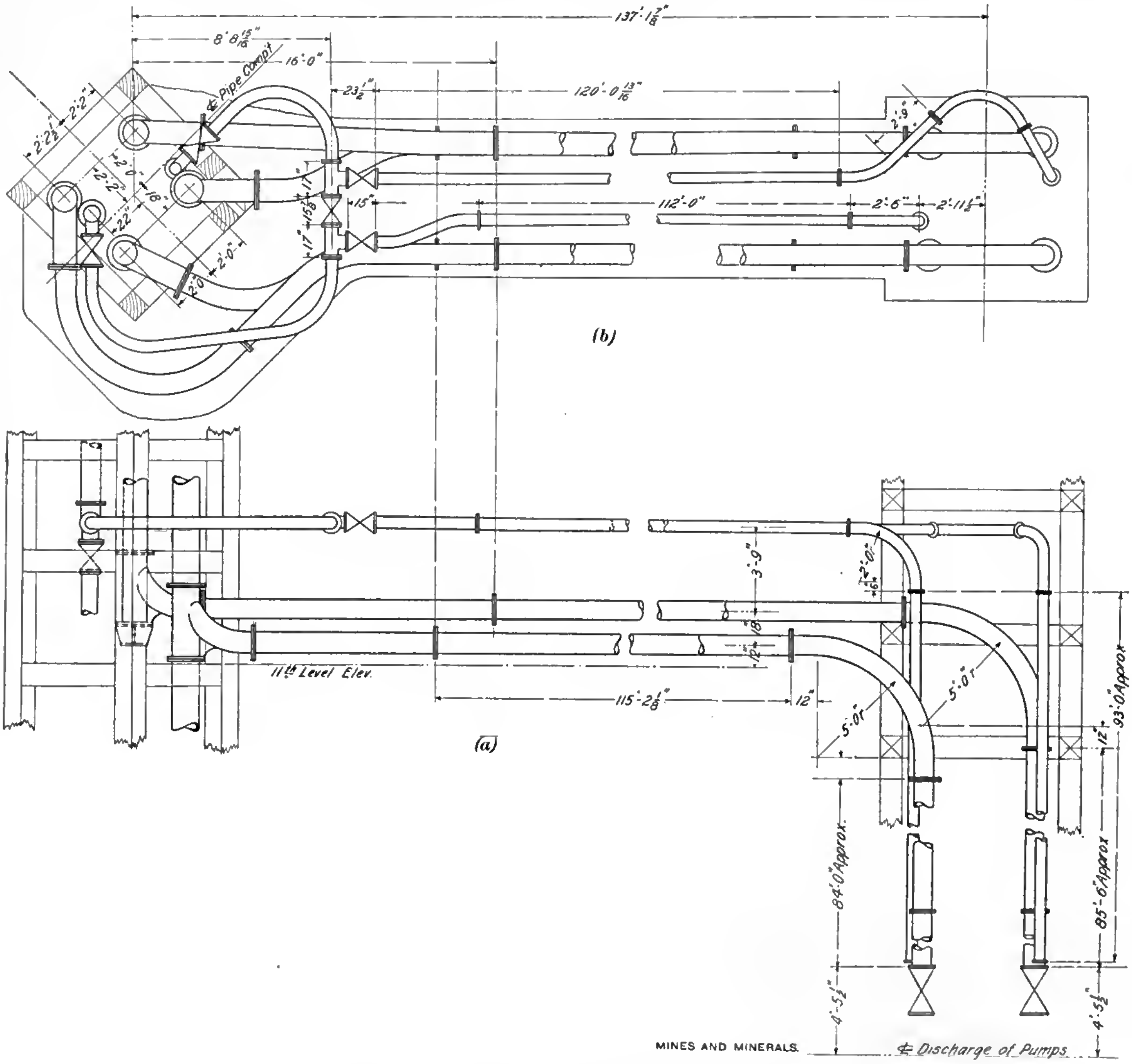


FIG. 5. ARRANGEMENT OF PIPES FOR CONNECTION BETWEEN CROSS-CUT AND SHAFT

a level permitting of profitable operation in the sheet-ground areas, where the deposits supply ore for many years, it seems the operators prefer this costlier mining in preference to the gouge method from which the profits are almost sure, but small.

The heavy weekly turn-ins from the Quick Seven Mining Co., in the Neck City camp, 20 miles north of Joplin, are attracting attention. The mine is a new one, having been discovered by a company of Neck City prospectors, comprising saloon keepers, hardware dealers, bank presidents, and physicians. The tract lies near a small stream and was conspicuous for years as being the only unproductive area in one of the most fertile agricultural regions of the country. Nothing would grow on this property. Drill prospecting was started after several close observers had noted the peculiar tallowness of the clay brought up by crayfish that had burrowed hundreds of holes. This surface indication hinted of mineral lower down. The ore was found below the 100-foot level, and occurs in large pockets both in soft and hard ground. Where it is richest heavy timbering is required. Three big concentrating plants are working on the land and the weekly output of zinc blende is from 200,000 to 600,000 pounds. The ore as a rule runs better than 62 per cent.

Civic improvement leagues in Joplin and Webb City, Mo., have undertaken to improve the landscape of the mining district by planting creeping vines to cover the hundreds of unsightly dump piles and tailing heaps. Superintendents of the public schools of these two cities have distributed seeds to the students with the request that they be planted as soon as possible.

In the Wentworth, Mo., camp, near the site of the famous Gobbler Mine, in which Frank Rockefeller placed so much money in development, the Mocking Bird Mining Co. is reported to be opening a good deposit of calamine. Several small companies in the Wentworth district also report shallow ore strikes.

The "Fellow Servant Law" recently enacted in Missouri has caused the casualty companies, with one exception, operating in this district, to abandon the field. This law makes it possible for employes to secure damages for injuries due to the negligence of their fellow workmen. The original liability insurance rate of 3 per cent. of the pay roll has grown to 4 per cent., and the single remaining company now is contemplating a raise to 4½ per cent.

Workmen risked their lives when they removed the Avondale concentrating plant at Carterville from the brink of an ever-widening cave. The mill was almost swallowed by a sudden fall of earth which left the plant on the edge of a pit 200 feet deep and 100 feet across. It was purchased by a Joplin machinery company and removed to a place of safety.

The 250-ton plant, of the Boston Land and Mining Co. at Galena, has been completed and the work of opening a daylight mine has started. The ground is to be stripped, the ore coming to within a few feet of the surface.

The Newton mill of the Old Dominion Mining Co., at Galena, has just been finished and put in operation. It has a capacity of 200 tons per shift and is to handle ore from a stripped pit, 55 feet deep, 130 feet long, and 65 feet wide.

Pete, Kirke & Co. report drill strikes of zinc ore on the Clark land, 2 miles east of Alba. The formation occurs at a depth of 200 feet and is in virgin territory. A similar strike is said to have been made in several drill holes by Bowman & Ball, prospecting on the Hannum & Shaw land.

The Du Pont Powder Co. has purchased for $75,000 1,000 acres of land from 15 farmers living east of Joplin 8 miles. A dynamite plant, the estimated cost of which is said to be $500,000 is to be established.

It is said that the famous Blue-grass editor, Henry Watterson, requested the proprietor of a poker parlor to let him have $20 worth of chips, stating that he would send Ben around to the office for the money. "You can have the chips, colonel, on this condition: If that lad comes back with the coin your play counts; if he fails to make connections, it don't." "That's all right, Bill," said the silver-penned editor; "pass over the markers." By a singular chance Watterson won from the start and before long had chips representing $500 piled up in front of him. Then Ben came in and Watterson seeing him said: "Hello, Ben, what luck?"

"None, colonel, all the cash was banked; not a centavo in the till." Watterson looked regretfully at his winnings, and then solemnly said: "Ben, beware of cards. I've just lost five hundred hard-earned dollars."

Big Keys, a full-blooded Chickasaw Indian, was in the district recently en route to his home in Oklahoma after a tour of exploration through the Ozark Mountains in Southern Missouri. Numerous other Indians recently have been exploring the Ozarks in search of a cave, which tradition says, is filled with silver, both ore and metal. It is claimed the cave produced silver for many years. Several Joplin miners have sought in vain for the cavern.

More than 50 of the big mills in the sheet-ground districts of Webb City, Duenweg, Prosperity, and Carterville are closed, due to a slump in prices for zinc ore. This shut-down, however, has stimulated operations in the shallow, pockety districts where the cost of production is much less.

Prospects are being opened at Glen Haven, in Grant County, where surface mines were worked fifty years ago. The miners in early days did not stay out among the bluffs, as they soon got below Mississippi level, making pumping expenses too burdensome to bear. There are ranges of crevice ore along the river; some of the crevices being forty and fifty feet high, and half a mile to a mile in length underground.

RIO BLANCO COUNTY, COLO.

The plans of the recently incorporated Grand River, Meeker & Salt Lake Railroad Co. call for the immediate construction of the road from Chapman, a new town between Rifle and Newcastle, on the Denver & Rio Grande and Colorado Midland railroads, to Meeker, in Rio Blanco County, and the ultimate extension of the line down the White River valley through a rich section of Utah into Salt Lake City.

Between Chapman and Meeker, a distance of about 50 miles, the new road will pass through the Grand Hogback and Danforth Hills coal fields. There are 41,720 acres of coal land available for mining in the Grand Hogback field, and 159,700 acres in the Danforth Hills field. The total thickness of the coal at Newcastle, in the Grand Hogback field, is 105 feet; and one seam, the Wheeler, is 45 feet. The total thickness of the seams in the Danforth Hills field is 73 feet. Most of the coal in northwestern Colorado is a high-volatile product. It averages close to 12,000 British thermal units, and is equal in all respects to any of the coals in competitive markets.

Farther down the White River, along the line of the proposed railroad, is the Rangely oil field. The producing wells in the field have been plugged, awaiting the provision of transportation to market. The oil has a paraffin base and is free from asphalt. In color it is a clear light red, with a strong green fluorescence, closely resembling the Pennsylvania oil. Its odor is like that of kerosene, much resembling that of a refined oil. It is apparently free from sulphur. The specific gravity of the crude oil from Requina well No. 1 is .8092 or 44° B. It would, therefore, be described as a very thin light oil.

As the White River approaches its confluence with the Green, it enters a hydrocarbon field the like of which has been found nowhere else in the world except in the Ural Mountains. There are perhaps 20 varieties of these hydrocarbons, among them gilsonite, elaterite, and ozocerite.

The carnotite of the White River valley is an ore of uranium and vanadium, and is highly radioactive. It occurs as impregnations in sandstone along fracture planes. It is cheaply mined, but the recovery and separation of the uranium and vanadium present some difficult problems.

AMERICAN INSTITUTE PANAMA MEETING

Written for Mines and Minerals, by E. W. Parker

An Account of the Trip to Panama. The Conclusions of the Institute in Regard to the Panama Canal

The ninety-ninth meeting of the American Institute of Mining Engineers, held on the Canal Zone of the Isthmus of Panama, will long be remembered by those taking part, as one of the most enjoyable meetings in the history of the organization. Much of the success and enjoyment was due to the unselfish and cheerfully rendered services of Dr. Joseph Struthers, who made all the arrangements, and saw that they were carried out. Doctor Struthers was assisted by A. E. Vaughan, and if Struthers was not happy Vaughan was, and vice versa, with the result that no one but these two had trouble, while the others had the time of their lives.

The party, 120 in number, sailed from New York on Friday, October 21, making the first stop at Havana, on Tuesday, October 25. The sea, which was slightly rough the first two days, was for the rest of the trip as peaceful as could be desired. Three days were allowed the members to get accustomed to ocean travel, but on Sunday morning divine services were led by Dr. R. W. Raymond, LL. D., secretary of the Institute. In the afternoon, Doctor Raymond also talked (to those able to sit up) on Jamaica and its people. The first session of the Institute was held on the steamer, Monday, October 24, and it was devoted to the discussion of mine fires, which was begun by W. A. Lathrop, of Wilkes-Barre, Pa., who, with blackboard illustrations gave an account of the walling off of the fire in the Summit Hill colliery, at Summit Hill, Pa. The discussion was continued by S. D. Warriner, of Wilkes-Barre, Pa., on a fire in No. 8 shaft of the Calumet & Hecla Co., Michigan; R. V. Norris, of Wilkes-Barre, Pa., on the fire of 1901 at the Big Lick slope of the Lykens Valley Coal Co.; W. J. Richards, of Pottsville, Pa., on the prolonged fight of the fire in the Pineknot colliery in the Heckscherville Valley; S. A. Taylor, of Pittsburg, Pa., on the recent fire at the Monarch Mine, near Rock Spring, Wyo.; President Brunton, of Denver, Colo., on the fire in the Anaconda Mine, at Butte, Mont.; C. W. Goodale, of Butte, Mont., on the fire at the Leonard Mine.

Fig. 1. Gamboa Bridge on Panama Railroad Near Canal

The features of the stop in Havana were visits to Cabana Fortress and Morro Castle; a reception by the President of the Republic, drives about the city and the purchase of cigars. Promptly at the hour set, on Wednesday, departure was made for Kingston, Jamaica. On Thursday, October 27, the second session of the Institute was held, the discussion of mine fires being continued as follows: Fire in the Luke Fidler colliery, at Shamokin, Pa., by R. V. Norris. Four fires in the Vulcan iron mine, by Wm. Kelly, of Vulcan, Mich. Recent explosion at Palau No. 4 Mine, near Las Esperanzas, Mexico, by Edward W. Parker, of Washington, D. C. Fire in the De Beers diamond mines, South Africa, by Gov. Gardner F. Williams, Washington, D. C. Only one paper was presented at the session held on Friday afternoon. It was by Prof. Jos. W. Richards, on the electric smelting of pig iron in Sweden. Professor Richards illustrated his lecture with blackboard drawings.

Kingston was reached early Saturday, October 29, and here, after practically sacking the town of light clothing, the party drove to Castleton Garden; took a trip by rail on Sunday to Bog Walk, and then a drive through the cañon of Rio Cobre to Spanish Town and returned to Kingston, whence they set sail for Colon.

Monday, October 31, at the fourth session, W. L. Saunders, of New York, gave a historical account of the conception of the Panama Canal with a description of the isthmus and of the work so far accomplished. He was followed by John M. Sherrerd, of High Bridge, N. J., with an address on the social life of the Canal Zone.

An accident to the ship's machinery caused a delay of 3 hours, and on this account the contemplated stop at Gatun Dam, en route to Panama, was put off until a later day, in order that sufficient time could be given after arrival at Panama for the members of the party to get settled.

The strenuous program for the Canal Zone meeting began on Wednesday at 8 A. M. when the party started on a special train for the Culebra Cut. Here the most profound impression made upon the visitors was by the perfect system of handling the dirt trains. These were kept going with a clock-work regularity, over a bewildering network of tracks in a comparatively narrow ditch, that becomes more narrow as the channel deepens. The system of moving trains so that a minimum of time is lost by the steam shovels, in the switching and hauling to the dumps, and in discharging the loads, impresses the observer. Fig. 2 shows a Lidgerwood unloader en route for the dumps. The plow is on the end of the train and is drawn across the entire length of train by a wire cable. The unloader is similar to a mine hoist. A train of 20 cars can be unloaded in about 25 minutes.

After passing through the cut the special train was taken over a part of the new location of the Panama Railroad, which must abandon most of its present right of way when the water is turned into the canal. In the afternoon the Institute party was given the pleasure of greeting the President of the Republic, Don Pablo Arosemena.

Thursday, November 3, was Independence Day in the Republic of Panama. Those who desired were taken by special train to Culebra for an inspection of working models of the locks at the Pacific end of the canal. In the afternoon the visitors were lookers on at the fiesta, and in the evening were guests of Mr. Saunders at a Spanish presentation of the "Merry Widow" at the opera house.

Starting at 8 A. M. Friday, the party was taken by special train to the locks at Miraflores and Pedro Miguel, these being the steps at the Pacific end from tidewater to the main canal. A visit was made to the pumping station which furnishes water for two "giants" that hydraulic the soft material in the Rio Grande flats and pump the resulting mud out of the canal

channel. In the afternoon a trip by tugs was made to the harbor end of the canal, where interesting subaqueous excavation is in progress. At an evening session, held in the Hotel Tivoli, Colonel Gorgas delivered an address on the sanitary work on the isthmus and the successful fight against dysentery and the yellow fever mosquitos. So completely have the latter been eradicated that there has not been a case of yellow fever on the

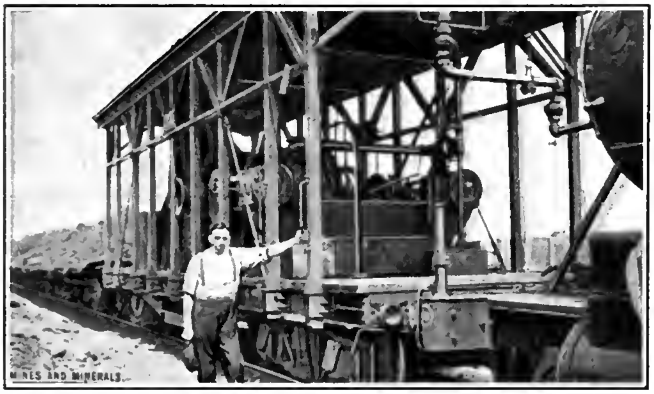


Fig. 2. Lidgerwood Car Unloader

isthmus in 3 years. Colonel Gorgas was followed by Dr. C. W. Hayes, who gave a concise statement of the geologic structure of the isthmus. The session concluded with a lantern-illustrated description of diamond mining at Kimberly, South Africa, and gold mining on the Rand, by Gardner F. Williams, formerly general manager of the De Beers Consolidated Mining Company.

On Saturday the party were tugged to the islands in the harbor and a stop for bathing and luncheon was made at Taboga. Sunday gave an opportunity for a visit to the hospital and a not unwelcome rest. Sunday evening a session of the Institute was held in the hotel. The first paper was by Edward W. Parker, on "Recent Developments in the Undercutting of Coal by Machinery." David B. Rushmon showed some views on the applications of electricity to mining.

The visit to Gatun dam Monday morning was to many the most interesting day of the meeting. The party was taken over the great fill, which will eventually form the dam, and shown through the uppermost of the three locks that will lift and lower the traffic to and from the main stretch of the canal. Tuesday morning the party left Panama for Colon, and at 11 A. M. sailed for New York, stopping en route at Kingston on Thursday for cargo. President Brunton took advantage of this stop to entertain the party at luncheon at the Myrtle Bank hotel. At a session of the Institute held on Wednesday, the Panama Canal was made the subject of discussion. Each speaker was limited to 5 minutes. Absolute unanimity of expression was given to the opinion that the present type of canal is the only one dictated by wisdom and by the conditions existing on the isthmus. High tributes were paid to the men engaged in the construction of the canal and to the sanitation of the isthmus, the feeling being that every man, down to the water carriers, was on "his job." The opinions of the party will be printed in the "Transactions."

On Friday an animated discussion followed a description by C. W. Goodale, of the compressed-air hoisting plant at Butte. This plant is being installed to hoist the ore from 30 mines within a radius of a mile and a half, the power for compressing the air being brought 130 miles from the electric-power plants developed on the Missouri River near Great Falls. The discussion following Mr. Goodale's paper occupied the remainder of the 2-hour session, except for a few minutes allowed D. M. Riordan for reading his sentiments regarding the constructing and constructors of the Panama Canal, which paper was in reality the crystallization of the opinions expressed at the session held on Wednesday. The session on Saturday was given up to a discussion of the draft of a bill which it is proposed to submit to the legislatures of the several metal-mining states in the hope of securing more uniformity in the mining laws and greater safety to mine employes. The bill as submitted was the one prepared by a committee of the American Mining Congress and has the support of that body. W. S. Ayres read a paper on "The Losses Incurred in the Various Methods of Preparing Coal for Market."

Divine services were led by Doctor Raymond on Sunday morning, and in the afternoon the party listened to an interesting address on South America, by Robert P. Porter, of the editorial staff of the *London Times*.

On Monday, November 14, the men in the party signed the following statement:

S. S. Prinz August Wilhelm, at sea,
November 14, 1910

We, the undersigned, members and guests of the American Institute of Mining Engineers, after a visit to the Isthmus of Panama, and an inspection of the work of the United States Isthmian Canal Commission, and after full discussion of our individual impressions, find ourselves in unanimous agreement as to the following conclusions:

1. The present plan of the work is clearly practicable, and the best in our judgment that could be devised under the conditions imposed. It is perhaps a question whether by the choice of a higher level some of the difficulties and uncertainties of excavation in the Culebra Cut might not have been minimized; but a higher level has its disadvantages also; and no one now seriously proposes such a plan. On the other hand, we are convinced that a canal at a lower level, and especially at sea level, is practically out of the question; that no man can estimate its cost, or even guarantee its satisfactory completion and maintenance at any cost. We are satisfied that the sea-level canal, as proposed, if actually completed, would be inferior to the present lock canal, by reason of its necessarily narrow and tortuous channel, its liability to many disturbances from which the lock canal is comparatively free, etc. The experience gained in the Culebra Cut throws additional light upon the sea-level plan, and renders that scheme less worthy of approval by engineers than it was when, with less information, some eminent authorities favored it. In a word, we do not think that any prudent engineer would now recommend the deepening of the Culebra Cut below the level now fixed for it.

Fig. 3. Miraflores Portal of Tunnel on Panama Railroad

The creation of the great Gatun Lake by means of the Gatun dam seems to us to be the best possible way of dealing with the floods of the Chagres and other streams. The location of the Gatun dam, spillway, and locks is singularly favorable for such constructions; and there is, in our judgment, no reason for any anxiety as to their stability.

The one serious remaining problem is presented by the nature of the ground in the Culebra Cut. There have been

extensive slides on the sides of this excavation, and more of them are to be expected; but they involve nothing more than the cost and delay of removing the material which they will force into the cut. They will ultimately end; and we regard as reasonable the calculation of the engineers in charge as to the time and money which they may call for. The results of these calculations are included in the estimates of the Commission as to the cost of the canal and date of its completion.

2. We are unanimous in our praise of the manner in which sanitation, excavation, transportation, and construction are performed with rapidity, skill, and economy. A spirit of loyalty, emulation, industry, and pride seems to animate employes and officers alike. This spirit, so difficult to arouse among workers in tropical climates, is due in this case to two causes: first, the inspiring example of Colonel Goethals and his associates, and, secondly, the splendid work of the sanitation department under Colonel Gorgas. The cities of Panama and Colon, though politically outside of the Canal Zone, have shared in the benefits of the sanitary administration, and reflect an unwonted cleanliness, comfort, and safety.

3. We acknowledge the entire freedom and fullness with which everything we desired to see was shown to us, and everything we desired to know was told us, by the officers of the Commission. There was evidently no wish to withhold or conceal. On the contrary, inquiry and criticism were frankly sought and heartily welcomed.

This is but a meager summary of the points on which we are agreed. The details of individual opinion will appear later in the published report of our discussions. Meanwhile, we unite in this common declaration, which covers our conclusions on all main points. We think the present plan of the canal is good; that the work is in thoroughly capable hands; that it is progressing satisfactorily, and that it will be completed by the date set for it, January 1, 1915, and probably earlier, provided Colonel Goethals and his associates receive the hearty support of the American people, and its representatives in Congress. The canal engineers are the right men in the right place. The great work in which they are engaged is not connected with partisan politics, and citizens of all parties should combine to secure its early triumph and completion. In that consummation every American should take greater pride than in any victory of military or political conflict. (Signatures)

The concluding paper of the session and of the meeting was

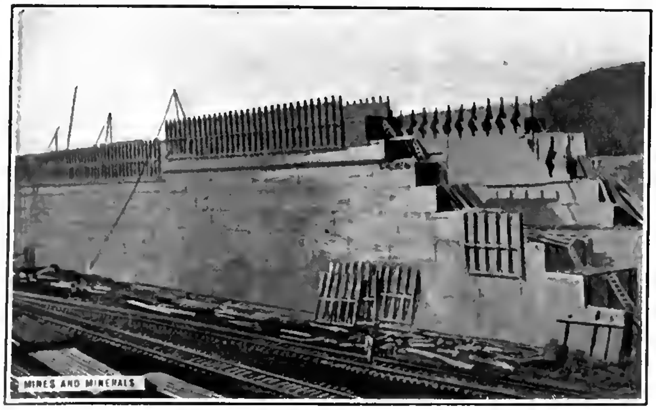


Fig. 4. Section of Pedro Miguel Locks, Pacific Side

on the "Electrical Manufacture of Steel in the United States," by Prof. Jos. W. Richards.

New York was reached Tuesday afternoon, the steamer having been delayed by adverse winds. There had not been a single storm, however, nor had there been one untoward incident during the entire trip. The sea voyages were attended with exceptionally favorable weather, the sessions were all well attended, discussions were full and interesting, and with the week of inspection of the work on the canal, made the ninety-ninth session of the Institute one of the red-letter meetings in its history.

TEPIC TERRITORY, MEXICO

Although the Territory of Tepic, Mexico, has several hundred miles of coast line on the Pacific Ocean, its few ports are

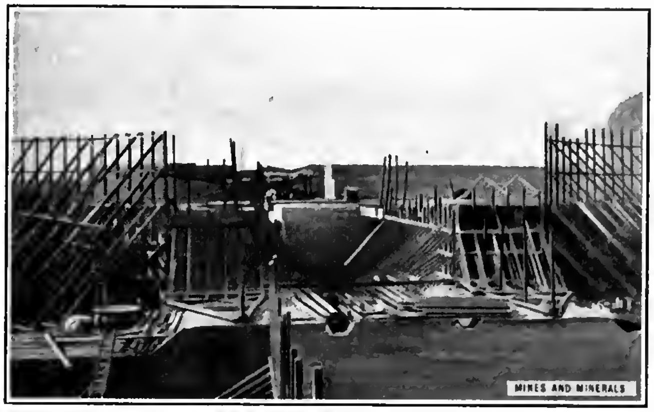


Fig. 5. Lock Construction Work at Gatun

comparatively unknown and cannibals inhabited some of the islands not longer ago than 1900. Immense mountains form its eastern boundary and it is shut off from direct communication with Guadalajara and the south by deep and wide gulches or barrancas.

This isolation has been broken by the Southern Pacific Railway, which has already entered the Territory from the north and proposes to continue building on to Guadalajara across the barrancas, and thus Tepic will have for the first time good communication with the rest of Mexico.

The temperature rarely goes lower than 48° F. in the winter and above 90° F. in the shade in the summer. There are no severe storms and sunstroke is unknown.

Some mines are worked in the Territory, but it is thought that only a beginning has been made, for the mountains which form the eastern boundary of Tepic should be as rich in minerals as those of the same chain in Sonora and Chihuahua.

The plateau lands of Tepic are not as high as those of the tablelands of Central Mexico, of which they form the western boundary, being only about 3,000 feet above the sea. From this plateau there is a gradual rise to the higher lands of Jalisco to the south and east and the natural fall to the Pacific on the west. The soil on this high land is a composite of volcanic ash, and capable of producing good crops, which can be readily marketed in this country when transportation is supplied.

There are few large towns in Tepic. The capital, Tepic, has a population of about 10,000. The chief port is San Blas. Other places of importance are Compostela, Santiago, Ixcuentla, Acaponeta, and Ahauchitlan.

The Southern Pacific Railway connects its Mexican system with its system in the United States at Nogales, Ariz. Plans are being considered for crossing the great barranca north of Tequila, Jalisco, and then entering Guadalajara on the lines of the National Railways of Mexico, which will be met at Orendain, Jalisco, a little northwest of Guadalajara.

The main shaft of the Temiskaming Mine, Cobalt, will soon be the deepest in the district. It is now down more than 460 feet, and is being rapidly sunk to the 500-foot level. At this level the sixth station will be cut. At each 100 feet there is a level. There is also one at 350 feet. Below 425 feet there is no ore in the shaft itself. From the 500-foot station crosscuts will be driven to explore for all the main veins.

A GRAPHICAL NOTEBOOK

Written for Mines and Minerals, by F. W. Gray

Graphic Method of Recording Different Data in Connection with Mining Work

The modern industrial organization is arranged in departments, all interdependent and reporting to one central head. It is manifestly impossible that one man can assimilate all the details of many departments, and the chief must rely upon his departmental heads to take care of these details, wherein, needless to state, is the reason of the departmental system. This has been said many times before, but nevertheless the force thereof is not always appreciated, and many a chief executive dissipates his energies on chasing details, or in doing work which he should require of his subordinates. There are times, however, when the chief finds it is

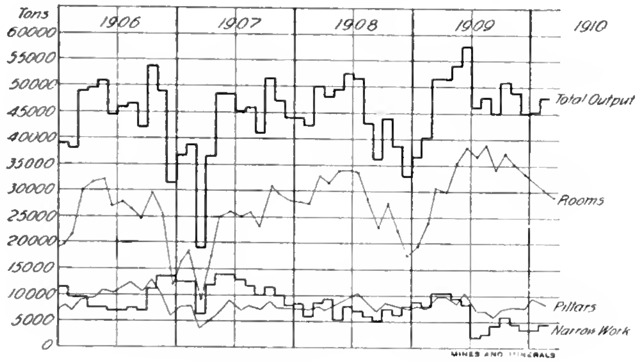


Fig. 1. Dissection of Output of Colliery

very necessary that he should delve into details, and the utility of all the reports and statements which find their final resting place in the office of the general manager is to indicate to him just in what direction he should direct his energies so that they may accomplish the most. The work of the staff immediately surrounding the general manager consists in large part in condensing these reports into such compass, and putting the information contained therein into such shape as will enable him most easily to grasp the essential points, and to judge where personal investigation is most likely to result in improvement. The more complex the operations of a corporation the greater is the necessity of conserving the energy of the head by relieving him of detail, and of so presenting operating results as to utilize to the fullest extent the time of the man who holds the highest

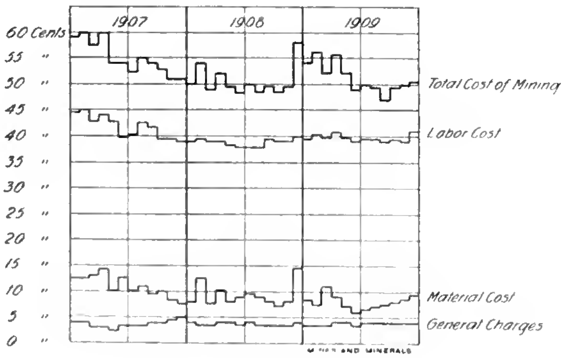


Fig. 2. Cost of Mining Coal

place and is paid the largest salary in the corporation. Therefore it follows that one of the most important parts of the equipment of the chief executive is his notebooks and reference data. These divide themselves into two; namely, compilations of accurate data, more or less bulky, which are usually kept in the private office and deposited for safekeeping in the vault at night, and pocket notebooks which the chief carries about with him in his journeyings around the mines and works under his charge. The graphical notebook affords a means of condensing a large amount of information in a remarkably small compass, and to the general manager whose training has been that of an engineer rather than that of an accountant, the facts are presented in a far more convincing and direct manner than is possible in any other way.

The graphical notebook can be taken out and studied in odd moments of leisure. It also affords a useful reference when discussing the operations of a department with the man in charge, and it can be instantly referred to in corroboration of a statement made or to refresh the memory. The plotted lines in a graphic record show the general trend of affairs, and often-

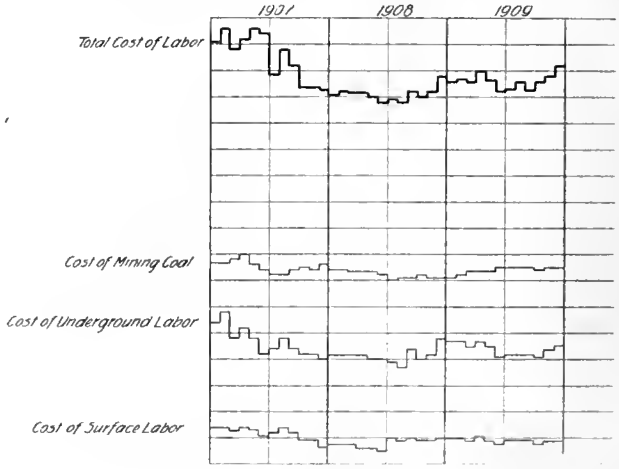


Fig. 3. Cost of Mining. Dissection of Labor Charges

times by the study of the relation of these lines to each other the mind receives illumination that it would fail to get from the study of a mass of figures.

The coal company with which the writer is best acquainted operates about a dozen collieries and is also a transportation company, owning and operating a railway, an electric tram line, numerous loading and discharging plants and a fleet of coal-

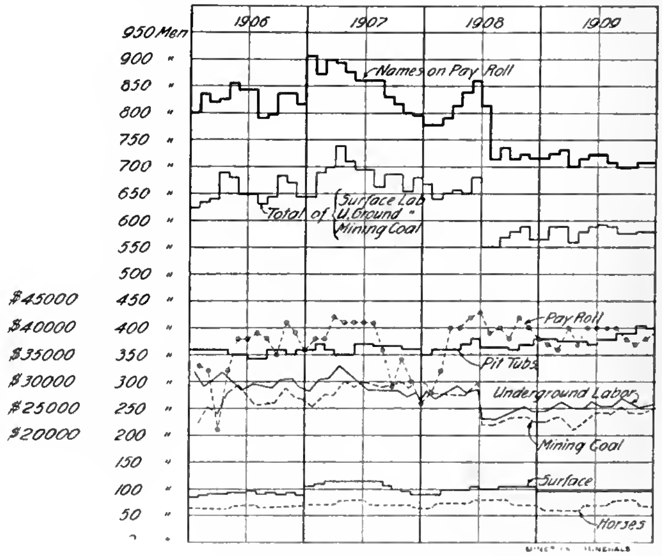


Fig. 4. Organization. Dissection of Mine Force

freighting ocean steamers. The subsidiary operations include a coal washery, retail stores, machine and car shops, foundry, etc. A record which shall contain the essential data of a concern of this nature affords extensive scope for a graphical notebook and for varied methods of plotting. In practice, the labor required in connection with the graphical record is mostly in the initial arrangement of the book. The actual entering up of the record month by month entails small labor compared

with the blocking out of the sections, the arrangement of the graphic scales and in deciding just what is necessary and what is not.

The illustrations accompanying this article are merely intended to be representative. In the actual compilation of a graphical notebook, great use can be made of different colored inks, and some of the records shown cannot be properly appreciated when reproduced in black and white lines.

Figs. 1, 2, 3, and 4 show typical graphs for a colliery, dealing with four of the most important features; namely, output, cost of production, pay rolls, and the distribution of the force employed.

Fig. 1 shows the dissection of the output in a mine worked on the room-and-pillar method, showing how each stage of coal extraction has contributed to the output. An important feature in a mine worked on this method is that the "narrow work," or development work, should be kept well ahead and, also, that the pillars should be extracted systematically to avoid loss by crushes due to allowing them to stand too long. From the diagram shown, the chief could deduce that in the latter months something had gone wrong, as the line denoting "narrow work" has dipped down.

Fig. 2 shows the total cost of production and the relative proportions of its components; namely, labor cost, material cost,

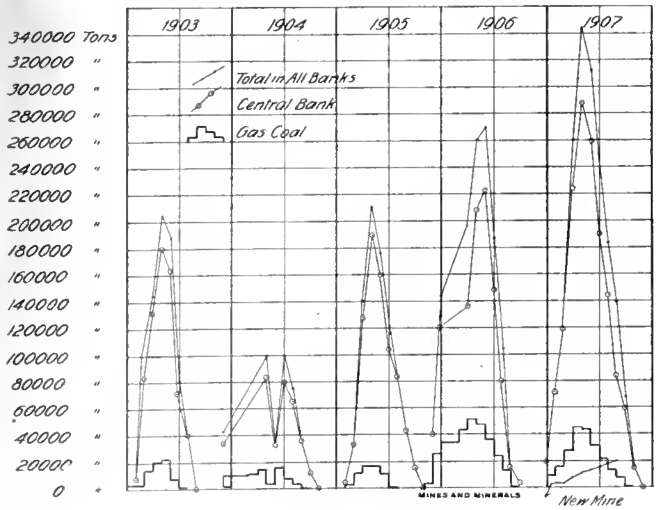


FIG. 5. COAL STORAGE BANKS

and general charges. It will be noticed that there is an increase in the cost during the first four months of each year. During this period it is usual to repair the mines and put them in good order for production of a large output during the summer season.

Fig. 3 shows how the cost of labor is again subdivided into the cost of surface labor, underground labor occupied in the transportation of coal and other auxiliary operations, and the cost of the labor actually engaged in cutting coal at the face. The last-named class are all paid on contract tonnage rates, and the cost line remains very constant. The high cost of underground labor in the first year is noticeable, due to special repair work in the slack time in the spring. While the actual number of men employed on the surface did not greatly vary, owing to the low output during this same period a proportionate increase is noticeable in the cost of surface labor.

Fig. 4 (Dissection of Mine Force) shows the relation existing between the number of men whose names appear on the pay roll and the number of men-days actually worked, these men-days being computed as follows:

$$\frac{\text{Total shifts paid for}}{\text{The number of days mine worked}}$$

The difference between these figures affords a rough indication of how the men are attending their work. The total number of men-days is subdivided, like the cost, into surface men, underground laborers and men mining coal. The relation of the two last-named lines is of peculiar significance, being an indication of the proportion borne by the non-producers or auxiliary workmen underground, to the men cutting coal at the face or the producers. By an adaptation of the scale, the amount of the pay roll can be plotted, as shown, and its relationship to the number of men employed can be followed. If it does not complicate the sheet too much, the number of pit tubs and the number of horses employed in the mine can also be shown, and the one graph will then include the main features of the mine organization and will afford a very interesting record as the years go on. This graph would, of course, show to very much better advantage if lines were plotted in different colors. For example, in the book used by the writer, the lines representing the number of men employed as underground laborers and on cutting coal are plotted in red and blue ink, and their relation one to the other is an unfailing indication of what may be expected in the way of mine costs. Where there are a number of collieries it is usual to keep an "All Collieries" sheet for each separate record, and this sheet can be amplified in the graphical record to suit the larger figures.

Where several seams of different quality contribute to the general output they can be shown graphically as components of the total production and as percentages. In cases where the more valuable seams have been extensively worked for long periods and are becoming exhausted, it is necessary sometimes

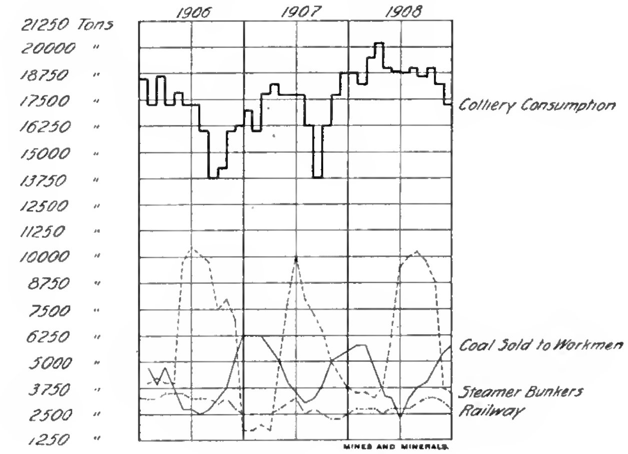


FIG. 6. COMPANY'S COAL CONSUMPTION

to have a correct idea of the proportion of new seams being opened up and their effect on the general output.

The growth of coal-storage banks in the stocking season and their diminution in the shipping season can be well shown. Fig. 5 is an example showing two separate banks at different points. In the last year a third bank is plotted, which accumulated at a new mine before the railroad was put through. The graph shows an increase in the amount of coal stored in bank each year, the gross total being indicated by the peak of the graph, which, it will be noticed, is generally reached in the month of April, at which time the shipping season usually commences. One of the main endeavors of the company, whose stock piles are represented by this diagram, has been to avoid the marked disproportion between the summer and winter work necessitated by climatic and trade conditions. This inequality has, naturally, a bad effect on the organization. The graph shows that the efforts of the company in this direction have been fairly successful.

Fig. 6 illustrates the amount of coal used in "Company's Consumption," showing colliery engine coal, coal used in the bunkers of freighting steamers, locomotive coal used on the railway, and coal supplied to the workmen. The effect of the winter season is seen in the lessened amount of coal supplied for bunkers and the increase in the amount of coal used by the

workmen. Among some of the records that are peculiarly adapted for graphical record are shipments to sales agencies and large consumers, the cost of freighting coal, railway costs, cost of shipping coal, cost of horse keep and many other records which will occur to those who are acquainted with colliery data.

This article has concerned itself more particularly with the notebook of a chief executive, but it does not exhaust the field of the usefulness of the graphical records in colliery work. A colliery engineer will find it useful in keeping a record of his boiler repairs, tube renewals, performance of wire ropes, performance of pumps, etc.

In keeping a graphical record it will be found necessary to give the work to some person having some little knowledge of engineering and draftsmanship, and it will be found necessary to have the plotting done carefully and neatly. It is also well to avoid crowding too many lines into one graph, or carrying the method to an excess and wasting valuable time by arranging and plotting data that is not essential nor useful.

VANADIUM STEEL

Written for Mines and Minerals

To meet the special engineering conditions of recent years, it has become necessary to develop and introduce alloy steels. This has led to the production of a new class of material known as vanadium steel, which may contain, in addition to the usual ingredients, vanadium alone, vanadium and chrome or nickel, and vanadium, chrome, and nickel, and probably some other ingredients as well.

The increasing demand for vanadium steel is due to the high ratio of its elastic limit to its ultimate tensile strength, its great resistance to failure under dynamic stress repeatedly applied, and its great strength and toughness. All of these qualities are necessary requirements in the design of locomotives, automobiles, bridges, and those parts of moving machinery requiring increased resistance to failure through repeated progressive applications of dynamic stresses.

Of the many metals used as alloys in steel for enhancing its good qualities, vanadium occupies the premier position on account of its universal range of applicability in comparison with that possessed by any other steelmakers' alloy. The element vanadium seems to act in several different and well-defined directions, and its incorporation in steel is accompanied by preliminary chemical reactions of the utmost value. It has been noted and confirmed that the increase of strength due to the addition of some method of forming alloy with steel, even if it does not detract from the ductility of the steel to which it is added, or if it enhances the static ductility of that steel, generally renders the metal greatly inferior in its powers of resisting continued repetition of stress, and therefore negatives the improved service value of the material.

Chrome vanadium steels are noted for their great superiority to the maximum resistance which they offer to the breaking-down action of repeated alternations of stress. It is now conceded that the usual tensile-test figures alone do not give any indication regarding the service properties of a metal, and it has been shown that two steels giving the same physical tests for strength and ductility will give widely varying results when placed in use and subjected to moving loads of greater or less magnitude.

The reason for the superiority of vanadium steel is due to at least two causes, namely, the inherent excellence of texture of such steel, and its almost complete freedom from oxides and nitrides, which tend to promote disintegration and gradual rupture under repeated stresses.

Vanadium indirectly toughens steel by sending oxides and nitrides in a fluid form to the slag. Hence any steel containing vanadium is generally free from the occluded gases. Under good normal conditions of operation .7 per cent. of metallic vanadium is sufficient to do this work. Again, under normal conditions vanadium will pass into solid solution in the ferrite or carbonless portion of the iron, which it will directly toughen to a high degree. An alloy containing free vanadium or vanadium combined with another element which will also go into solution in ferrite, such as silicon, must be used, or otherwise direct toughening action will not be obtained. The early experimenters used electrically made ferro-vanadium alloy, but the contradictory results terminated in its abandonment. The high melting point and low degree of solubility of such alloy also added to the difficulties of the steelmaker and contributed to its suppression. It is thoroughly demonstrated by microscopic investigation that ferrites which contain vanadium in solid solution retard segregation of carbides through them to a marked degree. Vanadium steels therefore are susceptible to the improvements conferred by what is generally known as "heat treatment," a term which really comprises the proper annealing and tempering of steel.

The strengthening effect of vanadium on steel is due to the fact that it enters into the formation of carbides of a complex nature, but the strength will largely depend upon the nature of those carbides and the amount of them present for the vanadium to work upon.

Vanadium steel is capable of greater physical improvement by heat treatment than any other class of steel, but this susceptibility to heat changes carries with it the need of performing the heating and cooling with great accuracy. The Carnegie Steel Co. has issued a book on vanadium steel in which is given the minimum physical properties of different types of vanadium steel after heat treatment.

It is stated that to obtain satisfactory results in heat treating vanadium steel, there are three essential features: determination of the correct temperature; an accurate pyrometer for measuring temperatures; and careful heating of the piece to be treated so that the temperature obtained by the pyrometer at any given point is uniform throughout the piece. This means that the element of time must be considered in heating the piece. Most failures in heat-treated steels are caused by not heating uniformly during the treatment. The Carnegie Steel Co. makes suitable grades of vanadium steel to meet the various requirements of the trade. The types are designated alphabetically and the modifications termed "Mild," "Regular," and "Full," are due to variations in carbon content only, the balance of the composition remaining unchanged.

Taking a Type A "Mild," the approximate heat treatment is given as follows: After raising the steel to 1,650° F. it is quenched in water unless the pieces are of a size and shape liable to crack with water treatment when they are quenched in oil, the drawing temperature, however, being lowered about 212° F. After this, the steel is reheated to 930° F. and cooled in air. The physical properties of vanadium steel Type A "Mild" after heat treatment are said to be as follows:

Elastic limit, pounds per square inch, 70,000; tensile strength, pounds per square inch, 90,000; elongation in 2 inches, 22 per cent.; reduction of area, 50 per cent.

Taking again the same brand of steel, raise its temperature to 1,650° F. and quench in water as before. Reheat this steel to 1,290° F. and cool in air. The material after treatment will give, elastic limit, pounds per square inch, 40,000; tensile strength, pounds per square inch, 60,000; elongation in 2 inches, 35 per cent.; reduction of area, 65 per cent. This 30,000 pounds difference between the tensile strength of the same kind of steel is due entirely to heat treatment. From this it will be seen that it is impossible to give a universal strength figure for vanadium steel, and since it may be alloyed with all grades of steel, from the mildest boiler plate to the hardest tool steel, it is safe to say that the strength imparted by vanadium is equal to that of any other alloy, besides it possesses toughness to a greater extent.

GOLD DEPOSITS OF SAN JUAN, COLO.

*Written for Mines and Minerals, by Warren C. Prosser**

Geology of the Country With a Description of Some of the Veins and Mines

The first company to mine in what is now San Juan County, Colorado, was the Little Giant Mining Co., organized in Chicago in 1872. The Little Giant was a gold property and the $150-per-ton ore was treated in arrastras from which the gulch in which they were located was called Arrastra Gulch.

Subsequent developments were mainly on sulphide ores, principally argentiferous tetrahedrite and galena, and Silverton was named as the result of an extensive production of these ores. Lixiviation and smelting works were erected in many places in the county at an early date and several sampling works were operated. Ores were readily marketable and the production increased with the decrease in freight and treatment charges.

Up to 1893 very little gold was paid for in the ores, by the sampling works. A few mines were able to mine gold quartz, in which the native gold was plainly visible, such as the Sunnyside, Sunnyside Extension, and Gold King, but the practical exploitation of gold has been left until today.

San Juan County covers the crossing of the San Juan range with the Rocky Mountains and has for a portion of its eastern border the Continental Divide.

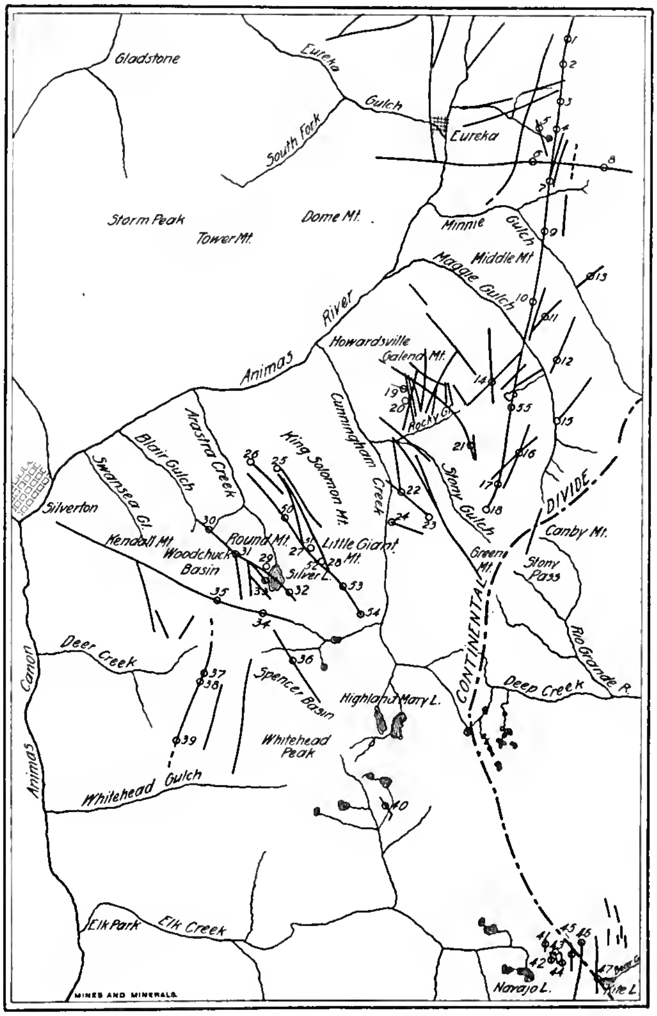
The fissuring attending the formation of the several ranges is complex and greatly diversified.

The directions in which the fissures predominate are northeast to southwest; northwest to southeast, and north to south. The northeast to southwest and the northwest to southeast fissures are attributed to compressive stress, acting along the range of which they are a part in a nearly north and south direction. In the Silver Lake and Galena Mountain districts, the northwest and southeast fissures contain ores of galena, sphalerite, chalcopyrite, pyrite, and tetrahedrite, associated with considerable silver and free gold, usually in the oxidized areas near the surface.

Toward the northern part of the section, the north and south fissures, which are large and well mineralized, are evidently the oldest, as they are faulted by the others.

Besides the stresses in an east and west direction which produced the north to south fissures, and those in a north and south direction which produced the other two sets, minor stresses produced smaller fissures in great numbers.

In the section extending from Hunchback Mountain and the Vallecito River, on the south, to Woods Mountain on the north, silver and gold ores occur. This area is extensive, and, aside from the mines of Silver Lake Basin, Galena, and King Solomon mountains, has been hardly prospected.



MAP OF SAN JUAN REGION, COLO.

The rocks of Bear Creek and Kite Lake sections are Algonkian quartzites and slates in contact with Archæan schists and gneisses, and partly covered with Tertiary andesites and rhyolites.

The fault contact to the north between the quartzite and schists is marked by a long narrow dike-like block of quartzite. The white quartz veins cut through the slates and quartzites at nearly right angles to the folds and faults. They are more clearly defined in the quartzites than in the slates.

The Gold Bug Mine is located on the northwestern shore of Kite Lake. The main tunnel is driven over 900 feet to the Repeal vein, on which a raise has been driven to the surface. The Gold Bug vein was cut at 600 feet in this tunnel, and has been partly developed by shaft from the surface. Sylvanite comprises the principal part of the ore and occurs in pockets of 6 inches thickness or less. Several levels started above the cross-cut, which have all encountered ore, show the existence of ore shoots in the veins at several places. At the Kankakee Mine, to the southeast of Kite Lake, considerable sylvanite and calaverite ore was found while drifting on a vein. The vein is of brecciated quartzite, with the interstices filled with secondary quartz and the telluride minerals disseminated through it. The ore increases in value and quantity as the drift approaches a contact with schist.

In the Hercules property, close to Beartown, the same kind of ore has been found in a porphyry-rhyolite contact.

Other properties in this district are the Sylvanite, Good Hope and Wonder, Robinson, Camp Bird, Golden Shear, Summit, and Little May.

The Gold Mines and Power Co. property is located at Navajo Lakes, across the range, and to the west of Kite Lake. Here a power site is located and the equipment is on the ground to supply this district with electrical energy. The development of this district has been slow, mainly on account of its inaccessibility and the necessity of hand work in hard rock.

The following assays show the proportions of gold and silver in the ore:

	Gold Ounces	Silver Ounces
Gold Bug Mine	158.5	282.5
Kankakee	5.0	1,160.0
Kankakee	20.0	56.0
Hercules	34.0	1,176.0

To the north for several miles are mountains of gneiss and schist, covered at the high altitudes by andesites and rhyolites. The northern contact of this schist is at Mountaineer Creek,

* Mining Engineer, Silverton, Colo.

where there are a set of fissures, one of which traverses the country for many miles across the length of King Solomon Mountain, and has been mined extensively, producing considerable tetrahedrite carrying from 20 to 800 ounces in silver.

Mountaineer Creek heads in Spencer Basin, where considerable free gold in panning quantities has been found on the Gold Lake claims. The gold occurs in quartz veins and stringers in an iron-stained rhyolite. Here wires, nuggets, and flakes of gold are in the quartz and scattered through the spaces between the interlocked crystals.

To the west of Spencer Basin, in Deer Park, considerable free gold has been found in pockets in quartz on the Mabel, Little Ray, and Titusville veins.

In Silver Lake Basin, beside many intersecting fissures, one main fissure crosses the lake, and trending northwest passes through Woodchuck Basin and into the head of Blair Gulch. On this fissure are located the Royal Tiger, Silver Lake, Whale, and Nevada mines, all of which have produced considerable gold. Some of the ore in the Tiger Mine gave returns of 34 and 62 ounces in gold per ton. It was found in the hanging wall to which the sulphide vein had been clinging.

On the northwestern slope of Hazelton Mountain, by assiduous prospecting gold was found in the summer of 1909, on the Big Bull vein, bordering the Aspen Mine, which has been working since 1872. Here the valuable minerals are in quartz veins in porphry dikes.

The Little Giant and Big Giant mines produced gold during the 70's and 80's. Both the veins were apparently minor fissures, branches of the Mayflower, Shenandoah, and Highland Mary fissures. It is singular that these large fissures should have contained gold, while the main fissures produced principally silver.

Galena Mountain, a peak of andesite, surmounting a deposit of Eureka rhyolite, contains many large fissures of varying width, several of which are distinctly visible from Silverton, 5 miles to the southwest. The Old Hundred Mine includes a number of these important fissures, which were developed in 1905–7. From the upper levels $30,000 to $40,000 in gold was caught on the plates of a 40-stamp mill in 30-day intervals.

The dump of No. 2 level of the old Green Mountain Mine shows free gold, which had been overlooked by those who operated the mine in the past. It occurs in the upper oxidized part of the vein, above the sulphides in the lower part of the mine.

In Stony Gulch several important strikes show gold and silver in considerable quantities.

View in San Juan Co., Colo

In the Summit Mine, at the head of the gulch and on the crest of the Continental Divide, ore has been found carrying 4 to 5 ounces in gold and 400 ounces silver.

The Stony Pass Mining Co., just to the northwest of this place, when operating a large quartz vein cut into some gold and silver ore in sinking a shaft, and being forced to abandon the shaft on account of water, a tunnel was driven down the hill to come under the shaft. Where the vein was met in this cross-cut good ore was found, and on driving across the vein 12 feet a concentration of the minerals on the foot-wall was found.

In Stony Gulch the ore is different from the deposits of Bear Creek, or in the properties surrounding Kendall and King Solomon mountains. It consists mainly of finely disseminated argentite and tetrahedrite carrying gold and silver. Some of the mineral taken from the Stony Pass property is sectile and under the blowpipe yields metallic silver readily.

In the Buffalo Boy Mine, in the same gulch and near this property, 5 feet of argentite, profusely sprinkled with proustite, forms an ore worth from $400 to $500 per ton. This was met while drifting on a quartz vein from 6 to 12 feet in width.

To the north of the Buffalo Boy, the Golden Egg group has a deposit of argentite that contains 2 ounces in gold and from 30 to 700 ounces in silver per ton. The quartz in this vicinity is white and the minerals are in bluish streaks and narrow bands which look very little like ore.

The continuation of this vein is judged to be the Gold Nugget vein on the northeast slope of Maggie Gulch on Middle Mountain. Here the values are in argentite, although some free gold occurs in iron rust spots, scattered through the vein matter. Ten thousand dollars was taken from one pocket on the surface of this property.

On the Maggie Gulch slope of Middle Mountain, four prominent veins showing outcrops traverse the side of the gulch diagonally. These are the Gold Nugget, Dewey, Little Maud, and Victor Hugo veins. These fissures are roughly parallel and the vein matter and argentite in them is similar in appearance. They will average from 10 to 12 feet in width.

The Ridgeway vein, which outcrops on the ridge above Crystal Lake, has been developed beyond the prospect stage. It has produced many thousands of dollars, mainly to leasers, operating $40 to $50 ore, streaks of which run 5 ounces in gold and 158 ounces in silver. The ore is chiefly argentite with free gold scattered throughout the quartz gangue in bluish and black streaks in which the ore minerals are scarcely visible to the naked eye.

At the head of Maggie Gulch where it begins to rise abruptly to the foot of Canby Mountain, the Intersection Mine is located on three intersecting veins. A long tunnel has been driven on the Maggie Creek vein and a shaft sunk near the intersection of the Maggie Creek and Iron Mask veins to a depth of 160 feet. The vein is drifted on both ways from this shaft and some stoping has been done above this level. The ore is finely divided argentite disseminated through the quartz gangue, giving it a dark bluish appearance. It is associated with pyrite. The veins are from 3 to 6 feet wide. The property has a five-stamp mill and operates a vanner and concentrating table, which produces a heavy pyrite concentrate, besides amalgam on the plates.

Farther down Maggie Gulch is the Victor Hugo Mine. The vein at this mine is white and yellow quartz from 3 to 12 feet wide, and outcrops on the slope of Maggie Gulch and over on the other side of Middle Mountain in Minnie Gulch. It has been opened on three levels from which considerable $40-per-ton ore was extracted by stoping to surface from the third level. The ore is argentite, associated with iron oxide, pyrite, and free gold. The lower tunnel, in over 900 feet, shows white quartz for over 800 feet, when the formation changes abruptly from a rhyolite tuff to a hard pyroxene andesite. The ore is said to be better and more stable after it passes into the andesite. A mud streak almost at right angles to the vein shows the fault line between the two formations.

Between the Victor Hugo and the Gold Nugget veins is the Little Maud Mine, located on a large quartz vein roughly parallel with them. This mine, opened by a long tunnel, has been developed mainly by leasers, who have worked some rich argentite ore containing more or less gold.

In Minnie Gulch the extension of the Gold Nugget vein is

probably the All O. K. which outcrops in the cliffs on Crown Mountain. At the point where it is first exposed near the base of the cliffs an ore shoot was found that gave assays of 22 ounces in gold and 127 ounces in silver per ton. Several tons taken from this place returned $86 per ton from the smelter. About 600 feet farther up the hill ore has recently been found assaying 2.65 ounces gold and 400 ounces silver per ton. On this property the vein is from 5 to 20 feet wide and is held in andesite walls. It is faulted by several east and west veins and veins of the other two general sets, showing it to be a member of the oldest family of fissures. To the east of this vein is the Parallel vein, which is 25 feet wide, of solid quartz. Its course is slightly divergent from the O. K. vein, and as they pass over the crest of Crown Mountain and down into Niagara Gulch the Parallel vein is several hundred feet to the east. On the northern slope of Crown Mountain, the O. K. vein outcrops down the mountain side where it is hardly accessible. It continues over Niagara Mountain and through the Golden Hammer and Grand Potentate veins. On Jones Mountain it is prominent, and on the ridge is nearly 50 feet wide, showing about half and half white quartz and calcite. It descends into Burns Gulch on the Broadgauge vein, where considerable sulphide ore was found with free gold. It has been traced up over the succeeding mountains through Grouse Gulch and Cinnamon Pass into the faulted area of Woods Mountain.

It is noticeable that the east and west veins, crossing the north and south fissures are heavily laden with sulphides as a general rule. In Silver Lake Basin and on King Solomon Mountain this may be observed in the Silver Lake-Tiger vein and on the North Star-Shenandoah vein. On Galena Mountain the northwest to southeast veins contain sulphides, as do also the Pride of the West and the old Green Mountain veins. In Maggie Gulch, near its mouth, the Hamlet vein, one of this set, is strongly mineralized. It has been explored 1,400 feet in vertical depth and 1,800 feet into the hill, disclosing heavy copper and zinc sulphides.

In Minnie Gulch the Tantallon-Joyce vein of the Peerless, San Juan, and Kittemac mines shows abundant lead, copper, and zinc sulphides, and in Niagara and Burns gulches are more of the same kind.

Two periods of dynamic action are represented by the most important veins of this section. The north and south set were probably formed by the upheaval of the San Juan Range and the second set by the rearrangement that followed. That a third disturbance followed is evidenced by the veins on King Solomon Mountain, Galena Mountain, and other places along this section, which fault both the other sets of fissures, but which lend little to the mineralization of the section.

Cyanide tests made on the ores of the Bear Creek region, from the Buffalo Boy, Ridgeway, Intersection, Gold Nugget, and Esmeralda mines indicate over 90 per cent. recovery. There appears to be no difficult metallurgical problem involved in the cyanidation of the gold ores of this section, as none of the undesirable factors are present in amounts that would seriously interfere with the process.

Below are several assays from the mines of the Bear Creek section, which shows the relation of the gold to the silver, but not the average gold and silver content:

	Gold Ounces	Silver Ounces
Summit Mine	5.00	400.0
Buffalo Boy	3.80	200-3,400
Golden Egg	4.68	67.6
Intersection	25.00	
Victor Hugo	4.00	1,160.0
Little Maud	80.90	1,160.0
Little Maud	11.50	80.0
Gold Nugget	28.00	1,333.0
All O. K.	22.00	127.0
Ridgeway	5.50	158.0

This paper does not cover all the mines and good prospects of this section, but includes some of the most important to show the existence of an area mineralized with the gold and silver. The interest displayed in San Juan County is bringing out almost daily important features which shed more and more light upon the ore occurrence.

METAL MINE VENTILATION

Good ventilation is not merely a legally recognized necessity for the preservation of health, but has an important economic aspect in its effect on the industrial efficiency of the workman and the cost of mineral production.

The Final Report of South Africa Mining Regulation Commission

The necessity for ventilation in metalliferous mines arises mainly from the injury to health caused by the presence in the mine air of fine dust, of the poisonous gases produced in blasting, and of human emanations, respiratory and other.

The ventilation of mines may be (*a*) natural, and (*b*) artificial or mechanical; natural ventilation depends upon (1) the difference in temperature and humidity, and consequently in weight, between the air in and out of the mine; (2) the absolute humidity of the outside air; and (3) the difference of level between the mouths of the connected shafts; natural ventilation has the advantage of being inexpensive, but is subject to the following drawbacks; viz., (1) the amount of air supplied depends upon conditions beyond control, and independent of the varying mine requirements and hygienic necessities; (2) as the temperature and humidity of the outside air approximate to that of the mine air, the ventilating current diminishes and may completely disappear; (3) the direction of the air-current is variable.

Natural ventilation is supplemented on the Witwatersrand by the exhaust air from machine drills; this supply is of great local importance as it is delivered at the working place, but the amount only constitutes a fraction of what is required for the proper ventilation of a mine, and its beneficial effects do not extend for more than 100 feet from the face; some miners are prejudiced against compressor air, though there is no evidence that it is injurious to health; it stirs up from the sides and floor of the working place the dust which is the principal cause of miners' phthisis, and occasionally contains some of the poisonous gas *CO* derived from the accidental combustion of the oil used for lubrication.

Judged by the relative frequency of gassing accidents, natural ventilation is on the whole not so effective on the Witwatersrand as on some other fields, this inefficiency being probably due to (1) the very slow rise of the rock temperature as depth increases; (2) the high temperature and humidity of the outside air at certain seasons; (3) the great extent of the underground workings and large number of men at work; and (4) the large consumption of explosives in blasting, and consequent production of great quantities of poisonous gases.

Artificial or mechanical ventilation is effected by means of either (*a*) the extractive force of a furnace at the bottom of the upcast shaft; or (*b*) fans—usually exhaust fans—at the top of the upcast shaft; at the East Rand Proprietary and Cinderella Deep Mines the total running cost of an extremely effective installation is under 1d. per ton milled.

The degree of ventilation in a mine may be measured either by (*a*) a quantity standard, that is, the quantity of pure air entering the mine per minute; or by (*b*) a quality standard, this being a determination of the amount of impurity present; the existing Transvaal law provides for a quantity standard of 70 cubic feet per man per minute, and also for the splitting of the current and distribution of the air over the working faces; the application of this standard is open to very serious practical difficulties; the provision as to distribution is not enforced,

probably because attempts to direct natural air-currents usually merely retard them, and the quality standard is better adapted as a general indication to mine inspectors than as a basis for legislation.

The object of a quality standard is to fix the permissible amount of air vitiation, and for this purpose the quantity of carbon dioxide present is accepted as bearing a roughly constant proportion to the amount of impurity present.

Our knowledge of the exact nature of such impurity is incomplete; there is no reliable evidence to support the former belief that during respiration the lungs exhale a volatile organic poison; the immediate ill-effects, headache, general discomfort, etc., of bad ventilation are probably the results of the oppressive smell which arises from the breath, bodies, and clothes of those present, and is caused by very minute quantities of volatile substances present in the air.

According to Pettenkoffer (1858), Dr. J. S. Haldane, F. R. S., and other modern observers, this smell becomes distinctly perceptible when the carbon dioxide CO_2 in the air exceeds 10 volumes per 10,000; but in 1875, De Chaumont recognized it in barrack rooms when the CO_2 reached 6 volumes per 10,000.

Continued subjection to bad ventilation means increased liability to disease, and especially to infection produced by organisms present in the mouth and air passages, and conveyed directly through the air from person to person.

Doctor Haldane, F. R. S., who has closely investigated the subject and is the foremost British authority thereon, considers that the worst consequences of a defective air supply are "the evil effects produced by inhaling poisonous or infective dust"; he doubts that "constant exposure to volatile respiratory impurities has by itself a very great influence on health," and agrees that carbon dioxide is "the best objective criterion of the sufficiency of ventilation."

In mines, the highly poisonous gases, carbon monoxide CO and nitric oxide NO_2 are also present, the result of the detonation or of the burning of explosives; in dead ends, and immediately after blasting, the ratio of CO_2 to CO in the gases produced by explosives in local use averages 1 to 12, and therefore CO_2 serves in some degree as an index of this danger (presence of CO).

Carbon dioxide, unaccompanied by any injurious substances, is in itself innocuous in quantities under 1 per cent. Of this nature are the normal atmospheric CO_2, amounting to 4 volumes in 10,000 of air, the CO_2 produced by open lights and by the action of acids on carbonates.

The difficulty in fixing a quality standard lies in the impossibility of distinguishing between the noxious and innocuous CO_2 when both are present together.

On the Witwatersrand the only innocuous CO_2 known to exist in appreciable quantities is the normal atmospheric CO_2, and that produced by open lights.

Exhaustive inquiries made by the Mines Department show that a considerable body of carbonates which could give rise to CO_2 by the action of acid water, is present in the workings of one mine only, the remainder having either no carbonates at all, or only inappreciable quantities in the shape of flakes in fault planes, or as an occasional stringer adjoining a dike; in view of the minute percentage of acid in mine water, there would generally be in these places a sufficient excess of water to hold in solution the CO_2 generated, and as mine air is nearly saturated with moisture, it is improbable that any dissolved CO_2 would subsequently be released by evaporation.

In the Lydenburg and other districts, and in base metal mines, masses of carbonates occasionally exist in proximity to very pyritic reefs, and a considerable amount of innocuous CO_2 may be locally produced.

It was suggested to us that additional quantities of CO_2 might be produced from the following sources; viz., (1) "ground" gas contained in rock cavities, or occluded in quartz pebbles in the banket and in the grains of quartzite; (2) from the oxidation of food and timber, evaporation of stagnant water, and decomposition of animal waste.

We find that cavities containing CO_2 are practically unknown in the Witwatersrand metalliferous mines; that the occluded gases would not be liberated under the conditions of underground work, and are in any case noxious, containing a large percentage of the poisonous CO; that the oxidation of timber and food would produce quantities of CO_2 too small to affect any standard of vitiation, and that the gases from stagnant water and animal waste are inappreciable and offensive or noxious in character.

Careful experiment was made at the Langlaagte Deep Mine, under the supervision of the Mines Department, to ascertain whether any production of "ground" CO_2 could be inferred from the difference between the estimated amounts of CO_2 entering and leaving the mine; the possibilities of error in the assumptions upon which the estimates were based appear so considerable, and the results calculated on different assumptions so widely discordant, that we regard the various Langlaagte Deep results as inconclusive, and in this view we are supported by the Government Mining Engineer.

In the absence of any theoretical source of any appreciable quantity of "ground" CO_2 in the Witwatersrand metalliferous mines, we consider that practically the whole of the noxious CO_2 is due to respiration and explosives, including fuse, etc., and that the innocuous CO_2 is derived from the atmosphere and open lights only.

The limits for noxious CO_2 previously recommended or embodied in legislation are as follows:

(*a*) Five volumes per 10,000, by the Roscoe Committee (1896) on the Ventilation of Humidified Factories. This was legalized under the Cotton Cloth Factory Act, and also in the Factory and Workshops Act, 1901.

(*b*) Eight volumes per 10,000, by Haldane's Committee (1902) on Ventilation of Factories and Workshops. This standard now regulates all factories, including humidified, in the United Kingdom.

(*c*) Eight volumes per 10,000, by West Australian Mines Regulations Act.

(*d*) Six volumes per 10,000, by New South Wales Mines Act.

(*e*) Eight volumes per 10,000, by Victoria Mines Act, 1907.

Because we cannot definitely state that there is no material quantity of "ground" CO_2 in the mines of the Rand, though we believe that the actual amount, if any, is small, we propose that a working allowance of 5 parts in 10,000 be made for it, and for CO_2 from other uncertain possible sources, with the sole object of fixing a standard which is practicable from the administrative point of view, and which will enlist the voluntary cooperation of the mines in its enforcement; and, in view of our recommendation (see below) as to a CO limit, and as to sectional and local ventilation, the prevention of dust and fumes, the total limit of 20 parts of CO_2 per 10,000 is well within the limits of safety, is reasonable and easily obtainable, and should be enforced.

We therefore recommend as follows (see Draft Regulations 56–63):

GENERAL VENTILATION

(*a*) That the legal maximum for noxious CO_2 permissible in mines in the Transvaal be fixed at 8 parts by volume in 10,000 of air.

(*b*) That an amount of 4 parts of CO_2 by volume in 10,000 of air shall be allowed in addition to the aforesaid maximum as representing innocuous CO_2 normally present in the atmosphere.

(*c*) That where candles or similar illuminants are in use, a further addition of 3 parts of CO_2 by volume in 10,000 of air shall be allowed as representing innocuous CO_2 resulting from the combustion of such illuminants.

(*d*) That in order to meet, from the point of view of practical administration, difficulties in regard to possible innocuous

CO_2 from "country rock" and other uncertain sources in the mines of the Rand, a further allowance of 5 parts per 10,000 be made, making a total limit of 20 parts of CO_2 per 10,000 of air.

(*e*) That in the Lydenburg and other districts where there is geologically strong presumptive evidence of a production of ground CO_2, early investigation be undertaken by the government, and that a proper and reasonable allowance be made therefor, the total amount in the mine air not to exceed 1 per cent. by volume.

(*f*) That all samples for testing purposes under these provisions be taken not less than one hour after blasting.

(*g*) That each mine be informed of the results of any official analyses of the air therefrom, and notified that the ventilation is defective when the above proportions have been exceeded, and at the same time supplied, so far as practicable, with information as to the nature of any defect noticed; and that legal proceedings be not taken against a mine unless, after a reasonable interval following such notice, the stated proportion is found on examination of one or more samples to be again exceeded, and the mine is unable to show that steps have been taken reasonably calculated, in the opinion of the Government Mining Engineer, to secure the requisite ventilation.

(*h*) That any analysis on which a prosecution immediately depends shall be made by a specially qualified person.

(*i*) That arrangements be made for inspectors of mines to have the use when desired of a properly tested portable apparatus for estimating on the spot the proportion of CO_2 in the air.

With regard to the very interesting question as to the necessity for correcting the proposed CO_2 limit for the altitude of the Rand, the evidence is somewhat conflicting, and many samples will be taken at considerable depths, and therefore we do not recommend any addition on this account to the proposed standard.

In view of the extremely poisonous effects of CO and NO_2 on the human system, and the frequency of gassing fatalities on the Rand, we also recommend that the maximum permissible amount of CO in any part of a mine shall not exceed .01 per cent., and no practically determinable amount of NO_2 shall be permitted in any part of a mine.

SLIME SETTLER OR DEWATERER

*By Rollo E. Huntley**

The settler here described has been in use at the Brownhill Consols Gold Mine, Kalgoorlie, W. A. for the past 18 months, handling the product of a public crushing mill, varying from exceedingly light oxidized, to dense sulphide, ores, and has always given satisfaction. The points to be noted in its favor are: (*a*) Cheapness; (*b*) efficiency; (*c*) simplicity of erection; and (*d*) very little attention necessary.

The settler consists of a circular vat with central inflow of the pulp, and peripheral overflow of the clear water. The thickened slime is discharged by means of a centrifugal pump drawing from the center of the bottom.

The vat is made of 22-gauge corrugated galvanized iron, 20 feet in diameter, by 5 feet deep, with plain galvanized-iron bottom.

It is also provided with a false bottom in the shape of an inverted cone made of 1-inch jarrah†, with a fall of one in nine to the center. This false bottom is made of 6″×1″ jarrah, in 9′ 2″ lengths, sawed diagonally from ½-inch wide at one end to 5½ inches at the other. The large end is placed in the corrugation in the side of the vat, 12 inches above the real bottom, and the small end is butted up against a 2-foot diameter ring of 1-inch square iron, bolted in the center of the bottom.

These jarrah sectors are spaced about ⅛ inch apart, and allowed to slime up underneath, thus forming a solid conical bottom.

The overflow from the sand collecting vats is run to the middle of the settler, where it is received in an "inflow bucket" consisting of a 9-inch diameter funnel, which leads the pulp 6 inches below the surface of the settler when full. The funnel is surrounded by a 15-inch diameter bucket, the overflow of which is 3 inches below the surface.

This "inflow bucket" is most essential, as it secures a quiet and evenly distributed flow of the pulp into the settler, thus preventing any disturbance of the already settled slime.

The clear water flows over the entire rim into an annular launder made as part of the vat. To insure an even overflow, 6″×½″ pine matchboard is bolted round the periphery and leveled with a plane as desired.

When full of thick slime to within a few inches of the top, the inflowing pulp is sent to another similar vat. After settling for half an hour, any clean water is siphoned off, and the sludge transferred to the agitation vats, by a 4-inch centrifugal pump, driven by an 8-horsepower motor, with power from the Kalgoorlie Electric Power and Lighting Corporation.

The pump has a 4-inch suction and two 3-inch deliveries. It draws from the apex of the inverted cone bottom, and delivers part of the sludge to the filter-press agitation vats, and part back into the settler to dislodge and mix the densely settled slime. The amount sent to either place can be regulated by means of a valve on each delivery.

While the settler is being emptied, a jet of cyanide solution is run in to thin down the slime.

Instead of using a valve, the suction is opened and closed by means of a wooden plug at the end of an iron rod, which passes through a guide near the bottom of the vat and projects above the surface near the inflow.

When milling with fresh water, such a settler will give a clear overflow from five stamps on oxidized, and eight to ten stamps on sulphide or clean quartz ores. The use of lime nearly doubles the capacity for settling, but the resulting slime holds more moisture.

Running on average Kalgoorlie oxidized ores, the overflow from the sand collectors contains from 4 to 8 per cent. of solids, and is thickened in the settler to from 50 to 70 per cent. of solids.

The sludge is always thinned down with cyanide solution during the process of emptying the settler.

The time required to empty one vat containing from 30 to 40 tons dry slime is 1½ hours, and the cost when handling 900 tons per month is 2½d. per ton from mill to press.

The only attention necessary is that of one man for 1½ hours when emptying.

From a metallurgical point of view it is a success, the agitation through the centrifugal pump and the aeration caused by pumping some of the sludge back into the settler both tending to improve the extraction.

The cost of emptying could be considerably reduced by placing the settlers over the agitation vats, and discharging by gravitation, but it is doubtful if such saving would not be at the expense of the extraction in places where only mechanical agitation is used.

Vats of greater depth and diameter could be economically used, but would probably need larger pumps, and would either have to be made of steel, or corrugated iron buttressed with sand, as these vats in larger sizes than 20 feet tend to go out of shape when full.

*Australian Mining Standard.

†An Australian tree which is not attacked by any of the usual organisms of and or water that produce decay in wood.

TOTAL GOLD PRODUCTION OF ALASKA

The total gold production of Alaska at the close of 1909 was $162,686,455, of which $118,219,757 came from placers and $44,466,698 from lodes. The record of production begins with 1880.

ZINC MINING AT YELLOW PINE, NEV.

Written for Mines and Minerals, by Newman B. Gregory, B. S.

In Clark County, Southern Nevada, about 45 miles south and west of Las Vegas, is the Yellow Pine mining district, an area which, owing to the character of its ore deposits, presents a new feature in the western mining industry.

The Discovery and Development of Large Deposits of Zinc Carbonate

Ever since the Mormons discovered the Potosi Mine, in about 1860, mining of lead-silver ores has been carried on in a fitful manner. Until the advent of the Salt Lake Railroad in recent years, the remoteness of the district from railway and smelting facilities precluded the shipment of any but the best of these ores.

The discovery of oxidized zinc ores at Leadville 4 months ago in drifts which had been cut for over 20 years, and the oxidized ores plainly disclosed, but unrecognized by mining engineers, geologists, and mineralogists, illustrates the limitations of ordinary human discernment; and the discovery of zinc ores in the Yellow Pine district is only another example of such limitations. Deposits of zinc carbonate and silicate ores equaling any known deposits of their kind in the world occur in the Yellow Pine area, and it was not until 4 years ago that the true character of these deposits was accidentally discovered.

FIG. 1. YELLOW PINE MINE, YELLOW PINE, NEV.

In mining the lead-silver ores the miners found dikes of a brittle, light-colored material which they took for granted was nothing but vein matter, and carelessly threw it on the dump piles. This "vein matter" was eventually identified as zinc ore, which, on being assayed, showed a high percentage of zinc. An investigation was started, innumerable samples assayed, and it was discovered to the astonishment of every one in the district that the mines contained 10 tons of zinc ore to 1 of lead.

Outside of Goodsprings, the town nearest the mines, and the few neighboring camps, this discovery created almost no excitement, because, with the nearest zinc smelter some 2,000 miles distant, and the nearest railroad a dozen miles away, these ore bodies seemed to be valueless.

In the face of these discouraging conditions a few of the operators decided to determine the possibilities of the newly-discovered ore bodies, believing that with the opening up of a large quantity of an ore which is very much in demand, reduction works would necessarily be erected, thus making of these oxidized ores a commercial product.

Continued development work attracted the attention of the zinc smelters in the United States, and in 1906 the first carload of zinc ore, consigned to Kansas points, was shipped out of the district. Since that time probably 400 or 500 cars have been shipped to the Kansas zinc smelters, and the general average of these shipments has assayed in the neighborhood of 40 per cent. zinc. When it is considered that this production has been made in the face of high wages, almost without the use of machinery, a long and expensive wagon haul to Jean and other shipping points on the Salt Lake Railroad, and a freight rate of $8 per ton, results and achievements of the miners of Yellow Pine district show conclusively the richness of these newly-discovered zinc deposits.

A large tonnage of clean zinc ores has been developed in several of the mines, but an even larger tonnage of mixed zinc-lead-silver ores has been found, these ores assaying on an average 30 per cent. zinc, 15 to 20 per cent. lead, and 12 to 15 ounces of silver to the ton Owing to the difficulties attendant upon an effective separation of these metals, it was not until 1909 that a market for these mixed ores was established.

The smelting companies engaged in the business of manufacturing oxide found this ore to be an especially desirable raw product for making a high-grade zinc-lead pigment, and since 1909 a large quantity of such ores have been shipped at a fair margin of profit to the miner.

Goodsprings operators now find a ready market for all kinds of ore which they produce. In contrast with expensive methods of milling low-grade zinc ores, the ores of this district require merely breaking down, hoisting, hand sorting, and shipment.

Ore occurs practically from the surface to the greatest depth attained by any of the mines (600 feet), and will very probably continue downward to water level and beyond. Not a drop of water has been encountered in any of the mines, and the depth of ground water level has never yet been determined.

The country rock is a Carboniferous limestone and is traversed in every direction by porphyry dikes. The ore deposits are the result of replacement and oxidation in limestone, and are in all cases closely associated with the porphyry.

The Yellow Pine, the Potosi, Monte Cristo, Prairie Flower, Hoodoo, and a number of other mining properties, have been producing these zinc and zinc-lead-silver ores, and are now engaged in the development of an increased tonnage.

Closely associated with the zinc deposits, but occurring practically free from zinc ore, are found deposits of galena and cerrusite ($PbCO_3$), averaging about 60 per cent. metallic lead and carrying silver.

In addition to the production of oxidized zinc ores, the mixed zinc-lead-silver ores, and lead-silver ores, the Yellow Pine district has produced copper and gold ores.

Copper ores carrying some gold and silver are found generally throughout the district, and about 2,000 tons of the higher grade ore has been shipped to the smelters. Copper occurs principally as a carbonate, but when found in porphyry dikes, is in the form of a sulphide.

Previous to the building of the San Pedro, Los Angeles & Salt Lake Railroad, the production of gold from the Keystone and two or three other mines, was the chief resource of the district. The metal is found in irregular masses of iron oxide or mineralized porphyry and requires a close study of the ore formations, due to the extensive faulting to which the region has been subjected, in order to avoid losing the ore chutes. As an example, the Keystone Mine has been repeatedly closed down, the miners thinking the ore was exhausted. Intelligent cross-cutting by some more venturesome operator has disclosed a large body of ore. This mine was discovered 17 years ago, has produced over $1,000,000, and today has sufficient low-grade ore developed to keep a large mill running for a number of years.

The Spring Mountain range, about 25 miles in length and 8 miles in width, is included in the Yellow Pine area, and it is in this range that nearly all the mines are located. Practically every square mile of the entire area has mineral croppings of some kind.

The discovery of zinc ores has naturally overshadowed all other resources. Recently strong financial interests have entered the district, taking over 14 of the better properties and starting construction work on a standard-gauge railroad from Jean, the shipping point on the Salt Lake Railroad, to the Yellow Pine Mine, a distance of 12 miles. Switches will undoubtedly be run as needed from this new spur to everything on the east side of Spring Mountain range. In order to eliminate the heavy cost of freight to Kansas smelters and Wisconsin oxide plants, these same interests are putting in an oxide plant at San Pedro.

Fig. 1 shows the derrick and hopper at Yellow Pine Mine with sorting table near the foot of the hopper. The ore is trammed from the sorting table to loading bins shown at the right, and from there is loaded into wagons to be hauled to the railroad at Jean.

The acquisition of 14 properties practically guarantees to the new company a plentiful supply of ore; the construction of a standard-gauge railroad will almost eliminate the cost of hauling by wagon, effecting at once a saving of nearly $3 per ton of ore; and the erection of their own oxide plant at San Pedro effects a saving in freight charges of about $6 per ton. In addition to this the profit derived from the manufacture and sale of oxide must be considered.

In the event that this pigment finds a ready market in the western states, the saving effected by these improvements will certainly exceed $10 per ton. Should it become necessary to send the oxide to the central or eastern states to find a market, the cost of the freight on the finished product must be deducted from this saving.

Such conditions cannot fail to directly benefit every miner and to result in a markedly increased activity throughout the Yellow Pine area.

LAKE SUPERIOR COPPER ORE

The Lake Superior copper producers are spending millions of dollars in efforts to reduce the cost of manufacturing copper and they have been successful to the extent that all of the money spent toward this end, and probably much more, has been returned during the last three years.

Costly regrinding machinery, new and the latest improved slime tables, tubular mills and other machinery for the big stamp mills have been purchased. Large buildings constructed entirely of steel, and covering acres of ground, have been erected at Lake Linden, by the Calumet & Hecla Co. in an effort to cut down the cost of manufacturing copper.

The Copper Range Consolidated, Wolverine, Mohawk, Quincy, Franklin, Hancock. Osceola, and other companies are following closely in the footsteps of the Calumet & Hecla in this regard. The Wolverine probably manufactures copper at less cost per pound than any other company in the district, and yet this company is making efforts to further reduce costs. The Calumet & Hecla is installing at some shafts more powerful and improved hoisting apparatus, rock crushers, and other machinery. The Hancock is erecting steel head-frames and rock houses, as the steel buildings will last indefinitely. The Quincy is replacing wooden with steel structures. High-salaried mining engineers, metallurgists, and millmen, are employed to figure out places where producing expense may be cut down.

The Calumet & Hecla Co. is striving to so reduce costs that 10 pounds of copper will pay for the milling, mining, etc., of the rock and the marketing of the copper from 1 ton. The company's rock probably average 20 to 23 pounds of copper to the ton. Figuring that the average is 20 pounds to the ton, the company will make as much from its tonnage, when it reaches its goal in the reduction of costs, as it expends on this tonnage. The Calumet & Hecla has made more profits for its stockholders than any other copper mining company in the world.

It is feared that No. 7 shaft of the Quincy Mine, which has sent out into the industrial world the finest copper ever mined, and probably more of it than any other one shaft, is nearing the limit of its depth. As the shaft goes down the runs of stoping ground are rapidly growing shorter. This does not mean, however, that the ground developed will play out in the near future, for the higher runs of stoping ground will furnish an enormous tonnage for many years to come. Quincy is earning more than $1.25 per share per quarter, or $5 per year, and has been paying dividends longer than any mine in this district, about a half century.

There are no leaks in the Quincy system, the power plants, mills, and underground workings being kept in the best of condition. The company's underground motive power is electricity, the trains in the drifts at every level practically being hauled by trolley locomotives. It is probable that electricity will be substituted for steam, within the next two or three years, to haul the rock to the mills.

The Isle Royale, which was a losing proposition for many years until the last year, and which is now under Calumet & Hecla management, is beginning to make a fine record. The property is being developed at a rate that will turn out 9,000,000 pounds of copper per annum and it is only beginning to produce.

The Algoma lode took so sharp a change in dip that the shaft ran over the ore at approximately 100 feet. At 104 feet sinking was stopped and a cross-cut driven in the foot-wall with the result that within 10 feet ore was found and drifting commenced at this depth although the intention was to commence at the 200-foot level.

Calumet & Hecla, the largest planet in the copper firmament, scintillates with increased brilliancy as the decades of its existence are extended. This inspiration comes not from Kentucky, nor yet from lake moonbeams; on the contrary from Beanville where tightwads are paying $600 per ticket.

The Tamarack is said to be recovering from 2½ to 3 pounds copper more than formerly from each ton of ore stamped. Assuming this to be true and that as much ore was stamped in 1910 as in 1909 the additional yield for 1910 would be 1,722,747 pounds copper.

After a long time prospecting at the Seneca Mine, the Kearsage formation was found at the 920-foot level, and in the drifting so far as advanced the mineralization is satisfactory.

The advantages to be derived from persistent and intelligent prospecting are nowhere so apparent as in the Lake Superior copper region. As an illustration, take the Ojibway, which a few months ago offered little encouragement, but which for several weeks past has been drifting in ore on the 500-, 650- and 800-foot levels. The La Salle Co. has a large acreage, which is traversed by all the paying formations in the district yet its Tecumseh shaft is producing only a lean ore carrying .6 per cent. of copper per ton. Judging from the history of the field the La Salle problem is one of sufficient money for continued and intelligent prospecting and the company has probably the former for the latter.

The Michigan company has a territory covering 5,000 acres, traversed by most if not all the lodes in the Lake Superior copper district. Drifting at the adit level exposed bunches of copper and drifting on a level 180 feet below gave better indications. The rock contains wire and shot copper. Diamond drill sections are being made for a distance of 9,000 feet to obtain a complete cross-section of the land from the Butler lode to the eastern sandstone. From a geological standpoint these should be extremely interesting. In the drilling so far conducted, the lode of greatest promise is the Ogima, from which three cores have been taken, all of which carry copper. The lode is a chocolate-colored amygdaloid with quartz and analcite gangue from 12 to 15 feet wide at all three places, with copper uniformly distributed.

DE WILDE PRECIPITATION PROCESS

*By G. Witteveen**

In 1895 P. de Wilde, Honorary Professor of the University of Brussels, published a process for the extraction of gold from the solutions obtained in cyanide practice. Shortly, this process consists in acidulating the solutions by means of sulphuric acid with the addition of cuprous chloride (Cu_2Cl_2). The yellowish precipitate which is immediately formed, consists of an intimate mixture of aurous cyanide ($AuCN$) and cuprous cyanide ($(Cu_2CN)_2$). Professor de Wilde patented his process in the Transvaal, and about 10 years later, in February, 1908, extended his claim to the extraction of silver. The writer made some experiments as to the practical utility of this latter method. According to the Belgian patent the process is carried out in the following manner:

Precipitating Gold and Silver From Cyanide Solutions Either Alone or Both Together

First Operation.—The solutions contain the gold and silver, as soluble salts of alkaline cyanides, moreover an excess of alkaline cyanide, and often caustic soda, carbonates, lime, etc. A strong acid, such as sulphuric acid, diluted to about 20 per cent., containing in solution an alkaline chloride such as chloride of sodium, is added while agitating until blue litmus paper shows an acid reaction. The salt may be dissolved first, and the acid added afterwards.

The acid neutralizes the alkaline matter, as well as the free and combined cyanides and liberates the hydrocyanic acid which remains in solution, while the cyanides of gold and silver are set free. The sulphuric acid also decomposes the metallic chloride, liberating hydrochloric acid, which acts on the cyanide of silver and transforms this into insoluble chloride of silver and hydrocyanic acid. The amount of chloride required is calculated from the amount of silver cyanide to be decomposed, and to make sure a small excess of salt is used. The precipitate is filter pressed and gives a pure bullion.

Second Operation, Precipitation of the Gold.—The liquid coming from the filter press contains free cyanhydric acid and gold cyanide. The gold is now precipitated by adding a copper salt, by preference cuprous chloride (Cu_2Cl_2) dissolved in the solution of chloride of sodium. Only a very small quantity of cuprous chloride is necessary to obtain a complete precipitation of the gold. The cuprous chloride forms with the cyanhydric acid copper cyanide ($(Cu_2CN)_2$), which combines by molecular attraction with the gold cyanide. This precipitate settles easily after some agitation, and can be separated after decantation or filtration. To have a complete precipitation of the gold, there must be about seven times as much copper cyanide as there is gold cyanide. Treating the gold-copper precipitate with chlorhydric acid, or a mixture of diluted sulphuric acid and chloride of sodium, the copper cyanide is dissolved without being decomposed and the gold cyanide is left behind. This is collected and smelted. The chlorhydric solution of the copper cyanide can be used again.

Third Operation, Regeneration of the Cyanide.—After the separation of the gold-copper precipitate the liquids contain all the cyanide as free cyanhydric acid, less the small quantity combined in the gold-copper precipitate, and the very small quantity carried away by the volatilized cyanhydric acid. It is then only necessary to neutralize the solution by an alkali to regenerate the cyanide.

First and Second Operation Combined, Simultaneous Precipitation of the Gold and Silver.—The first two operations may be executed at the same time. After precipitation of the silver and without removing this, the gold can be precipitated by the solution of cuprous chloride in chloride of sodium, or by the chlorhydric solution of the copper cyanide. Thus, the copper cyanide can be used over and over again.

* Paper read before the Institute of Mines and Metallurgy, Mexico.

In the treatment of the gold-silver cyanide solutions, Professor de Wilde claims as his invention: First, the complete precipitation of the silver as a chloride by the action of chlorhydric acid, or by a mixture of sulphuric acid or some other strong acid, with a metallic chloride (first operation). Second, the complete precipitation of gold cyanide by a copper salt and its fixation by molecular attraction on small quantities of copper cyanide, which nevertheless are sufficient to get a precipitate formed by a mixture of the two cyanides (second operation). Third, the combination of the two operations, so as to get in the same liquid a mixed precipitate, containing all the values as a cyanide of gold and copper and as a chloride of silver. Fourth, the repeated and indefinite utilization of the copper cyanide. This is about the process as it is described in the Belgian patent. The reactions upon which the process is based are doubtless correct. Taking the case of silver, this is present as the potassium silver cyanide salt $[KAg(CN)]_2$. By adding sulphuric acid, the silver cyanhydric acid is set free, thus, $2KAg(CN)_2+H_2SO_4=2HAg(CN)_2+K_2SO_4$. The silver cyanhydric acid, however, is such an unstable compound that it immediately is decomposed to cyanhydric acid and silver cyanide $HAg(CN)_2=HCN+AgCN$. In the same way the insoluble aurous cyanide is set free, but its quantity in the liquid is so small that it has to be carried down by another insoluble compound. That is the reason that for the precipitation of the gold a copper salt is added. Since there is a great molecular attraction between the aurous cyanide and copper cyanide, copper salts answer very well for this purpose. The weak point in the process, especially for silver ores, where generally strong solutions are used, is the escape of a considerable quantity of cyanide as cyanhydric acid after acidifying the solutions.

In the description of his process, Professor de Wilde says in the third operation: "The solution contains all the cyanogen, less the small amount combined in the gold-copper precipitate and the very small quantity volatilized as cyanhydric acid." This is true for very weak solutions, such as are generally used in cyaniding gold ores, but with strong solutions it is not the case, since cyanhydric acid is only slightly soluble in water.

The following tests will show this:

250 cubic centimeters of cyanide solutions of .05 per cent., .1 per cent., .15 per cent., 4 per cent. KCN, were acidified with diluted sulphuric acid; .4 cubic centimeter being sufficient for every .05 per cent. KCN to give an acid reaction on blue litmus paper.

After stirring, the solutions were left for 2 hours, and then a 6-per-cent. caustic soda solution was added to make them alkaline. For every .4 cubic centimeter of sulphuric acid, 2.5 cubic centimeters of caustic soda was sufficient to give a plain alkaline reaction on red litmus paper. The solutions were titrated again, and the results tabulated after correction for dilution. The following table is the average of two sets of tests:

KCN Per Cent. Before Treatment	KCN Per Cent. After Treatment	KCN Per Cent. Loss
.05	.0450	.0050
.10	.0900	.0100
.15	.1375	.0125
.20	.1800	.0200
.25	.2250	.0250
.30	.2650	.0350
.35	.3175	.0325
.40	.3625	.0375

It will be seen that, per ton of solution treated in the case of a solution of .05 per cent. KCN, the loss in cyanide is 50 grams per ton, and in the case of a solution of .4 per cent. KCN the loss amounts to 375 grams per ton. So the loss in cyanide, which is not regenerated by the alkaline matter, increases almost at the same rate as the strength of the original solution.

Much better results were obtained in the next tests, where the alkali was added immediately after the solutions were made

acid. The same quantities of solution, acid, and alkali were used as in the foregoing tests, but now the loss in cyanide was practically nil with the solution of .05 per cent. and .1 per cent. *KCN*, and amounted to only 100 grams with solutions of .3 and .4 per cent. *KCN*.

These tests show that the amount of cyanide escaped depends largely on the time the acid solution is left unprotected by alkali.

In practice, where the solutions have to be stirred thoroughly to coagulate the precipitate, where they afterwards have to be decanted or filtered, there is any amount of opportunity for the cyandhyric acid to evaporate, since evaporation depends as well on the surface exposed to the air. Both time and surface will exceed in practice the conditions of my tests.

Now I decided to make an experiment with a silver-bearing solution, and for that purpose imitated the solution of the San Rafael mill in Pachuca, as it enters the zinc boxes. The data were taken from an article by Mr. E. Girault, published in the December number of the *Informes y Memorias* of the Institute of Mines and Metallurgy, of Mexico.

This solution has the strength of .25 per cent. *KCN*, contains 275 grams of silver per ton solution, and .1 per cent. of lime.

Two liters of solution were made up with: 1,730 cubic centimeters lime water; 66.3 cubic centimeters standard $AgNO_3$ solution (13.05 grams per liter); 203.7 cubic centimeters water. A total of 2,000 cubic centimeters.

This gave a solution containing 275 milligrams silver per liter and 1 gram of lime.

Now, 5 grams of pure potassium cyanide were added and the solution titrated, which showed .22 per cent. The loss of .03 per cent. being the amount of cyanide combined with the silver, the theoretical loss would be .003 per cent. more.

Now two bottles were taken, each with half a liter of the solution. To each, 150 milligrams of cyanide were added to make the solution up to .25 per cent. and 100 grams of chloride of sodium. Then 5 cubic centimeters of 20-per-cent. sulphuric acid were run in, the contents of the bottles stirred, after which the precipitate was easily filtered. Two hundred and fifty cubic centimeters of the filtrate were taken and 25 cubic centimeters of a 6-per-cent. caustic soda added. The liquids were titrated and showed in both tests .23 per cent. *KCN*, or with allowance for dilution, .255 per cent. *KCN*.

If there had been no escape of cyanhydric acid the solution would have titrated .28 per cent., so the amount of cyanide not recovered is 250 grams to the ton of solution treated.

Although the liquid was acid after adding the 5 cubic centimeters sulphuric acid, all silver had not precipitated. The filtrate treated with some more sulphuric acid still gave a precipitate of $AgCl$. This shows that an excess of acid is necessary to obtain a complete precipitation of the silver. The tests were therefore repeated with two quantities of 250 cubic centimeters of the original solution. After making up the solution to .25 per cent. and adding 50 milligrams of salt, it was found that 3 cubic centimeters gave a permanent precipitate, but left some silver in solution, and that 4 cubic centimeters caused a perfect precipitation. To compare lime and caustic soda as a regenerator, one bottle was filtered through lime and the other one through clean paper, and afterwards made alkaline by caustic soda. The one filtered through lime showed .245 per cent., and the other one, after making the correction for dilution, exactly the same. The solution passed through lime titrated .122 per cent. of oxide of calcium, thus being practically saturated with lime and containing only a little more than the original solution.

In this test the amount of cyanide not recovered was 350 grams to the ton of solution treated, or 100 grams more than in the foregoing tests, this probably due to he fact that in the last test a small excess of acid was used, which is necessary to get a complete precipitation.

Where such an inconstant factor comes into play, as the volatilization of the cyandhyric acid, it would be difficult to base upon these experiments executed on a very small scale anything like a correct estimation of the cost of the process on a working scale. In the *Mining and Scientific Press* of December 18, 1909, R. P. Wheelock publishes the results of some experiments made on almost the same subject; viz., the regeneration of cyanide in copper-bearing cyanide solutions. Mr. Wheelock, who also made his experiments on a larger scale, found that the discrepancy in that case was more than in his laboratory tests.

In my experiments the original strength of the solutions was .25 per cent., and after precipitation of the silver and making the solutions alkaline again the strength was in the first tests .005 per cent. more, in the last tests .005 per cent. less. We thus may assume that practically the strength of the solutions has remained the same, the amount of cyanide volatilized after acidifying being the same as the amount combined with the silver in solution. So the expenses of cyanide are not lowered.

Now for 250 cubic centimeters solution in the second test 4 cubic centimeters of a 20-per-cent. sulphuric acid was consumed. This means 16 liters diluted acid per ton of solution, or 5.75 kilograms strong acid (66° B.). Putting the price of sulphuric acid at 9 centavos per kilo, this means 52 cents (Mexican) per ton of solution. In the San Rafael mill the amount of solution sent to the zinc boxes is about twice the amount of the ore treated. So the precipitation costs would be on consumption of sulphuric acid alone $1.04 (Mexican) per ton on ore, not counting the lime necessary to neutralize the solution and the salt to precipitate the silver. This is an excessively high cost, the precipitation and melting in the San Rafael mill under present conditions costing only 44 cents (Mexican).

Of course the great consumption of sulphuric acid is largely due to the excess of lime the San Rafael mill is using. According to the formula,

$$CaO + H_2SO_4 = CaSO_4 + H_2O.$$

One kilogram of CaO consumes about 1.75 kilograms pure sulphuric acid, or about 2 kilograms of 66° B., meaning 4 kilograms per ton of ore treated, or 36 cents (Mexican).

PERSONALS

Walter Brinton, Superintendent of the Manganese Steel Department of the Taylor Iron and Steel Co.'s plant at High Bridge, N. J., since 1895, has resigned, and has accepted a position as consulting engineer for the Edgar Allen American Manganese Steel Co., who are manufacturing manganese steel at Chicago Heights, Ill., and at New Castle, Del. Mr. Brinton's headquarters will be in the New Castle plant.

Charles Watson will go to Porcupine, Ontario, to be manager of the Armstrong McGibbon properties in North Tisdale. Mr. Watson has been superintendent of Nipissing; manager of the Chambers-Ferland, and of the Cobalt Townsite mines.

A. C. Bailey, manager of the Casey Cobalt, will assume management of the Cobalt Townsite Mine.

John Shaw, who has been connected with the Trethewey, King Edward, and Cochrane Cobalt properties, has been appointed manager of the Cobalt Casey.

Thomas English, of Streator, Ill., has been appointed superintendent of the La Salle mine rescue station in Illinois. This is one of the three stations in Illinois mentioned in the November issue of MINES AND MINERALS.

Allerton L. Cushman, Henry A. Gardner and Dr. N. Monroe Hopkins, specialists in their respective lines, have combined in order to open an Institute of Industrial Research at Washington, D. C. The combination is a strong one, and there is no reason why success should not go with them.

GRADING ANALYSES

*By H. Stadler**

The universal tendency of modern metallurgical practice toward fine grinding made the desirability of introducing and adopting a uniform system of series of standard screens for comparative purposes strongly felt, and after laborious and careful investigation a standard table of laboratory screens has been established and adopted by the Institution of Mining and Metallurgy.

Discussion Relative to Mines Trials Committee Standard Grades and Screens

Although fully appreciating the merits of the work achieved and the considerations and principles which have guided the Institution of Mining and Metallurgy in establishing and adopting the present table of standard laboratory screens, I have found it necessary to establish a slightly different ratio, and to extend the classification, in order to more exactly secure a grading which is equivalent to a successive reduction by one half of the volume (or weight) of the particles, or, if only the alternate grades are taken, of a reduction by 1 : 4. As, however, the sizes of this grading scale are in agreement with, or very close to, certain of the Institution of Mining and Metallurgy standards, I have shown in the attached Table A the nearest corresponding standards of laboratory mesh.

The following notes will show how the varying proportions of certain standard sizes of particles may be valued in units of energy (E. U.), which will allow of an exact expression of the efficiency of the crushing operation for purposes of useful comparison.

GRADING ANALYSES

1. *Mines Trials Committee* (*M. T. C.*) *Grades.*—The most rational and logical system of classification into grades, is undoubtedly to base it on the reduction of the volume (or weight) of the particles, especially for mining purposes, as the assay values are given with reference to this unit.

2. By reducing the cube of the unit successively by one-half of its volume and assuming these fractures to be again of cubical shape, each size of this series of theoretical cubes obtained represents a grade of a reduction scale of the ratio of 2.

3. If to each of these grades an ordinal number be given, beginning with 0 as representing the unit of 1-inch cube, the formulas in Table B determine the values of the data for the series of theoretical cubes.

4. By taking the sides of this series of cubes, as clear mesh apertures of a set of screens, it is admissable to assume that the functions of the irregularly-shaped average particles, determined by two consecutive screens, vary in the same ratio as the grades of the theoretical cubes.

For the sake of completeness the relations existing between the factors determining wire-woven screens are given in Table C.

5. The absolute values of the true average sizes are smaller than the theoretical cubes, in a proportion which varies with the physical nature of the ore, method of crushing, shape of mesh hole, etc. As far as relative values only are concerned we have not, however, to trouble about their exact relative place in the scale, and we may take the cubes of the respective grade as representatives.

6. Over nine grades, from 1-inch mesh down to 30 mesh (.0166-inch aperture), the true average weight of the particles of each grade has been ascertained, and the ratio of decrease in weight of these particles proved to correspond remarkably well with the correct reduction scale of the theoretical cubes, especially in consideration of the fact that the measurements had to be executed with the product coming from an ore breaker, containing consequently many splinters and flat-shaped pieces, not amenable to perfect classifying by size. The relation of the true average size of the particles to the corresponding theoretical cubes representing the mean grade between the grades of the screens, varied in the nine tests between 58.3 per cent. and 66.9 per cent., and the average of 61.4 per cent. can therefore be considered as fairly representative for the Rand ore and for a product coming from ore breakers.

7. With this factor, the values (as E. U., etc.) of the larger pieces remaining on the 1-inch screen, which would offer difficulties to classifying by screening, can easily be determined, by using the formulas of Table B, from the average weight of pieces of approximately equal size, of divers groups, into which the material to be examined is first assorted by hand.

8. The average value of screen products determined by two consecutive—possibly far distant—screens, are those of the grade having a number which is the arithmetical mean of the ordinal or E. U. numbers of the two screens, and therefore corresponding to those obtained by the application of E. J. Laschinger's formulas ($_9$).

9. The product passing the finest screen used for grading may by practical experience be assumed to be represented by the following values:

Average Product Passing	Ordinal or E. U. Number	Corresponding Mesh Aperture Inch
200 m. (.00246 inch) screen............	28	.00155
90 m. (.00620 inch) screen............	27	.00125

For the estimation of the average grade of the remainder on the coarsest screen, the battery screen used gives, in most cases, valuable information concerning the size of the coarsest particles, allowing to take the mean as above. The amount of the percentages may also be used as a guide in this direction, as a large percentage indicates evidently a coarser average grade than a small percentage, as in the case of final pulps, where the average grade is bound to be very near to that of the coarsest screen used.

10. The number of grades obtained by this scale at the crushing ratio 1 : 2, being too great for general use, each alternate grade only has been retained, and this restricted set determines therefore a reduction scale at the ratio 1 : 4.

11. The present scheme, as will be noted, standardizes not the screens, as such, but establishes an unerring standard for the sizing and classifying of the screen products into grades. It is consequently valid for all screens of any description, and completely independent of purely practical and commercial considerations, which have to guide mill men and manufacturers for meeting the varying requirements in practice.

12. The standard grades apply to screens in practical use as well as to laboratory screens, and the only difference to be made is, that for the latter, as instruments of measurement, a high degree of accuracy must be required. In this regard commendable efforts have been made by the Institute of Mining and Metallurgy, resulting in the creation of a specially made set of standard laboratory screens, which individually are most accurate with regard to mesh aperture and uniformity of discharge area, but, in part do not fill the requirements of the system of classification which I advocate, for reasons given below.

13. Summarizing the advantages of my system of standardization of grades, they are:

(*a*) No really new departure from standard practice.

(*b*) Equality of steps with regard to reduction in volume.

(*c*) A sufficient number of grades.

(*d*) Independence of wire gauge, and therefore, immediate applicability to screens of any description, with round or square holes.

(*e*) The standard grades are exactly the same for screens in practical use in metallurgical operations as for laboratory

* A paper submitted for joint discussion to the Institute of Mining and Metallurgy, read at the meeting of May 19, 1910, in London, and to the Chemical, Metallurgical and Mining Society of South Africa, read at the meeting of May 21, 1910, in Johannesburg.

purposes, whilst the screens may vary in accordance with the gauge of wire used, with degree of accuracy in their manufacture, etc.

(*f*) No limit to extension in either direction or by interpolation.

(*g*) The variations of all other functions of the particles, besides the diameter, can be expressed by equations, or plotted out in regular curves.

(*h*) Any odd screen of which the clear aperture is known can readily be inserted in the scale by the use of the formulas given in Table B.

(*i*) The ordinal numbers represent at the same time the relative value of the energy required to produce the respective grade from the unit.

(*j*) The average values of screen products determined by possibly far distant screens are readily obtained.

(*k*) Any projected standardization of battery screens can easily be made to fall in with the present scale.

(*l*) The so-called 60- and 90-mesh screens (aperture .01-inch and .006-inch, respectively), are already in general use on the Rand.

14. *Accuracy.*—The measurement of the size of particles by grading with screens can never claim to be scientifically exact, as too many factors of inaccuracy have to be reckoned with, as, for instance, the limits of accuracy in the manufacturing of the screens, the alterations in their apertures by wear, and the fact that the actual maximum size of the particles passing through any given screen varies with different ores, different grinding systems, and different methods of conducting the grading analyses, etc.

15. Even if it were possible to bring the measurement by screening to a high degree of perfection, it would be futile to pretend to excessive refinements, as the working conditions of milling plants, are in practice, subject to unavoidable fluctuations, and the samples subjected to grading are only more or less representative averages of mill products, varying within wide limits.

TABLE A. STANDARD GRADES AND SCREENS.

MINES TRIALS COMMITTEE—(M.T.C.) STANDARD GRADES.

Ordinal Number or Mech. Value.	Mesh Aperture (=sides of theoret. cubes.)		Volume (or weight) of theoret. cubes.		Area of Fracture from Unit.	Commercial Denomination of Screens. Approx. number of Meshes.		Ordin. Number or Mech. Value of Grade.
EU.	In.	mm	$\frac{1}{1000}$ cub. in.	cub. mm	sq. in.	p. lin. in	p. sq. in	EU.
30	·00098	02480	·000000	·00002	3072·0	↑		30·0
29	·00123	·03125	·000001	·00003	2438·1	−200	Grade incl.	29·0
28	·00155	·03937	·000003	·00006	1935·4	Wa	shed Slimes	28·0
27	·00195	·04961	·000007	·00012	1536·0			27·0
26	·00246	·00250	·000015	·00024	1219·1	200	40000	26·0
25	·00310	·07875	·000030	·00049	967·68	150	22500	25·0
24	·00391	·09922	·000059	·00098	768·0	130	17000	24·0
23	·00492	·1250	·000119	·00195	609·52	110	12000	23·0
22	·00620	·15750	·000238	·00391	483·84	90	8000	22·0
21	·00781	·19844	·000477	·00781	384·0	75	5500	21·0
20	·00984	·250	·000954	·01562	304·76	60	3500	20·0
19	·01240	·3150	·001907	·03125	241·92	50	2500	19·0
18	·01562	·39688	·003815	·06250	192·0	40	2000	18·0
17	·01969	·50	·007620	·1250	152·38	35	1200	17·0
16	·0248	·630	·015259	·250	120·96	30	900	16·0
15	·03125	·07938	·030517	·50	96·0	25	600	15·0
14	·03937	1·0	·061035	1·0	76·19	20	400	14·0
13	·04961	1·260	·122070	2·0001	60·480	18	300	13·0
12	·06250	1·5875	·244140	4·0002	48·0	14	200	12·0
11	·07875	2·0	·488281	8·0003	38·102	11	120	11·0
10	·09922	2·520	·976562	16·000	30·240	8	65	10·0
9	·1250	3·1750	1·9530	32·001	24·0	6	35	9·0
8	·15750	4·0003	3·9060	64·003	19·051	4	15	8·0
7	·19850	5·040	7·8120	128·02	15·120	3·5	12	7·0
6	·250	6·350	15·625	256·03	12·0	Ap'rt're ¼″	9	6·0
5	·3150	8·0005	31·250	512·06	9·526	5/16″	6	5·0
4	·39690	10·080	62·50	1024·1	7·560	⅜″	4	4·0
3	·50	12·70	125·0	2048·3	6·0	½″	3	3·0
2	·630	16·0	250·0	4096·5	4·763	⅝″	2	2·0
1	·79370	20·160	500·0	8193·0	3·780	¾″	1	1·0
0	1·00	25·4	1000·0	16386·0	3·0	1 inch	—	0·0
Arithmetical Progression R=1	$R=\frac{1}{\sqrt[3]{2}}$ = ·7937		R = ½ = 0·5		$R=\sqrt[3]{2}$ = 1·2599			
	Geometrical Progression.							

Institution of Mining & Metallurgy. Standard Laboratory Screens.

Ordin. Number or Mech. Value of Grade.	No. of Mesh.	Mesh Aperture.	Diam. of Wire.	Area of Discharge.
EU.	lin. in.	in.	in.	%
26·02	200	·0025	·0025	25·0
24·81	150	·0033	·0033	24·50
23·77	120	·0042	·0041	25·40
23·01	100	·005	·005	25.0
22·60	90	·0055	·0055	24·50
22·08	80	·0062	·0063	24·60
21·49	70	·0071	·0071	24·70
20·81	60	·0083	·0083	24·80
20·0	50	·01	·01	25·0
19·03	40	·0125	·0125	25·0
17·80	30	·0166	·0167	24·8
16·99	25	·020	·020	25·0
16·02	20	·025	·025	25·0
15·06	16	·0312	·0312	24·92
13·81	12	·0416	·0417	24·92
13·01	10	·05	·05	25·0
12·08	8	·062	·063	24·60
10·0	5	·10	·10	25·0

The M.T.C. grades are based on the assumption of a successive reduction in volume (or weight) of the screen products by one-half from grade to grade.

16. For scientific and accurate work the full set of Mines Trials Committee grades should be used but for the daily control of operations, on the mines, the 60 and 90 Mines

Trials Committee grade (50- and 80-mesh Institute of Mining and Metallurgy screen) are quite sufficient.

17. Considering the above-mentioned factors of inaccuracy, it is unnecessary to work closer than to the first place of decimals in the percentage ratios.

18. *Methods of Conducting Grading Analyses.*—Although the wet method may give more accurate results, the dry method is for practical reasons almost exclusively in use on the Rand. If a sufficient number of screens be taken, a displacement of washed product freed from the finest grade is dried in the same screen frame, and the grading is then continued and finished on the redried sand.

20. The following advantages are claimed for this mixed method:

(*a*) Greater accuracy.

(*b*) Saving in time in spite of the double drying, as with washed sands, in a comparatively short time a clearly-defined end of the operation of screening is reached.

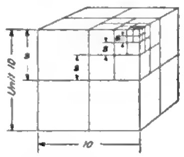


TABLE B. RELATIONS BETWEEN VALUES OF THE DATA FOR SERIES OF THEORETICAL CUBES OBTAINED BY SUCCESSIVE REDUCTION OF CUBE OF UNIT

	Number of Pieces P	Sides of Cubes S	Volume of Cubes V	Area of Fracture F	Ordinal or Energy Number N
Number of pieces produced from unit P=		$\frac{1}{S^3}$	$\frac{1}{V}$	$\frac{F^3}{27}$	2^N
V. & S. ratio values in regard to unit: Sides of cubes (mesh apertures) S=	$\frac{1}{\sqrt[3]{P}}$		$\sqrt[3]{V}$	$\frac{3}{F}$	$\frac{1}{\sqrt[3]{2^N}}$
V. & S. ratio values in regard to unit: Volume of cubes (or weight) V=	$\frac{1}{P}$	S^3		$\frac{27}{F^3}$	$\frac{1}{2^N}$
Area of fracture with reference to unit F=	$3\sqrt[3]{P}$	$\frac{3}{S}$	$\frac{3}{\sqrt[3]{V}}$		$3\sqrt[3]{2^N}$
Ordinal or energy number of grades N=	$\frac{\log P}{\log 2}$	$\frac{\log \frac{1}{S^3}}{\log 2}$ $= -10 \log S$	$\frac{\log \frac{1}{V}}{\log 2}$	$\frac{\log F - \log 27}{\log 2}$ $= \frac{N}{10} + .477$	

N. B.—The surfaces exposed are double of the area of fracture (F).

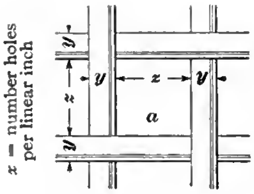


TABLE C. WIRE SCREENING. RELATIONS BETWEEN DIAMETER OF APERTURE, DIAMETER OF WIRE, SCREEN AREA AND NUMBER OF HOLES PER LINEAR INCH IN ANY SCREEN HAVING THE SAME NUMBER OF HOLES EACH WAY AND MANUFACTURED WITH ONE GAUGE OF WIRE ONLY

	Disch. Area a Diam. Wire y	Disch. Area a Apertures z	Disch. Area a No. of Holes x	Diam. Wire y Apertures z	Diam. Wire y No. of Holes x	Apertures z No. of Holes x
Per cent. of discharge of area: a=				$\left(\frac{10z}{y+z}\right)^2$	$100\,(1-xy)^2$	$100\,z^2x^2$
Diameter of wire: decimals of an inch y=		$\frac{10z}{\sqrt{a}} - z$	$\frac{10-\sqrt{a}}{10x}$			$\frac{1-xz}{x}$
Diameter of mesh aperture: in decimals of an inch z= .	$\frac{y\sqrt{a}}{10-\sqrt{a}}$		$\frac{\sqrt{a}}{10x}$		$\frac{1-xy}{x}$	
Number of holes per linear inch x=	$\frac{10-\sqrt{a}}{10y}$	$\frac{\sqrt{a}}{10z}$		$\frac{1}{z+y}$		

The diameter, in decimals of an inch, of a steel wire is approximately: $y = .00315\sqrt{w}$, where w = the weight in milligrams per linear inch of the wire.

even a few percentages from one grade to the next adjacent one, will not appreciably affect the result. More care must be taken with regard to the slime, which adheres to the coarser particles, and if the operation of screening be not prolonged over a very long time, a displacement of about 2 per cent. takes place from the very finest to the coarser particles. This inconvenience can, in practice, be overcome very satisfactorily by adopting the mixed method.

19. A convenient weight of the sample is dried and then the sands are washed in a suitable circular metal frame having the finest screen it is proposed to use fitted into its bottom. The

(*c*) No dusting.

21. For the drying of the sand, steam or water baths should be used, as too great a heat in drying the sand causes the bursting of about 2 per cent. of the coarse particles, the splinters going to increase the +120 grade especially.

22. The practical limits of manufacturing are reached with the 200 mesh (.0025-inch aperture) screen, and for further classification the metallurgist will have to resort to the elutriation method by which the particles are, in a current of water of varying velocity, classified on the basis of their hydraulic value. In consequence of the influence of the varying shape and specific

gravity of the particles, this method gives, with regard to the volume of particles, no clearly defined grades, but on the other hand, the gradings are more in accordance with practical working conditions The portion to be examined by elutriation must be separated from the wet sands before they are dried in order to avoid any alteration, due to heat, of the particular physical properties of the slimes.

APPLICATION OF GRADING ANALYSES

23. *Theory.*—The area of fracture over which the cohesion of the molecules has to be destroyed, multiplied by a coefficient determining the resistance which the molecules oppose to their separation by the exercise of any stress (crushing, tensile, shearing, etc.) represents the force required to cause the fracture.

24. In order to perform mechanical work this force has to run through a distance represented by the deformation which the body can stand before reaching the breaking point. It is in this connection immaterial that this distance of deformation within the limits of elasticity and plasticity is, for not perfectly homogeneous bodies, subject to variations, which for highly elastic and unplastic bodies as quartz, glass, etc., are too insignificant to be considered, and in addition they are by the nature of our crushing appliances averaged to such an extent that these averages are as good as exactly defined figures. Dealing with relative values only, we have not even to care about the exact extent of this deformation, and all that we need is to be satisfied that this factor is a constant function of the diameter of the particles to be crushed.

25. The mechanical work done is represented by the product of the force by the distance, but as in a regular scale of reduction by volume the diameters of the particles decrease at the same ratio as the area of fracture increases, the product, or the mechanical work required for reducing tbe volume (or weight) of the unit from one grade to the next following, is a constant for each grade, called the crushing or energy unit (E. U.)

26. The same conclusion is also arrived at by the application of Kick's Law, which reads:

"The energy required for producing analogous changes of configuration of geometrically similar bodies of equal technological state varies as the volumes or weights of these bodies."

The volumes of the particles decrease from grade to grade at the same ratio as the number of the particles, constituting in their total the volume of the unity, increases, and the product of the volumes into the number of the particles of that grade is, therefore, constant for each grade. As in conformity to the above law the amount of energy absorbed is proportional to the volume of the body to be crushed, it follows again also that the total energy required for reducing the weight of the unit is constant for each grade.

27. The ordinal numbers of any arithmetical progression given to these grades represent consequently the relative values of the energy which has to be spent upon producing this respective grade from the initial unit, or the mechanical value of the grade.

28. For obtaining the mechanical value of mixed sands we need only to multiply the percentages of the gradings by the number of the energy units of the respective grade and add the products. This possibility of having the grading of pulps condensed and expressed in one representative figure proves to be of great value.

29. The useful work done per unit by any crushing machine is determined by the difference between the mechanical values of the samples taken at the inlet and the discharge of the machine, and for obtaining the total work done this difference has to be multiplied by the tonnage dealt with.

30. The relative mechanical efficiency is the value obtained by dividing the total work done by the unit of energy (for instance, horsepower):

$$\text{Relative mechanical efficiency} = \frac{\text{Tonnage} \times \text{work done per unit in E. U.}}{\text{Unit of energy (H.-P.)}}$$

31. *Examples of Efficiency Calculations* (*Rand Ore*).

(*a*) *Stamp:*

Battery screen, 64 mesh, .097-inch aperture.

Running weight, 1,150 pounds (energy 1,174 pounds foot-seconds).

Power consumption, 2.6 horsepower.

Duty, 9 tons per 24 hours.

Mines Trials Committee Standard Grades			Feed	Discharge Pulp		
Denomination of Screen	Aperture	Mechanical Value of Mean Grade		Grading Analyses		Mechanical Value of Discharge
Number	Inch	E. U.		Per Cent.		E. U.
+ 12	.03937	13	Subject to variations, but by practical experience it has been ascertained that the 0 grade (1-inch screen) represents a fair average size.	19.9	+60	2.59
+ 20	.02480	15		12.1	53.2	1.82
+ 30	.01562	17		9.7	+90	1.65
+ 50	.00984	19		11.5	11.5	2.19
+ 80	.00620	21		10.9	−90	2.29
+120	.00391	23		2.6	35.9	.50
+200	.00246	25		4.2		1.05
−200 including washed slimes	.00246	28		29.1		8.15
				100%		20.34
				Less Mech. value of feed		0.0
				Work done by stamp per unit (say 1 ton)		20.34
Total work done per stamp in 24 bours = 9 tons × 20.34 E. U. =						183.06 E. U.

$$\text{Relative mechanical efficiency per H. P.} = \frac{9 \text{ tons} \times 20.34 \text{ E. U.}}{2.6 \text{ H. P.}} = 70.40$$

(*b*) *Tube Mill:*

Dimensions, 22 ft. × 5 ft. 6 in diameter.

Banket ore feed, 6.75 tons per 24 hours.

Speed, 30 revolutions per minute.

Power consumption, say, 100 horsepower.

Duty, 290 tons per 24 hours.

Mines Trial Committee Standard Grades			Intake			Discharge		
Denomination of Screen	Aperture	Mechanical Value of Mean Grade	Grading Analyses		Mechanical Value of Intake	Grading Analyses		Mechanical Value of Discharge
No.	Inch	E. U.	Per Cent.		E. U.	Per Cent.		E. U.
12	.03937	13	27.6	+60	3.59	–	+60	–
20	.02480	15	15.7	76.9	2.36	–	29.9	–
30	.01562	17	13.7		2.33	7.4		1.26
50	.00984	19	19.9	+90	3.78	22.5	99.0	4.28
80	.00620	21	15.5	15.5	3.26	29.0	29.0	6.09
120	.00391	23	2.0		.46	6.0		1.38
200	.00246	25	1.6	−90	.40	7.7	−90	1.93
200 including washed slimes	.00246	28	4.0	7.6	1.12	27.4	41.1	7.67
			100%		17.30	100%		22.61
			Less mechanical value of intake					17.30
			Work done per unit, say 1 ton					5.31

Total Work Done by Tube Mill in 24 Hours	*E. U.*
Reduction of pulp feed = 290 tons × 5.31 E. U. = ...	1,539.2
Reduction of banket ore = 6.75 tons × 30 E. U. = ...	202.5
Total work done in 24 hours..........................	1,742.4

$$\text{Relative mechanical efficiency per H. P.} = \frac{1,742.4 \text{ E. U.}}{100 \text{ H. P.}} = 17.42$$

32. Under the conditions of the above examples taken from practical work, the efficiency per horsepower of tube mills (17.42) is only about one-fourth that of the stamps (70.40).

33. For general guidance some instructive figures of mechanical values might be of interest:

Battery pulps varying with the increase of the fineness of the screening from 17 E. U. to 24.5 E U.

Tube mill intake depending upon more or less perfect removal of the finest particles by the classifiers two to three E. U. less.

Tube mill discharge, varying with working conditions, especially coarseness of feed, from 21 E. U. to 24.5 E. U.

Final (cyanide) pulp varying between the following extreme limits:

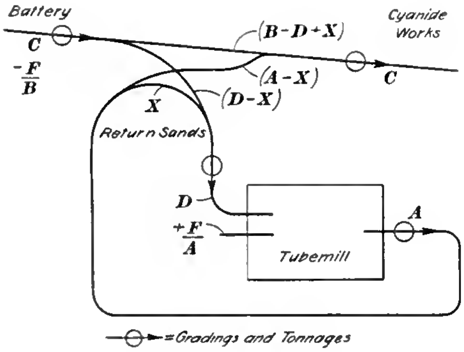
	+60	+90	−90
Coarse pulp................	15%	20%	65% = 24.45 E. U.
Fine pulp..................	3%	10%	87% = 26.13 E. U.

34. The difference between the mechanical value of the final (cyanide) pulp and the battery pulp represents the work done per unit (say, 1 ton), by the secondary crushing appliances, and their total work done, *i. e.*, the sum of the products:

Tonnage × (mechanical value discharge minus mechanical value inlet) of the single units must therefore be equal to battery tonnage × (mechanical value final pulp minus mechanical value battery pulp).

From this fact the correctness of the measurements can readily be checked, or inversely the efficiency of the secondary grinding machines can be controlled by the difference of the mechanical values of the final and the battery pulps, of which reliable average samples are taken in any case for assay purposes.

35. The great advantage connected with my method of expressing the values of the gradings in one representative figure is that it allows of the solution of many interesting problems, for instance, the redistribution of the battery and tube-mill discharge pulps, which in the spitzkasten become mixed up, can by calculation be exactly analyzed into the amounts delivered to the cyanide works or returned to the tube mill for regrinding.



X = tons per 24 hours tube-mill return sands;
A = tons per 24 hours total sand and banket feed;
B = tons per 24 hours total net battery duty;
C = tons per 24 hours total ore milled;
D = tons per 24 hours tube-mill sand feed only;
F = tons per 24 hours banket feed only;
c = difference mechanical values of cyanide and battery pulps;
e = difference mechanical values of tube-mill intake and cyanide pulp;
h = difference mechanical values of tube-mill intake and discharge pulps;
g = mechanical value represented by reduction of banket pieces to size of cyanide pulp;
i = mechanical value represented by reduction of banket pieces to size of tube-mill discharge pulp.

The mechanical work done in tube mills being determined (1) By difference of work done represented in intake and discharge pulps $= D\,h + F\,i$. (2) By difference of work done represented in the portion of the tube-mill discharge delivered to the cyanide works and the tube-mill inlet pulp $= (A - X)\dfrac{D\,e + F\,g}{A}$. (3) By difference of work done represented in cyanide and battery pulp $= B\,c + F\,g$, we obtain by equalizing the above values for instance for the amount of the return sands:

$$X = A - \frac{(D\,h + F\,i)\,A}{D\,e + F\,g}, \text{ or, } X = A - \frac{(B\,c + F\,g)\,A}{D\,e + F\,g}$$

By neglecting the banket ore feed the formula would become reduced to:

$$X = A - \frac{D\,h}{e}, \text{ or, } X = A - \frac{Cc}{e}$$

36. The energy absorbed in the reduction of the banket ore feed to the size of the final pulp can as an average be estimated at 30 E. U. (26 E. U. being the mechanical value of the final pulp with reference to the initial unit, plus 4 E. U. representing the energy absorbed for the reduction of 4-inch pieces to the size of the initial unit or 0 grade.) For 22-foot tube mills, as in general use on the Rand, the work done in reducing the banket ore represents, depending upon the working conditions, 5 to 25 per cent. of the total work done.

37. The mechanical values of the pulps may in many cases, where a great accuracy is not required, be calculated from the gradings of the three grades only, determined by the usual screens, Nos. 60 and 90. This simplified calculation compares for instance for above tube-mill example, as follows:

Grade	Intake			Discharge		
	Weights	Mechanical Value of Mean Grade	Mechanical Value of Intake	Weights	Mechanical Value of Mean Grade	Mechanical Value of Discharge
	Per Cent.	E. U.	E. U.	Per Cent.	E. U.	E. U.
+60	76.9	15.5	11.92	29.9	18	5.38
+90	15.5	21.0	3.26	29.0	21	6.09
−90	7.6	27.0	2.05	41.1	27	11.10
	100.0		17.23	100.0		22.57

CONCLUSION

38. All through the experimental work, executed on behalf of the M. T. C., it could be ascertained that this method of efficiency calculation is for fine as well as for coarse crushing, correct and reliable, and in perfect accordance with the results of practical experience. Based on these facts it can be stated that we are now in a position to determine, with a comparatively high degree of accuracy, the relative merits of different crushing appliances, or the mechanical efficiency of one and the same machine working under varying conditions.

39. In the author's reply to the discussion on his paper on "Heavy Gravitation Stamps," Doctor Caldecott comes forward with a method of his own for computing efficiency, in tacit opposition to mine, the merits or otherwise of which, as an active and interested member of the Mines Trials Committee, he has had under close observation for more than a year. The discussion on his paper being closed, I may be allowed, under these circumstances, to take the controversy up in the present paper. In order to stimulate Doctor Caldecott to redouble his efforts to discover flaws in the correctness and reliability of my methods I begin by saying squarely and in plain terms that his method is not only unscientific, but also inaccurate. All the figures obtained by, and the conclusions drawn from it, are incorrect, misleading, and therefore worthless.

Doctor Caldecott first finds it very convenient to introduce a new "nominal crushing unit." This he defines as a stamp of an arbitrary description as regards weight, drop, and number of drops per minute. With no reasons given, it is difficult to see the desirability or necessity of introducing a new unit for the measurement of the energy used by different stamps when such units as foot-pounds, seconds, or horsepower are already available and easily understood. This device is so much the more needless as the value in normal crushing units of any other stamp of different weight or working conditions is to be determined by the ratio of the foot-pound, second, or horsepower of the particular stamp to that established as the normal crushing unit. There is, therefore, no reason why the usual units should not be taken as a standard. It is then

further arbitrarily assumed that a tube mill of assumed dimensions, with an assumed power consumption, is equal to 30 such nominal crushing units. The admission that this ratio "varies somewhat according to the conditions of operating" is appropriate, provided that the limit of elasticity of the term "somewhat" be made wide enough to stand the enormous strain of the very great effect of these operating conditions on the efficiency of the tube mills of equal cubical capacity inside shell.

I do not lay too much stress on the above points, which are only "somewhat" inaccurate; but a very serious error is made by taking the production of the −90 grade as a standard of the work done in crushing. By the use of his theory, which Doctor Caldecott bases on his finding that "the more tons of this material that can be obtained per stamp, or per tube mill, or per horsepower, the better," undue credit is given to the "finishing" work performed by tube mills on an already very fine feed; whilst the hard work done by other crushing appliances in reducing the coarse particles to the size of this feed is completely ignored. By similar reasoning Doctor Caldecott might have said that the ultimate object sought is the reduction of the +60 grade, and in this case he would have come to just the opposite conclusion, namely, that practically all the work done in crushing is performed by the preliminary crushers and very little is left for the tube mill to do. The fact that the figures obtained by his method do not show any advantage in favor of coarse crushing will perhaps shake his confidence in his method more than any possible criticism. Indeed, according to his figures, the production of the −90 grade is, for the same stamp weights, about the same for any coarseness of battery screens, whilst by my method, which never fails to accord perfectly with practical experience, the greater efficiency claimed for coarse crushing stamps is clearly shown by the figures obtained. They are for a stamp of 1,250 pounds running weight, as follows (see table):

Mesh and Aperture of Battery Screen		Duty Per 24 Hours Tons	Mechanical Value of Pulp E. U.	Relative Mechanical Efficiency Per H. P.
Mesh	Inch			
9	.272	13.0	18.0	234.0
200	.053	7.5	21.5	161.3
600	.028	6.5	23.2	150.8
1,200	.017	5.2	24.3	126.4

40. In conclusion, I wish to record my appreciation of the courtesy of the Mines Trials Committee in granting permission to use data obtained in the course of my investigations whilst in their employ, but I am personally responsible for the mathematical reasoning and considerations derived therefrom, together with the opinions expressed.

Besides many minor articles and contributions scattered in technical publications—too numerous to be quoted—especially the following literatore has been consulted on the subject:

1. The relative chapters in well-known mining books.

2. I. M. M. London. Bulletin No. 38, 1907.

3. Report of Sub-committee on the Standardization of Battery Screening. Supplement to the Journal of the Ch. M. M., Johannesburg, June, 1906.

4. C. de Kalb, Engineering and Mining Journal, July, 1905.

5. E. T. Henderson, Australian Mining and Engineering Review, November, 1908.

6. W. McDermott, Standardization of screens. Crowther and Goodman, Printers, London.

7. W. A. Caldecott, Ins. Min. and Met., November, 1905.

8. Philip Argall, Mining and Scientific Press, 1908.

9. S. H. Pearce and W. A. Caldecott, Ins. Min. and Met., September, 1906, and contrbutions from H. A. White (October, 1906); Prof. R. H. Richards (February ,1907); E. J. Laschinger (Januuary, 1907), etc.

10. Arthur Yates, Ins. Min. and Met., December, 1908, and contributions from H. Stadler (April and August, 1909), H. A. White (May, 1909).

11. Harry S. Denny, Journal S. A. A. of E., and contributions by Messrs G. E. Busch, J. A. Vaughan, etc.

12. G. A. and H. S. Denny, Journal S. A. A. of E., June, 1908.

13. H. Stadler, Journal S. A. A. of E., December, 1908, and February, 1910. etc

LIMIT OF PROPORTIONALITY

Written for Mines and Minerals

When one body is acted on by another tending to change its dimensions it is said to be subjected to *stress*. The force producing the stress is considered as acting across a certain area and at right angles to a section of the body. When the stress has a tendency to increase the length of the body it is termed *tension*, and when it acts so as to decrease its length the stress is termed *pressure*. Tensile strength is the resistance of the particles of a body to separation. It is proportional to their number in a given transverse section or area of the body and in metals it varies with the temperature.

If a body is rigidly suspended, the load it will sustain without actual rupture or separation of its particles is its *tensile strength*.

Just before rupture occurs the load producing stress is termed the *ultimate load*, and is usually measured by machines constructed for this purpose; at the same time the area and elongation of the body are measured.

If a body be subjected to stress of a definite amount so as to alter its form, the deformation resulting is termed a *strain*. Strain is measured by the ratio of the change in dimension of a body to its unstrained value. *Elasticity* is the property by virtue of which a body subjected to strain returns to its original dimensions and form.

If the body is permanently deformed it has passed its elastic limit, but just before the alteration takes place the body is said to have reached its *limit of elasticity*. The strength of a body is the value of the stress when the body breaks or its particles give way altogether.

It is a well-known fact that rupture may be produced by numerous applications of a load below the elastic limit of the metal; and such an occurrence in the case of shafts and axles is vaguely ascribed to the "fatigue of metals" in service. This term conveys a larger scientific truth than is generally appreciated; and in some testing works it is customary to measure, more or less empirically, the endurance of metals with regard to what may be termed molecular fatigue. If steel were a homogeneous metal, every applied load, less than the elastic limit, should produce a deformation proportionate to the load; and upon the removal of the load the test piece should regain its original dimensions. If, for example, on a steel rod under tension, a load of 1,000 pounds produces a stretch of $\frac{1}{1000}$ of an inch, then 2,000 pounds should produce double this stretch; and 4,000 pounds should cause four times the deformation produced by 1,000. If by successive increments of 1,000-pound loads there is obtained regularly increasing stretch, proportioned to the load, up to 10,000 pounds, but a load of 11,000 pounds produces $\frac{12}{1000}$ of an inch stretch, it is evident that, at 10,000 pounds load, the ratio between strain and stretch changes. This point is called by the French and German investigators the "limit of proportionality." It is far below the elastic limit of the test piece. In the case outlined, the load might be increased to 20,000 pounds before a permanent set would be produced; but it is very evident that the elastic strength of a metal is more accurately measured by the limit of proportionality, where a molecular fatigue begins, than by the commonly known "elastic limit," where an actual molecular distortion takes place.

GEOLOGY OF KOLAR GOLD FIELD

Written for Mines and Minerals, by C. S. Durand, Mysore Province, India

A Description of the Rock Formations, The Mines and General Conditions in Mysore

Practically the whole of the province of Mysore consists of a plateau of about 3,000 feet above sea level. It is quite undulating on the whole, but with the exception of the granite or gneiss peaks scattered about, the country is not hilly. With the exception of the region mineralized with gold-bearing quartz, the whole country is of a foliated granitoid rock which the geologists insist upon calling gneiss. It is uniformly gray in color, varying in crystallization from coarse to fine, and from light to dark in tint. The most striking feature of the landscape is the gneiss hills, which are scattered about without any regularity whatever in their arrangement. In size these vary from a few boulders in elevated places to single haystack-like cones more than a thousand feet high, some forming miniature mountain ranges 10 miles long and 1,500 feet above the surrounding country at the highest points. Much of this gneiss disintegrates rather rapidly, leaving a lot of sandy, barren soil. This rock furnishes the building stone of this whole region, being readily obtained in pieces from 4 inches to 1 foot thick and of any length and width that can be handled, making an ideal stone for steps, copings, and the like.

The laterite drift is one of the most puzzling phenomena presented in all this region, its very obscurity adding interest to it.

The whole surface of the country, with the exception of the gneiss hills and "Goldfield Ridge," is covered with laterite. Indeed, so nearly universal is it that the theory of its covering the entire surface up to certain altitudes at the time it was laid down seems quite correct. How far to the north the laterite extends beyond the sea north of Madras the writer does not know, but between Kolar and Madras it covers many thousand square miles. Near Madras the laterite is used for buildings, railway stations, and bridge piers, but that about Kolar is quite unfit for building purposes. Where found in large masses it is quite amygdaloidal in structure, the cells in it being from the size of beans to that of small marbles. The laterite occurs massive on the top of certain hills, which are uniformly flat, and on one hill 15 miles from Kolar, it is about 20 feet thick; whereas, all over the country—scattered everywhere—is the fragmentary laterite, nearly decomposed, giving the peculiar iron color, dark red or maroon, to the soil to a depth of from 2 to 4 feet, the portions remaining being little shot-like balls, either round or kidney shaped.

The state geologists have not discovered the origin of laterite, nor a place where it is found to great depth; so the theory is that it was spread over the country under water at some very remote period, coming from "parts unknown."

The hornblende schist formation extends roughly for 30 miles north and south, and varies in width from 1 to 4 miles. The Kolar gold field is approximately at the south end of the formation. The rock has a greenish color and its ability to withstand the weather varies everywhere, with the result that ridges of the rock are seen protruding above the surface in some places, while in others it has been completely decomposed into a clayey soil running through all gradations and combinations of color from light green and light yellow to nearly white—and this to a depth of from 6 to 12 feet. It is common in digging to pass through the iron stain from the laterite at the surface for a depth of 2 or 3 feet, and then come to the unstained clay of the decomposed country rock.

In this schist, running nearly north and south, are many quartz veins of various widths from a few inches to 10 feet and even more. These quartz veins carry free and almost pure gold; for example, at the Balaghat mines gold is about 97 per cent. pure as it comes from the mill. Many of the quartz veins are barren, although the quartz to all appearances is the same in one vein as another. The quartz is either white or gray, both carrying some or no gold as the case may be.

The quartz veins which showed gold at the surface were worked out by the natives a long time ago. There are no records extant of these old workings, but tradition indicates that the work ceased with the invasion of the Mohammedans under Arumgzeeb in the latter part of the seventeenth century. Be that as it may, in several places, both at Kolar and in other fields of South India, the natives did work out all the surface quartz down to depths varying from 60 to 300 feet. The present gold mines are working on the same veins at much greater depths.

The natives seem to have prospected this field thoroughly in the old days, for, although there are many quartz outcrops, none of them carry any gold, from which it is quite clear that whatever gold was at the surface was found and removed. So great is the confidence of the present-day mining engineers in the thoroughness of the work of the ancients, that there has been no systematic prospecting of these fields, except at or near old workings. No doubt some day in places where there are barren quartz outcrops with placer gold in the ground all about, there will be serious effort made to find out what exists at depth. There are many such places.

The Goldfield Ridge is one long and nearly straight upheaval bounding this last formation on the west. It is the most interesting feature to the student of geology of anything in this region. The combinations are mostly iron and quartzite, but, for variety in lamination, crumpling, and folding, as well as variety in colors and texture, it would be difficult to duplicate. It seems certain that the "ridge" is a comparatively recent upheaval. There is no laterite on it, although but half a mile from it is a hill 50 feet higher whose top is covered with laterite, from which it may be concluded that if the ridge had been in existence at the time of the laterite drift it would also have been covered with laterite.

Where the rocks of the ridge are in place the stratification is practically vertical, but there has been so much twisting and squeezing that all the forms of lamination read about in geologies are exemplified in countless places. In height the ridge is 50 to 300 feet above the surrounding country.

It should be added that there are masses and streaks of quartz (white) in the ridge, which in some places look suggestive of gold. In some places these bodies of quartz have been prospected but no gold of importance found, and the quartz itself suddenly terminated or gradually pinched out.

The Mines.—There are six producing mines in the Kolar field, all of which have been working with various vicissitudes for nearly 20 years. Another half dozen or so which were worked in the past are now idle. In some cases the idle mines had at one time good ore, which was worked out, while in other cases they used up their capital in prospecting.

All these mines are on the two principal veins, which have various short branches extending here and there from them. The principal veins are not entirely continuous either in quartz or in gold, the latter being mostly in "ore shoots" of varying vertical depths and lengths, all leading to the north along the veins.

The dip of the veins is about 30 degrees from the vertical, west, so that most of the mines have both inclined and vertical shafts.

Only one mine has sufficient water for its own needs. One does no pumping whatever—at all events it is called a perfectly dry mine, although down 300 feet.

The ore is free milling and each mine does its own crushing, amalgamating, and cyaniding. The process of ore treatment is identical at all the mines—stamping, amalgamating, and treating the tailing with cyanide solution. Details in treatment vary with the ideas of the superintendents rather than because of any essential difference in the ore. Some, for example, prac-

tice both battery and table amalgamation, and others only table. The slime is rather abundant and gives considerable trouble.

There is no opportunity to take advantage of gravity in arranging the milling plants, owing to the contours of the ground which furnishes no side hills, consequently the tailing has to be elevated to considerable height, the larger mines having tailing dumps as much as 50 feet high—often higher, and covering several acres. Mine labor is cheap; men, 9 cents a day; women and boys, 6 or 7 cents, above ground. So the tailing especially is handled by manual labor.

Any description of this field would be incomplete without mention of the great Mysore Mine, which began milling operations in 1884, with total amount of ore crushed of 454 tons, from which 454 ounces of gold was recovered. Two years later the mine paid $150,000 in dividends. The next year the dividends fell to $75,000, but since that have gone on increasing until the annual dividends have reached as high as $2,000,000.

ADDENDA.—At the thirtieth annual meeting of the Mysore Gold Mining Co., held in London, England, in March, 1910, the following information was given out: The Mysore government released the property to the company for 30 years at an additional royalty of 2½ per cent. of the output; however, the government reduced the price of electrical power from £24 to £10 per horsepower per annum, which probably offsets the increased royalty. The working costs, including royalty and income tax, absorb 33 per cent. of the total value of the ore. "The mine employs 10,000 men, women, and children, 3,000 of them miners, and the medical staff looks after 90,000 souls." "Whenever cholera breaks out, the healthiest place in the whole of India is Mysore camp." The ore reserve exceeds 1,000,000 tons, and the ore treated amounted to 234,500 tons, from which was recovered 228,249 ounces of gold, or 18½ pennyweights of fine gold per ton During the year, 1,053 feet of shafts and 754 feet of winzes were put down. The deepest point in the mine is 4,175 feet on the dip of the deposit. The mine has distributed since its commencement in 1880, £5,935,094 in dividends to shareholders. E. B. W

ORE MINING NOTES

The construction of the first practical gold dredge in California was in 1898. At present there are 63 gold dredges in the state, representing an investment of $8,995,000. From 1898 to 1908 over $25,000,000 in gold had been produced by gold dredging. With the advent of the modern dredge, handling from 250,000 to 300,000 cubic yards of dirt monthly at a cost of from 2½ to 3 cents per cubic yard, the industry has steadily expanded. The unsightly stone piles are broken up for macadam and concrete work, and the soil, which had been exhausted, was renewed by dredging. As an illustration, land that cost the dredging companies $25 per acre is now sold for agricultural purposes for $100 per acre, after it has been dredged and leveled. Eucalyptus and fruit trees grow on unleveled dredge tailing.—*Lewis E. Aubury.*

The Newhouse tunnel, 21,968 feet in length, is now completed. From the portal at Idaho Springs, in Clear Creek County, it extends northward under many of the principal mines of the oldest mining district in the state, to a terminal point in the Gunnell Mine at Central City, in Gilpin County. From the veins cut by the tunnel at depths ranging from 1,500 to 1,800 feet, the production of precious metals has been in progress for more than half a century. In many instances the operation of the mines finally became unprofitable on account of the expense involved in pumping water from the shafts. By connecting the lower workings of the mines with the tunnel, this expense can be eliminated, and the available ground for profitable ore production in Gilpin County practically doubled. Although the full benefits of the tunnel will not become apparent until the principal connections have all been made, a steady gain in the production of the mines affected may be expected.

The mining industries of Chili employ 60,000 and the railroads 65,000 people. Since the country's discovery gold has been mined valued at $222,923,005, silver at $311,093,058, copper at $658,575,153, and nitrate at $1,122,874,274. The 20,632 mineral claims cover 522,336 acres, of which 10,821 are copper, 3,893 gold, 904 silver, 86 gold and silver, 530 gold and copper, 173 iron, 99 manganese, 32 aluminum, 1 antimony, 1 nickel, 4 mica, 338 sulphur, 1,385 nitrate, 687 salts of potash, 399 salt, 4 diamond, 33 china clay, and 40 coal.

According to information received from the mining department, gold mining in private mines in the Orenburg district, Russia, is yearly increasing. In 1909 the amount of gold obtained was about 21,060 ounces, or 2,268 ounces more than in 1908. Reports from 83 gold mines state that in 40 work was done exclusively under the individual-effort system (pay according to the amount of gold mined by each miner); in the remainder of the mines 800 miners were employed under the boss system.

In making an excavation for a farm building near Hayford, Wash., gold nuggets were found. A chicken was killed and examined and other colors found. It is becoming tiresome to keep suggesting to Washington farmers that they should keep ducks on account of their scooping capacity.

The reorganization committee of the Ely Central Copper Co. has perfected the plan for reorganizing the company, and the details have been sent to the shareholders. The formation of a $12,500,000 company to be known as the Ely Central Development Co., which will take over the present $16,000,000 corporation, is the salient feature of the plan.

During the past year 30 miles of trenches were excavated in prospecting on the Nipissing property. Several veins were discovered, but in the future the surface will be prospected by hydraulicking.

In drifting in the Hudson Bay property, Cobalt district, Canada, at 150 feet in depth, a new silver vein from 2 to 5 inches wide was discovered. It is a solid compact vein worth 3,000 ounces to the ton. At the 100-foot level this company has what is termed the new Big Vein 12 inches wide, carrying around 6,000 ounces silver to the ton.

Because their car was delayed in unloading at the Deloro smelter the T. & H. B. Co. made a clear $3,000 profit. Silver rose while the car waited. There are many Cobalters who would be pleased to be held up in the same manner.

The mineral resources of Ontario are being developed by American capital. Large tracts of iron ore north of Kingston have been purchased by an American company, which is installing machinery preparatory to exporting the raw material to the United States. The actinolite mines at Bridgewater have passed into the hands of an American company. American machinery has just arrived here for a factory which will manufacture mica washers. Large exports to the United States are being made from the mica mines at Sydenham, owned and controlled by the General Electric Co. American capital is also invested in the feldspar mines near Verona, the total output of which is sent to American potteries. A lead mine a short distance from Kingston will be managed by an American and will have American machinery. The opening up of these various mines means an increased demand for American mining machinery, as well as coal. The Nova Scotian mine owners, however, are endeavoring to have the Canadian customs duty on coal increased, so as to shut out the American product, and to have the duty increased on pig iron.—*Felix S. S. Johnson, Kingston, Canada.*

The Negociacion Minera Santa Maria de la Paz y Anexas, located at Matehuala, S. L. P., Mexico, which is one of the largest mining companies in Mexico, has recently installed and just placed in operation two large vertical triplex Goulds pumps to handle their mine water. These are duplicates and each has a capacity of 250 gallons per minute against 700 feet working

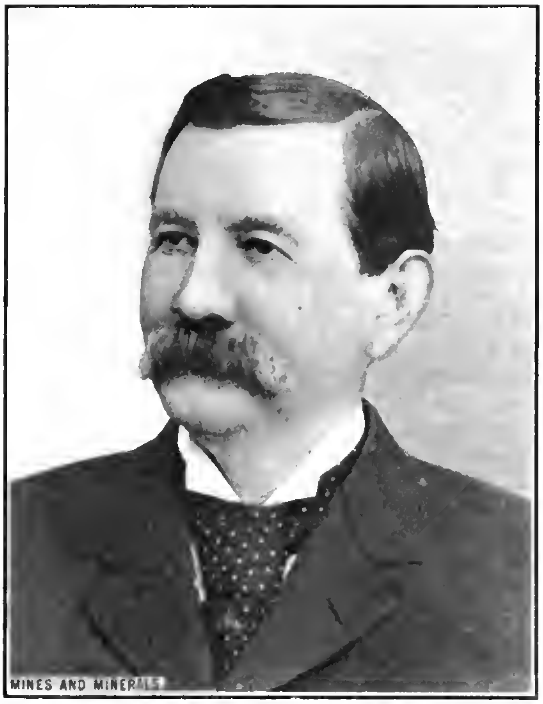
head. Both are operated by direct-connected electrical motors, one of the pumps being located at the bottom of the mine shaft, elevating the water half way to the top; the second pump handling the water from there to the surface. The motors employed are 75-horsepower Westinghouse. A generating plant on the surface supplies these motors with a 400-volt, 60-cycle current.

The Rio Plata Mining Co. has been chartered under Mexican laws; its subsidiary, the Compania Minera Rio Plata, will go out of existence. The mine operated by the company is in Chihuahua, Mexico, 85 miles from the Kansas City, Mexico & Oriental Railway. The output, amounting to about 85,000 ounces of silver a month, is carried by mule to Sanchez, the nearest railway station.

OBITUARY

HENRY FAIRCHILD DE BARDELEBEN

Henry Fairchild De Bardeleben, who was practically responsible for the Birmingham, Ala., iron and steel industry, died December 6, 1910, at his home in Birmingham, aged 69. He entered the Confederate army in 1861 as a dragoon; married



HENRY F. DE BARDELEBEN

Miss Ellen Pratt in 1862. He prospected the Birmingham, Alabama, district, in 1872, and becoming associated with Colonel Sloss and M. H. Smith, made the first ton of iron in Alabama at the Oxmoor furnace. Colonel De Bardeleben by industry, and perseverance in overcoming difficulties amassed a comfortable fortune and business. False friends in the Tennessee Coal and Iron Co. induced him to consolidate his interests with theirs, assuring him that they would lend him sufficient stock in addition to that which he received for his properties so he would control the company. His friends insisted on his signing an agreement not to sell his stock below a certain figure, then turned bears and hammered the stock loaned until he had put up all his own as margin. When that happened he was sold out. Most men at his age would have crumpled under such treatment, but not De Bardeleben; he started anew and succeeded. After the death of his first wife he married Miss McCrossin, of Birmingham, who with five children will miss one of the best of husbands and fathers.

THOMAS BELLIS

Coal men in Pennsylvania will regret the death of Thomas Bellis, which occurred November 21, at his home in Altoona. Mr. Bellis was born in Buckley, North Wales, and came to this country in 1880. He first located in Philipsburg, Clearfield County, where he engaged in mining with Thomas Barnes and other pioneers of that section. Later he went to Burnside, where with James Passmore, he operated the Bellmore Coal Co. Mr. Bellis had the distinction of being the first practical coal man in the Indiana County field and he superintended the opening of the first mine at Hastings.

CHARLES H. TUCKER

The late Charles H. Tucker, secretary and treasurer of A. Leschen & Sons Rope Co., St. Louis, who died in Clifton Springs, N. Y., October 30, was one of the best informed and most extensively known wire-rope men in this country, having been closely identified with that great industry in various capacities, for 26 years.

Mr. Tucker was born in New York City 50 years ago. As a ooy he took employment with the Western Union Telegraph Co., subsequently going into newspaper work, serving as a dramatic critic on the New York *World*. While yet a young man, he entered the New York office of John A. Roebling's Sons Co., growing steadily in their service until 1898, when he was secured by A. Leschen & Sons Rope Co., with whom he remained until his death.

STEWART WATT

Stewart Watt, vice-president and superintendent of the Watt Mining Car Wheel Co., died at his residence at Barnesville, Ohio, on December 10, at the age of 64. He was ill of bronchial pneumonia but a few days. Mr. Watt, with his brother, who died several years ago, was the joint inventor of the first self-oiling mine car wheel, and for the last 35 years has been actively engaged in the manufacture of mine-car wheels. He first operated a foundry in 1863.

WALTER L. PIERCE

Walter L. Pierce, who for thirty-two years had been connected with the Lidgerwood Mfg. Co., and for twenty-nine years its secretary and general manager, died suddenly at his winter home in the Hotel St. Andrews, New York City, on December 10, 1910. He was born at Dorchester, Mass., on June 8, 1855. He leaves a widow and son, Walter L. S. Pierce.

THE YEAR IN THE JOPLIN DISTRICT

Written for Mines and Minerals, by L. L. Wittich

The Webb City-Carterville camp, as has been the case for many years, led in the production of both zinc and lead ores in the Joplin district for the year just closed. Joplin was a not very close second, although the record for the year was encouraging and showed that the ore deposits lack a great deal of being worked out.

As a whole the 1910 shipments rank favorably with those of 1909, when the tonnage was the greatest in the history of the district, although the values in 1909 were not quite up to those of the record-breaking year of 1907.

The new year dawns with conditions favorable for another epoch of great prosperity. As was predicted at the beginning of 1910, the period has been one of unexcelled activity in this district, which includes more than 20 camps in Missouri, Kansas, and Oklahoma. The bulk of the zinc and lead production comes from the Missouri camps, although the newly-discovered Oklahoma mines, which have increased steadily in production, continue to be producers of a great tonnage of mineral, the lead ore output of this region being of especial note.

The year opened with zinc blende selling for $49, basis of 60 per cent. while the top price for choice lots running better

than 60 per cent. was $52 and the average price, all grades, was $46.50. The year closes with the basis price of 60 per cent. grades running about $47, while $50 is the maximum paid for extra choice qualities. Within these 12 months the basis price has been down as low as $39.50, for the week ending May 7, and this basis figure represents the highest basis paid. From this figure the price ranged down to $36 a ton, carload lots.

After that week the market showed indications of reviving and there is not another instance throughout the year when the high basis went lower than $41. Spelter, East St. Louis quotations, which started the year at $6.12½ per 100 pounds, dropped below $6 before the close of January and no more $6 metal was reported throughout the year, although it climbed up to $5.90 in November. At its lowest mark, spelter dropped to $4.90 for the week ending May 7.

Conditions in the zinc ore market remained about the same through the months of June, July, and August, the situation strengthening materially at the close of August when blende jumped to $43.50 basis, reaching $45 basis in September and $48 basis in November.

The shipments of blende have ranged from 3,400 tons per week to 6,700 tons a week, the heaviest production being noted in the closing months of the year. The total valuations for all ores sold, including blende, calamine and lead, have ranged from $189,114 per week, to $357,821 per week, and the total valuation for the year is in excess of $14,000,000.

While certain camps of the district have decreased in production, others have gained materially, the growth of the Alba-Neck City district in the north part of the field being attributed to the discovery of rich pocket deposits of high-grade zinc blende. The mines in this camp have produced a heavy tonnage during the past year, and the value of the ore has been above the average.

Galena, Kans., likewise has seen a revival of mining activities, and this old camp, which led the production in the pioneer days, but which has dropped to third place, has a showing that reminds the old timers of the early times.

The production of calamine has not been as heavy as expected and many small properties discontinued operations. In the production of this ore, Granby, in Newton County, Mo., has led the list by a big margin. Other camps that have shown a good production of calamine are Spring City and Duenweg, both being in Missouri. Calamine prices followed the general trend of blende offerings, the price being weak when the blende price slumped, and high when the price of blende was high. The year started with calamine selling for $28 a ton, basis of 40 per cent. and closed with the price at $27 a ton. Choice lots brought higher than these figures.

Lead ore brought $58 a ton at the opening of the year but the price slumped to $48 in May. Toward the close of the year the offerings became stronger, the price going to $58 a ton. The erection of a large smeltery in the Webb City camp is expected to increase the demand for the local product. East St. Louis pig lead quotations throughout the year ranged from $4.20 per 100 pounds to $4.70.

MEETINGS OF SOCIETIES

The annual meeting of the National Civic Federation is on January 12, 13, and 14. Great national problems will be considered and committees organized to propose legislation. The program includes discussion upon regulation of combinations and quasi-public utilities, compensation for industrial accidents, and mediation and arbitration laws. The Federation's compensation department will present a draft for model uniform state laws. Constitutional features are now being considered by uniform commissioners in 32 states. Application of Canadian arbitration law to industrial disputes concerning street railways and other municipal utilities through uniform state laws will be considered.

The School of Mines of the Pennsylvania State College is cooperating with miners' institutes in Pennsylvania. Practical lectures on the "Prevention of Accidents" are being delivered in various parts of the state by Dean W. R. Crane of the School of Mines.

ELECTRIC SIGNAL TO SHAFT BOTTOM, ENGINE, AND WEIGH ROOM

Written for Mines and Minerals, by M. M. Haley

Electric or other signaling appeals at all times to mine managers, but so far the most crude apparatus has given the best satisfaction. The electric signal described, is of more than ordinary interest, as it is controlled from the weigh room as follows: By placing the movable contact lever *a*, Fig. 1, on any one of the three contacts, *b*, *c*, *d*, the desired signal can be given immediately to both the hoisting engineer, the cager, and men at the shaft bottom. The signal system can be successfully operated as follows: By placing the contact lever on the *b* contact it will show the red light for "stop signal," on the *c* contact it will show the clear light for "all is well," and on the *d* contact it will show the green light for "did not dump," or "lost check," etc.

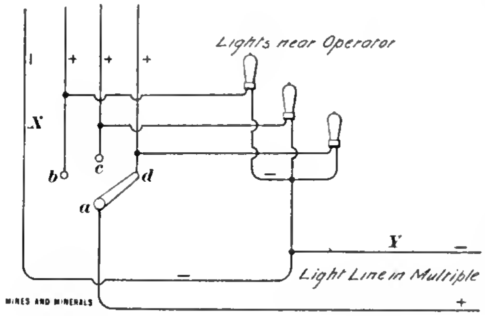


FIG. 1

By this method, if a car is being placed under the chute, or something happens that coal cannot be hoisted, the bottom man and the engineer are both aware of the fact at once and the cager will not be giving loaded signals while the top equipment is blocked. The lights are controlled by the weighman in the weigh office in the tipple.

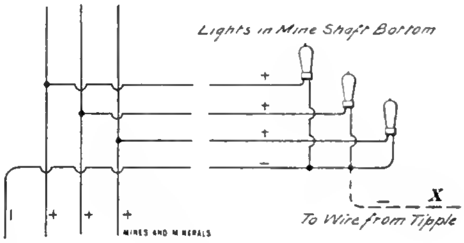


FIG. 2

In the event a mine is using a bonded rail, or a negative wire is close at hand, or at least closer to the bottom than the top of the shaft, the line *X*, Fig. 2, may be attached to same, thereby affording a light when the circuit-breaker falls on the bottom. When coupling out the rail or wire, the line *Y*, Fig. 3, may be attached to the negative of the lights if there be any in the hoisting room, or brought out and coupled to the line *X* coming from the bottom, or if the *X* line is not used, the former may be coupled to the *Y* line instead. The *X* line is only used when there is no rail or feeders accessible at the shaft bottom. It will be noticed that the lights should harmonize, otherwise the signal would be of no service, for instance, the red light on the contact in the weigh office should be on the corresponding contact to that of the red light in the shaft bottom.

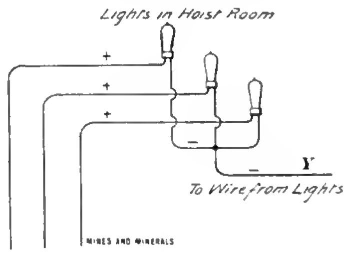


FIG. 3

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

Western Editorial Office: 728 Equitable Building, Denver, Colo.
Chicago Office: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to Mines and Minerals, Scranton, Pa.

CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER.......Manager
EUGENE B. WILSON, Scranton, Pa.......Editor
GEORGE F. DUCK, E. M., Denver, Colo.......Western Editor
P. G. MOORE.......Circulation Manager
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., Ltd., Dryden House, 43 Gerrard St., Soho, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 Pitt Street, SIDNEY.
SOUTH AFRICAN AGENT: E. HERBERT BYSSHE, CAPE TOWN, Mansion House; JOHANNESBURG, Trust Building; DURBAN, Natal Bank Chambers; BLOEMFONTEIN, Florence Buildings.

SUBSCRIPTION RATES

Single Copies.......25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada.......2.50
To all other countries.......3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always gives your FORMER as well as your present address.

TABLE OF CONTENTS

Tooele Smelter....... 321
Cost of Core Boring....... 323
The Old Dominion Pumping System, by R. L. Herrick....... 324
Joplin Zinc and Lead Ores....... 327
American Institute Panama Meeting, by E. W. Parker....... 329
A Graphical Notebook, by F. W. Gray....... 332
Vanadium Steel....... 334
Gold Deposits of San Juan, Colo., by Warren C. Prosser....... 335
Metal Mine Ventilation....... 337
Slime Settler or Dewaterer, by Rollo E. Huntley....... 339
Zinc Mining at Yellow Pine, Nev., by Newman B. Gregory....... 340
Lake Superior Copper Ore....... 341
De Wilde Precipitation Process, by G. Witteveen....... 342
Personals....... 343
Grading Analyses, by H. Stadler....... 344
Limit of Proportionality....... 349
Geology of Kolar Gold Field, by C. S. Durand....... 350
Ore Mining Notes....... 351
Obituary—
Henry F. De Bardeleben—Thomas Bellis—Charles H. Tucker—Stewart Watt—Walter L. Pierce....... 352
Electric Signal to Shaft Bottom, Engine, and Weigh Room, by M. M Haley....... 353
Editorial—
Employers' Liability—Legalized Slavery—Progress in Land Classification—Guests on Mining Institute Excursions....... 354
Starkville—An Explanation, by Geo. F. Duck....... 355
Book Review....... 355
The West Virginia Institute Meeting....... 356
Catalogs Received....... 357
Illinois Coal Statistics....... 357
Evolution of Hoisting, by E. B. W....... 358
Gas Blowers in Coyote Mine, by A. A. Galloway....... 364
Cement Concrete Vats and Tanks, by Albert Moyer....... 364
Two-Stage Air Locomotives....... 365
Explosibility of Coal Dust....... 369
The Mine Law of West Virginia, by P. A. Grady....... 370
The Organ Feedwater Heater....... 371
Coal Mining Notes....... 373
Delagua, Colo., Explosion, by Geo. F. Duck....... 374
Examination Questions—
Fire Bosses' and Engineers', Terre Haute, Ind., August, 1910....... 381
New Inventions....... 384
Squibs—
Alaska Gold Production, 1909, 326—Rio Blanco County, Colo., 328—Temiskaming Mine Shaft, 331—Total Gold Production of Alaska, 339—Meetings of Societies, 353—World's Calcium Carbide Production, 1908, 369—Swedish Steel Rails, 372—Tariff on Breccia, 380—Conservation....... 384

EMPLOYERS' LIABILITY

Coal mine operators are becoming dissatisfied with insuring in casualty companies against loss from accidents in their mines. Statistics show conclusively that the premiums paid are out of all reasonable proportion with the amount of indemnity that reaches the sufferers, and frequently whatever is not paid involves a lawsuit. Under the present system of accident insurance, less than 40 per cent. of the premiums paid by operators is apportioned to those for whom it was intended, and nearly one-half of the 40 per cent. is taken as the result of an indirect method of disbursement in practice generally—that is, paying indemnity through the courts. This, however, is but one phase of the matter which, with others, has induced the National Civic League to attempt the passage of a uniform liability law. In nearly every state in the union more or less agitation along this line is being carried on which will eventually produce results.

The Cleveland-Cliffs Iron Co. and the United States Coal and Coke Co. have found the injustice of the present liability laws so intolerable, they have formulated plans and are carrying them out independently of the statutes provided for such cases. This speaks volumes for the need of a common law governing liability, that shall make it unnecessary for one or two corporations to make their own laws in order to have one worth while. R. P. Tarr has written an article for Mines and Minerals on "Compensation for Injury" which will appear in February, 1911, issue.

Murray M. Duncan's article on "Systems of Compensation to Workmen in Cases of Injuries Not Due to Negligence" was printed in volume 30, page 166, Mines and Minerals, under title "Insurance of Mine Workers."

Simeon Reynolds recently wrote an instructive article for the West Virginia Mining Institute, on "What Shall We Do With Our Industrial Wounded."

LEGALIZED SLAVERY

The United States Supreme Court will shortly decide whether that abomination called "the contract labor law" shall continue. The supreme court of Alabama holds that the law is legal, being a proper exercise of the police powers. While the Alabama contract labor law does not draw the color line, no white men are working as convict miners, showing that it was enacted to ensnare the negro laborers, that politicians might graft. Alabama has a law compelling coal mine operators to pay the miners $1 per ton for coal. The scheme is to force a contract on some ignorant farm laborer, then for some trifling alleged violation, drag the culprit into court and have him convicted for committing a misdemeanor, which may be punished by hard labor and long sentence. Once in jail the man is farmed out to a contractor who feeds him and works

him in the coal mine, receiving $1 per ton for all coal the convict mines.

Greenhorns from the farms are sent into gassy mines; and whipped if they do not dig their allotment of coal, and shot if they attempt to escape. Under trumped-up charges and such conditions they may be kept at work indefinitely. Some have tried to kill themselves, and others have run the risk of being killed by the guards rather than submit to the slavery.

The United States Department of Justice contends that this treatment of men is a form of peonage, and that the Alabama law was not enacted to prevent fraud, but to encourage law breaking and impose involuntary servitude. If the system is pronounced legal then it is up to the large corporations owned by northern capital to refuse to hire convicts as they are now doing.

PROGRESS IN LAND CLASSIFICATION

GREAT progress in public-land classification is being made by the United States geological surveyors who during the year 1910 classified 5,618,769 acres as "coal land" and who also appraised and placed its market value at $380,955,646. "At the minimum rates fixed by law the price would have been $87,816,599," the increase thus being nearly $300,000,000. "Here surely, is practical, productive work," the press bulletin states; evidently because the price of public coal land has been boosted on paper from $20 to $400 per acre.

Coal land in the East can be purchased for much less price. The editor is in a position to sell 1,500 acres of coal land to the government for $200 per acre, and as it has railroad transportation the government could put a conservation price of $400 on it. If "the practical productive work" continues in the West, the sale of coal lands will not pay for the Panama canal, but they will be conserved.

GUESTS ON MINING INSTITUTE EXCURSIONS

MOST mining engineers are as good fellows as their wives, and both believe in sharing the interesting and good times incident to mining engineers' excursions. This has been the custom since the establishment of the American Institute, and may it continue. In going over the list of those attending the American Institute Canal Zone meeting one will find that of the 120 in the party, 37 were ladies, and this is certainly as complimentary as anything that could be said for the members.

STARKVILLE—AN EXPLANATION

Written for Mines and Minerals, by Geo. F. Duck

Later developments show that some slight corrections should be made in the article which appeared in the December number of MINES AND MINERALS, describing the Starkville explosion.

The statement that the motor was found with power off and brakes set is not correct. It would appear that the motorman lost control of his trip near the top of the "hill," and that at that point or nearby it broke in two. This is sustained by the fact that the motor was found in parallel, with the controller in the fifth notch and the brakes off The motor was also off the track and into the rib.

Another interesting feature of the rescue work was the excellent discipline established shortly after the accident and maintained until the end. Soon after 10 o'clock that night the placing of electric lights at proper places near the drift mouth was almost completed, and before 6 the next morning comfortable sleeping and boarding quarters had been provided for all connected with the work.

Telephones played an important and interesting part in the work. Each rescue party was followed by another with canvas and they temporarily rebuilt the brattices. As soon as circulation was restored a party of linemen entered with necessary insulated wire and instruments, and it was generally not 2 hours before each rescue station was connected with a central exchange at the drift mouth. At each station a man to answer calls was left in charge with four helmets and 50 potash cartridges for emergency use. The rescue stations, in order of their establishment, were at the large fan at the fourth south, at the "booster" fan on the new haulage entry, and on one of the K entries further in, each station being abandoned as soon as the one further inby was established.

The coroner's jury, after being out 5 days, reported, charging gross negligence in not properly watering a dusty mine known to generate gas at times.

BOOK REVIEW

The first bulletin issued by the new Federal Bureau of Mines is entitled, "The Volatile Matter of Coal," and is by Horace C. Porter and F. K. Ovitz, who conducted their investigations at Pittsburg. The results of the fuel investigations, commenced several years ago by the Technologic Branch of the Geological Survey, showed that the work of determining the fuel values of the coals and lignites in the United States would be incomplete if it did not include chemical researches into the processes of combustion. This bulletin is a report of the volatile matter in several typical coals, its composition, and amount at different temperatures of volatilization. Quoted directly, the authors say: "The investigation has shown that the volatile content of different coals differs greatly in character. The volatile matter of the younger coals found in the West includes a large proportion of carbon dioxide, carbon monoxide, and water, and a correspondingly small proportion of hydrocarbons and tarry vapors."

GOLD DREDGING IN CALIFORNIA, Bulletin No. 57, compiled under State Mineralogist Lewis E. Aubury. Published by the State Mining Bureau, San Francisco, Cal. The bulletin contains 312 pages, 9 in. × 6 in., 235 illustrations, with 10 maps, and abounds in useful information, statistics of the dredging industry in California and its probable future. Bound copies of the bulletin can be had for $2, with 20 cents postage. Mr. Aubury says: "Should other gold forms of mining maintain their average, 2 years from now California will probably again recover its rank as the leading gold-producing state in the Union."

In a 96-page booklet entitled "High Efficiency of Centrifugal Pumps," the DeLaval Steam Turbine Co., of Trenton, N. J., have brought together some 75 charts, diagrams, photographs, and a vast amount of engineering information relating to such subjects as ways and means for testing centrifugal pumps, charts of the results of such tests, the interpretation of these charts for the purposes of the engineers, etc.

GEOLOGICAL MAP OF COBALT, fourth edition (enlarged), by Willet G. Miller and Cyril W. Knight, has recently been published by the Ontario Bureau of Mines, Ottawa, Canada. The fourth edition covers not only the geology embraced in that of the first, but it includes a complete map of the Gillies Limit, not heretofore published, and the map of South Lorrain, by A. G. Burrows, which was published a few years ago. Other areas, for instance, the Quebec shore of Lake Temiskaming, are included. The routes of the various power lines and other features have been added to this map. It naturally gives a more comprehensive view than the preceding editions of the geology of the Cobalt area.

"A MANUAL OF PRACTICAL ASSAYING," seventh edition, by the late H. Van F. Furman, revised and enlarged by W. D. Pardoe. In the new edition, the chapters that have been rewritten are on the determination of silica, lead, copper, cobalt, nickel, and vanadium, a method for the determination of uranium, as well as vanadium being given. Diagrams of the new Shimer crucible and train of apparatus used in carbon determinations have also been added. A description of the Pearce method for the determination of tin, the bromate method for antimony, a volumetric method for bismuth, Neher's method for arsenic, and additional methods for sulphur and tungsten have been incorporated. The book is so well and favorably known little comment is needed. Mr. Furman was the first to incorporate modern chemical analyses of the rarer metals in book form. His book instantly found favor in colleges and among engineers, so that at the present time it is a standard. The 1899 edition contained 463 pages; the present edition, 8vo., 530 pages, the price remaining the same, $3. The publishers are John Wiley & Sons, New York; Chapman & Hall, London.

THE BOARD OF TRADE MINE, WHEELING, W. VA.

THE BUSINESS PROSPECTS YEAR BOOK, 1911, is the name of a book published by the Business Statistics Publishing Co., 12 James Street, Cardiff, Wales. The Business Man's Guide, as it is also termed, was initiated in 1906, to deal with the conditions of trade in the past and as guide for the future. The contents are Coal, Iron, Copper, Tin, Tinplates, Oil, Money Market, Shipping, Rails, Wheat, Cotton, Rubber, Hog Products, Dairy Produce, Statistics. The book contains 279 pages and is 5 shillings net.

TESTING FOR METALLURGICAL PROCESSES. Much money has been lost by assuming that some particular process is adapted to an ore and then putting up a plant for that process. The wise man has his ore tested in a laboratory to ascertain if the ore is suited to a process, and on favorable reports has a carload or less tested on a commercial scale. James A. Barr has written a book descriptive of the apparatus and procedure in making laboratory tests, entitled "Testing for Metallurgical Processes." The contents of the book are the results obtained by experimental work, and the methods he recommends have been tested in practice. The book will appeal to engineers and students in metallurgy or chemistry. Where a man follows a certain line he is unable to keep all details in his head, and when comparatively new tests are to be made he must review, and often is compelled to read considerable unnecessary material to arrive at the details sought. This book does away with all that, provided the reader has a general idea of the subject. The book contains 216 pages and is illustrated. The price is $2 postpaid. The Mining and Scientific Press, San Francisco, and the Mining Magazine, Salisbury House, E. C., London, are the publishers.

THE WEST VIRGINIA INSTITUTE MEETING

The winter meeting of the West Virginia Mining Institute held December 6, 7, and 8, at Wheeling, W. Va., was well attended.

The papers presented dealt with practical matters that tend for economy and progress in coal mining. They were as follows: "Profit and Loss Account," Neil Robinson; "Application of Compressed Air to Mine Locomotives," C. B. Hodges; "Possibilities of a Federal Liability Law," Simeon Reynolds; "Commercial Classification of Fuel," Robert E. Rightmire; "American Language," J. F. Healy; "Mine Law of West Virginia," P. A. Grady; "Non By-product Retort Ovens," E. B. Wilson; "Mine Motors," Everett Drennen; "Local Geology of the Wheeling District," I. C. White; "Accurate Cost Accounting," J. C. McKinley. Demonstration in first-aid work was given by J. J. Rutledge and assistants.

The Wheeling Board of Trade, with aid of the Elks, entertained the visitors, as Dr. J. H. Holmes said, in a manner that, other cities in the state will have difficulty in duplicating. Visits were made Wednesday to the Wheeling Steel and Iron Co., Northwood Glass Co., Wheeling Stamping Co., Riverside and Warwick pottery companies, and other industries. The Board of Trade was assisted in the movement of the trolley cars conveying the excursionists by the Wheeling Cascaret and stogie companies. In the evening the Board of Trade entertained with a combined banquet, vaudeville, and mine explosion in the Elks auditorium.

The stage setting represented a West Virginia hillside in winter, with real hemlock and laurel covered with artificial snow.

To the left was a timbered drift mouth; to the right an engine house equipped with moving machinery and a live steam whistle. In the background between the two there was a mountain stream tumbling down the hillside.

George W. Lutz, president of the Board of Trade, acting as mine inspector, appeared at the mine mouth with a safety lamp,

spoke to engineer, Chas. F. Brand, who then called Mine Superintendent James P. Morgan with the whistle. Inspector Lutz left his safety lamp outside and with Superintendent Morgan entered the mine with an open light. After the inspection tour Mr. Lutz read his report, which was about 10 feet long of closely unwritten paper, he then became toastmaster. "He seen his duty and he done it." The guests having been served with an "oyster cocktail," Prof. H. M. Shockey, musical director, pressed the piano keys and put out the lights. Thereupon four dusky miners, A. H. Robinson, Wm. S. Leach, David Crawford, and Howard Nesbitt pushed a real mine car containing real coal out of the drift mouth, singing as they did so, "Down in a Coal Mine Underneath the Ground." "The Legend of the Coal Mine," with music and words arranged for the occasion by Judge Alan H. Robinson, was delightfully rendered by David Crawford and chorus. After repeated encores the audience were requested to join in singing "Down in a Coal Mine." The noise was so spontaneous that the professor took his hands off the piano keys and the lights went up. The impression prevails that Engineer Brand blew his whistle to hot water; "he seen his duty and he done it." During the next course the band and the diners made a duet from "Old Kentucky Home," "Old Black Joe," and other never-grow-old melodies. Again the lights went dim and Miner Robinson, leaning on the coal car, sang the "Stein Song." In the chorus to this song the quartet of miners hammered on drills. Were it not for the stage setting and the dim lights from the miners' lamps the spectators could have imagined they were listening to the Anvil Chorus sung by grand opera performers. As it was, they repeatedly encored the vocalists, until Mr. Shockey took his hands from the keys and turned the lights on so the members could assist the band in playing "Every little movement has a meaning all its own."

One of the hits of the evening was Mine Superintendent Morgan's monologue burlesquing several coal operators, and his leading the illustrated song "The Miner's Home." The vaudeville ended with the song "The Fire in the Mines." This was not all song: the mine blew up; flames shot from the drift mouth; the whistle blew itself to hot water; and during the excitement J. W. Kay, of the United States rescue service, equipped with helmet, rushed into the mine and brought out a supposed unfortunate.

The following officers were elected for the ensuing year: President, Frank Haas, Fairmont; First Vice-President, Neil Robinson, Charleston; Second Vice-President, Joseph Virgin, Plymouth, W. Va.; Third Vice-President, P. E Grady, Huntington; Fourth Vice-President, George Watson, Fairmont; Fifth Vice-President, J. C. McKinley, Wheeling; Secretary-Treasurer. E. B. Day, Pittsburg, Pa.; Executive Board, J. F. Healy, D. Howard, R. S. Ford, and Charles Connor.

The next meeting will be held in June, 1911, at the White Sulphur Springs.

CATALOGS RECEIVED

In writing for catalogs, please mention Mines and Minerals

THE ALDRICH PUMP DEPARTMENT, Allentown, Pa., Pump Data No. 14-A, The Aldrich Vertical Twin Triplex Pump, 16 pages.

AUSTIN SEPARATOR CO., Detroit, Mich., Catalog No. 16, Austin Steam and Oil Separators, 56 pages.

AMERICAN SHIP WINDLASS CO., Providence, R. I., Mechanical Control of Air and Coal, 15 pages.

THE BRISTOL CO., Waterbury, Conn., Bulletin No. 140, Bristol's Recording Gauges, 48 pages.

EAGLE FOUNDRY AND MACHINE CO., Fort Scott, Kans., Ventilation of Coal Mines, 8 pages.

ALLIS-CHALMERS CO., Milwaukee, Wis., Compressed Air for Industrial Purposes, 12 pages.

LINK-BELT CO., Philadelphia, Pa., Book No. 102, "Maximum" Silent Chain, 40 pages.

THE STANDARD TOOL CO., Cleveland, Ohio, Catalog No. 16, Twist Drills, Small Tools, 303 pages.

SULLIVAN MACHINERY CO., Chicago, Ill., Bulletin No. 58-H, Sullivan Small Air Compressors, Steam Driven: Classes "WA-4" and "WA-3," 12 pages; Bulletin No. 58-I, Sullivan Air-Compressor Accessories, Useful Compressed-Air Data, 32 pages.

JOSEPH DIXON CRUCIBLE CO., Jersey City, N. J., Graphite Products for the Railroad, 38 pages; Graphite, Vol. XII, 12 pages.

GENERAL ELECTRIC CO., Schenectady, N. Y., Bulletin No. 4721-A, Thomson Direct-Current Watt-hour Meters, Types C-6, C-7, and CQ, 16 pages; Bulletin No. 4775, Type KS, Single-Phase Induction Motors, 8 pages; Bulletin No. 4778, The Edison Carbon Incandescent Lamp, 14 pages; Bulletin No. 4782, Direct-Current Exciter Panels, 8 pages.

WESTERN ELECTRIC CO., New York, N. Y., Bulletin No. 1116, Magneto Telephone Sets and Accessories, 40 pages.

A. LESCHEN & SONS ROPE CO., St. Louis, Mo., Hercules for Heavy Duty of Every Kind, 8 pages; Pamphlet Descriptive of Wire Ropes, 20 pages.

JOHN DAVIS & SON (DERBY), LTD., 110 West Fayette Street, Baltimore, Md., Catalog Describing Mining Instruments, Safety Lamps, Etc., 20 pages.

STROMBERG-CARLSON TELEPHONE MFG. CO., Rochester, N. Y., Flat Type Metal Bell Box and Portable Desk Telephone, 4 pages; Pamphlet No. 33, Private Branch Exchange Swithboard, 8 pages.

STURTEVANT MILL CO., Boston, Mass., Sturtevant Crushers, 16 pages.

ILLINOIS COAL STATISTICS

Summary for the Years Ended June 30, 1909, and 1910.

	1909	1910
Number of counties producing coal......	55	55
Number of mines and openings of all kinds	886	881
Total output of all mines, in tons of 2,000 pounds..........	49,163,710	48,717,853
Number of shipping or commercial mines	384	390
Total output of shipping mines, tons	47,958,562	47,225,201
Average days of active operation for shipping mines..........	189	179
Average days of active operation for all mines..........	168	171
Average value per ton all grades at shipping mines..........	$1.012	$1.016
Aggregate home value of total product...	$50,303,757	$50,204,207
Average value per ton all grades at all mines..........	$1.023	$1.031
Number of motors in use underground...	210	216
Number of mines in which mining machines are used..........	107	114
Number of mining machines in use......	1,246	1,291
Number of tons undercut by machines...	16,407,692	18,176,254
Number of tons mined by hand.........	32,756,018	30,541,599
Average number of miners employed during the year..........	50,834	39,069
Average number of other employes underground..........	13,788	28,137
Average number of boys employed underground..........	1,752	1,154
Average number of boys employed above ground..........	71	47
Average number of other employes above ground..........	6,288	6,227
Total number of employes..........	72,733	74,634
Number employed at shipping mines....	69,518	71,520
Number of persons at work underground.	66,374	68,360
Number at work on surface..........	6,359	6,274
Average price paid per gross ton for hand mining, shipping mines..........	$.583	$.597
Average price paid per gross ton for machine mining..........	$.460	$.462
Number of kegs of powder used for blasting coal..........	1,2[illegible]0,607	1,254,095
Number of kegs of powder used for other purposes..........	3,963	3,128
Number of men accidently killed (Cherry mine disaster 259)..........	213	390
Number of men injured so as to lose a month or more of time..........	894	737
Number of gross tons mined to each life lost..........	230,816	124,917
Number of employes to each life lost	342	191
Number of deaths per 1,000 employed...	2.9	5.2
Number of gross tons mined to each man injured..........	54,993	66.103
Number of employes to each man injured.	93	101
Number killed to each million tons produced..........	4[illegible]	8.0
Number injured to 1,000 employed......	10.7	9.9

EVOLUTION OF HOISTING

Written for Mines and Minerals, by E. B. W.

(Continued from December)

Hoisting Machinery for Deep Mines. Development of Electric Hoisting Machinery

Quite a number of mines have depths ranging between 2,000 and 3,000 feet, a less number whose depths range between 3,000 and 4,000 feet, and there are a comparatively few in the Transvaal, South Africa, and Keweenaw Peninsula, Mich., whose depths range from 4,000 feet to nearly 5,000 feet, vertical. Hoisting from depths greater than 3,000 feet is a problem that few mining or mechanical engineers are called upon to solve, and when they are, the experience derived from deeper shafts is studied with most satisfactory results, although the various designs show that a single type of hoister has not as yet satisfactorily furnished a standard that meets all requirements. This is due primarily to the miners' demands prevailing over the economical desires of the mechanical engineer, and as a rule direct-acting engines with cone drums of various designs are as common as when hoisting from less depths. The necessity of constructing machinery that may be controlled at great depths and will move the cages and loads with precision adds somewhat to the task, but strength and freedom from stoppages due to breakdowns or complicated parts going wrong are of greater importance to the designer.

When a shaft is down 4,000 feet its cost represents a cash outlay between $500,000 and $750,000, and the interest on this must be paid by hoisting a certain tonnage of ore daily; in addition, the fixed charges connected with mining, dressing and treating the ore require that another certain tonnage be hoisted, thus making it absolutely necessary that no shutdown shall occur from the frailty of the hoister.

Usually the mine is capable of producing a certain daily tonnage of ore, and the mill is designed to treat that tonnage; if anything happens to the hoister the mill and treatment plants must shut down, and although the capacity of the hoister is greater than the mine, the mill and mine cannot catch up with the lost time.

Fig. 39. Drum of Hoist, Osceola Mine

The Red Jacket shaft of the Calumet & Hecla Mine at Calumet, Mich., has six compartments 6 feet 3 inches by 7 feet, a length over all of 25 feet, and a width over all of 15 feet 6 inches. It is one of the deepest vertical shafts in the world, being 4,900 feet from the collar to the sump. The company is notable from the fact that it has paid $110,000,000 in dividends to stockholders, is most progressive, and has the finest machinery probably of any mining company.

This machinery for the deep shafts was designed by E. D. Leavitt, who adopted the Whiting hoisting system at the Red Jacket. There are two hoisting plants at the shaft, one for sinking, which has 7-foot diameter rope wheels, and the other, for hoisting ore, with 19-foot diameter rope wheels. In each plant the rope makes three half turns in the grooves of the rope wheels, and as there are two such wheels coupled with parallel connecting-rods, the rope has the bite due to six half turns.

Another innovation at this plant is the introduction of duplex, triple-expansion engines, with the high and intermediate cylinders acting on one crosshead, the low-pressure cylinder on another, the two crossheads being attached by connecting-rods to the opposite ends of a short triangular walking beam pivoted between two guides.

From the third point of the beam a connecting-rod carries motion horizontally to one crank of the main sheave. The cranks of the two engines are set quartering.

Probably nowhere else in the Michigan copper districts are expansion engines used, and here the ratio of rock hoisted per ton of coal is given as approximately 40 to 1. Other companies on the Peninsula favor the simple non-condensing engines for hoisters, that are direct acting and work at a steam pressure varying from 80 to 120 pounds per square inch.

These hoist in balance with drums having partly conical ends and using round ropes. The drum shown in Fig. 39 is that used at the Osceola Mine and is shown merely as an illustration of the size of a Michigan drum. Its size is 18 feet 6 inches long by 12 feet 6 inches in diameter.

The Tamarack shaft is vertical and No. 3 shaft is probably the deepest in the world, 4,990 feet. No. 5 shaft, which is 4,935 feet deep, has five compartments 7 feet 2 inches by 5 feet 2 inches, four of which are for hoisting, and one a pipe and ladder way. The hoister at this shaft, shown in Fig. 40, has four engines of the Corliss type, placed on a massive triangular frame and direct-connected to the drum, which is 25 feet in diameter at the center, and tapering to 18 feet at the ends. The crankshaft is 37 feet 5 inches long from center to center of crankpins, with single cranks at each end set 135 degrees apart. When one of the four engines is on a dead center, the effort of the other three engines is applied at 45 degree-, 90 degree-, and 135-degree points of the crank circle. From 18 to 20 trips an hour are made with this hoister, which is as accurate in stopping and starting as a small one. The engine was designed to carry 6,000 feet 1½-inch rope, but the load is 17,519 pounds of rope, 4,200-pound cage, 4,000-pound car, and 1,200 pounds of rock, or 18.85 tons, and this it hoists at a velocity exceeding 90 feet per second, or about a mile a minute. A trip from the bottom to the landing is made in from 70 to 90 seconds, but men and materials are handled as well as rock, so that, with the caging and landing, the average number of trips per hour is 19. At No. 3 shaft a supplementary hoist handles men and timber.

In the Transvaal a company can only follow the ore deposit to a point perpendicular to its surface boundaries. The first openings were made along the outcrop of the deposit which has a dip of about 35 degrees that flattens somewhat with depth, and as a matter of policy and necessity shafts are sunk for development work as shown in Fig. 41.*

The outcrop is at *a*; the first line of vertical shafts are at *b*; the second line of shafts at *c*, are known as deeps; and the third line at *d* as deep deeps, the latter reaching the deposit at between 4,000 and 5,000 feet. Other shafts are sunk to cut the deposit and then curve as in Fig. 42, going down the dip to avoid the necessity of an underground hoist. The illustration shows the method adopted for curving the east shaft of the Vogelstruis Consolidated Deep Co., with the objects of avoiding an underground hoister and enabling development to proceed with the least possible delay.† The skip used for hoisting runs from the incline into the vertical smoothly, and generally without increase of speed. On the curve, 45-pound rails are used and the skip is guided by 4″×4″×½″ angle iron. The center of the rope pulleys is on a 73-foot 6-inch radius, and the bottom of the rail on an 80-foot radius, thus giving the latter a 48-degree curve.

* J. S. Lane, Vol. 24, page 594, Mines and Minerals.

† Thos. Haight Leggett, page 970, Transactions American Institute Mining Engineers, 1901.

It is thought that this curved shaft can be used to 3,000 feet, but that after that depth independent methods of hoisting on the incline will need to be adopted.

At the deep mines of the Simmer and Jack group, located at Elandsfontein, about 12 miles from Johannesburg, South Africa, is a hoister designed and installed by J. S. Lane. The engines are cross-compound condensing, 28 in. × 50 in. × 72 in., designed to work with 140 pounds steam, and the following description furnished by Mr. Lane is of interest, showing as it does the numerous problems to be met.

Example of Deep Hoisting.—This plant is located at the Hammond shaft, Elandsfontein, about 12 miles from Johannesburg. The Hammond shaft is one of the deep mines of the Simmer and Jack group, belonging to the Consolidated Gold Fields of South Africa.

The hoist was furnished by Chapin & Manion, Ltd., Johannesburg, dealers in mining machinery and representing the Ingersoll-Sergeant Drill Co. The plant was designed and built in America. It is cross-compound condensing, 28 and 50 in. ×72 in., designed to work with steam at 140 pounds. Both the high- and low-pressure cylinders are steam-jacketed, both on the barrel and heads. The main object of steam-jacketing, in this case, is to keep the cylinders hot and in readiness to start promptly at any time without danger from water, as it is best to leave the steam in the jackets all the time day and night. The high-pressure cylinder exhausts into a reheating receiver, the exhaust steam passing through 2-inch tubes surrounded by steam at boiler pressure. Steam is admitted to the high- and low-pressure cylinders through balanced throttles, both of these throttle valves being connected to the same lever on the engineer's platform. A secondary throttle is provided by which live steam may be admitted, if desired, into the receiver, between the high- and low-pressure cylinders; but it is only necessary to use this valve when first starting for the day, or after an unusual stop during working hours. An automatic back-pressure valve permits exhaust steam to go to the atmosphere should the vacuum fail. Corliss valve gear is used, but there are no eccentrics, the wristplates being driven by rods connected to cranks on either end of a cross-shaft located between the drums and engine cylinders, and driven by a train of gears from the main engine shaft. The gears being fitted with an adjustment operated by an air or steam cylinder which reverses the engine by revolving the cross-shaft forward or back to give the proper reverse motion and valve lead. The main engine shaft is $17\frac{1}{2}$ inches in diameter in the cranks and $22\frac{1}{2}$ inches in diameter in the cast-iron center driving wheel.

The two cylindrical drums are loose on the shaft and bushed with composition bushings put in in halves and removable for repairs. Each of the four drum-hubs is fitted with eight arms made of 9-inch channel, each carrying steel drum flanges, on to which are riveted angle irons carrying the drum shell; on to the outer flanges of each drum there is riveted a cast-iron brake seat or ring. The shells of the drums are 14 feet in diameter and 51 inches face; they are in four sections with a spiral groove $1\frac{45}{100}$-inch pitch turned in them for the rope. The $1\frac{1}{4}$-inch rope is wound in two layers giving a capacity of 3,000 feet on each drum. The drums are each driven by a band friction driver which clutches them to the center driving wheel, the clutches and brakes being operated by cylinders using either compressed air or steam.

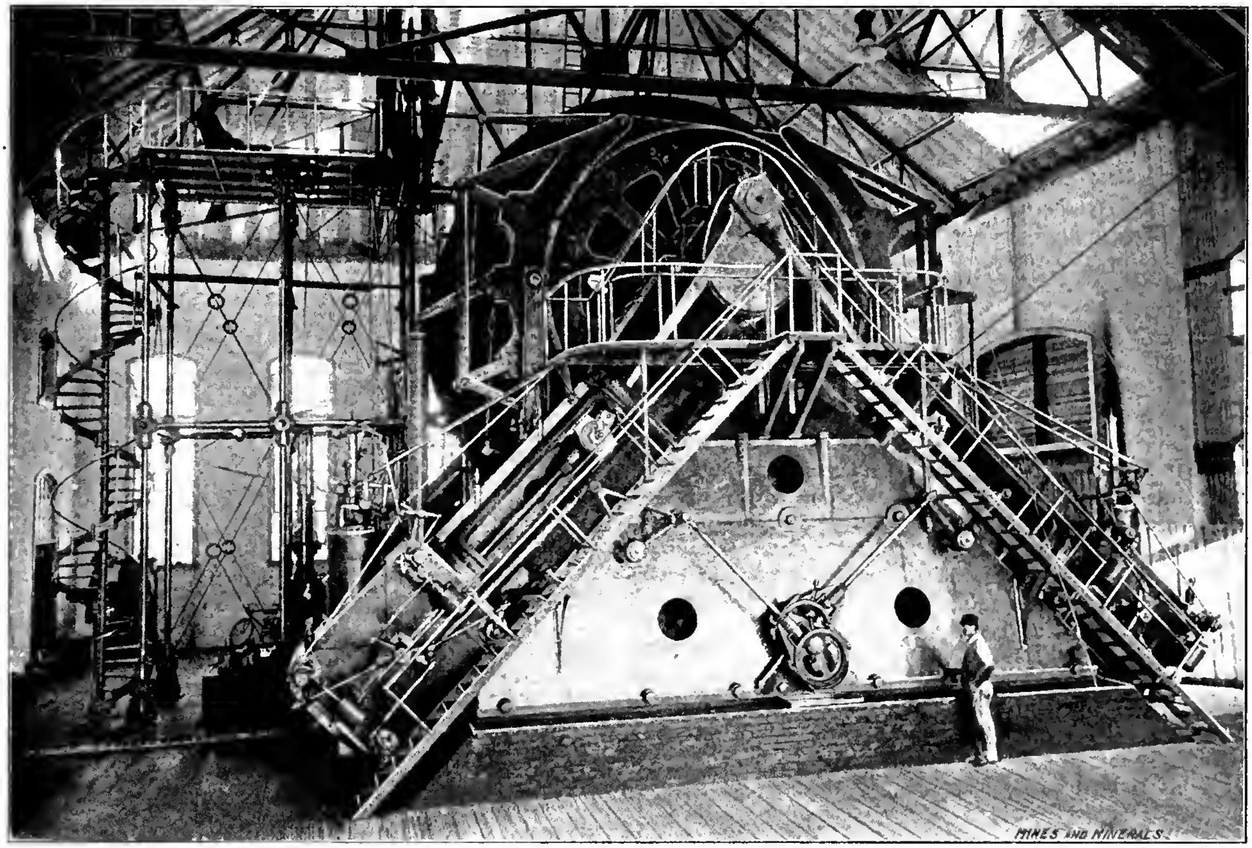


Fig. 40. Hoist at No. 5 Shaft, Tamarack Mine

The plant was designed to hoist in balance, the descending skip balancing the ascending skip, but the clutches were provided to permit adjustment of length of rope in order to change the hoisting from one level to another, and also to allow the operation of either skip independently in case of temporary derangement in either hoisting compartment.

The brake seats, or rings, on each drum are 10 inches wide. The post brakes are unusually heavy and move parallel simul-

taneously at top and bottom, and are put on by means of weighted compound levers, and released by cylinders which lift the weight. The combined weight is 1,850 pounds and the leverage 36 to 1, or equal to 66,600 pounds pressure of the brakes against the brake seat. There is a brake on each drum, and when hoisting in balance, as is usually the case, the brake levers

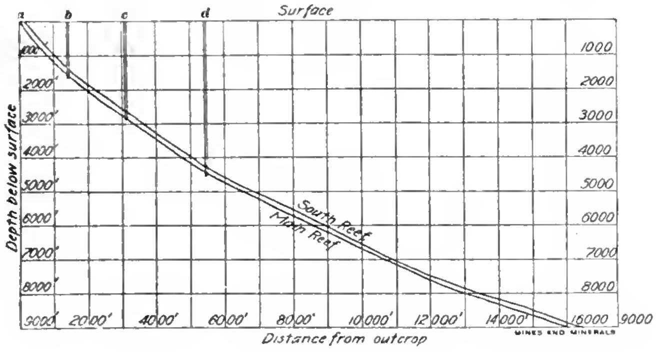


FIG. 41

are coupled together so that the motion of one moves both and the two pairs of brakes work as one.

Oil cataract cylinders are provided to regulate the motion given by the air or steam cylinders on both brake and reverse engines; the valves of both the air and oil cylinders are operated through a "floating valve motion" following the motion of the hand lever, stopping whenever the motion of the hand lever stops.

The point of cut-off can be varied by the engineer without interfering with the action of the governor when running slowly, and yet the governor will take control of the cut-off should the maximum speed be reached at any time.

There is one indicator for each drum, consisting of a light steel shell 24 inches in diameter. The surface of these indicator drums is painted black so that chalk marks and figures can be made indicating the different stopping points. A pointer moves up and down on a screw in front of the indicator drum. A spiral white stripe or line is painted on the surface of the drum, and the drum revolves with the hoisting drum, being connected by link belt and gearing; it makes a simple and graphic indicator, as only the marks on the front side of the drum can be seen at a time; a second pointer moves up and down by a vertical scale on the frame and indicates, approximately, where the skip is.

Should either skip be overwound the indicator nut comes in contact with a lever that disengages a weight, allows it to drop, and instantly, not only applies the post brakes, but also detaches all of the Corliss inlet valves, allowing them to close; nor can they again hook on to admit steam until the automatic stop is reset. The simultaneous closing of the inlet valves, closes the cylinders too, and the application of both post brakes' full power will stop the skip promptly. This device is especially valuable in case the engineer should by mistake open the throttle without reversing the engine; were it not for the automatic stop the skip would likely be against the head-sheave before the engineer realized what he was doing, but with the automatic stop the engine would be promptly stopped.

The engineer's platform, 7 feet square, on which are placed all the levers for operating the plant, is located between the cylinders.

Both high- and low-pressure cylinders have relief valves, all operated by a single lever on the platform. By opening these valves, steam is allowed to by-pass back and forth from one end of the cylinder to the other; these are used for controlling the speed in lowering men or timber, and may also be used for slowing after the main throttles are shut, thus saving the brakes.

The piston rods are extended through the back cylinder heads, and provided with slipper foot-guides. This is not usual in American practice but was required by the parties interested. The distance from center to center of engines is 21 feet 9 inches, and the length of plant 48 feet, including the tail-rod guides.

The shipping weight of this plant, not including the weight of condensers and foundation bolts, was 448,000 pounds; the net weight 415,000 pounds. This plant was designed to handle a total load of about 20,000 pounds including rope, skip, and rock.

ELECTRIC HOISTS*

The evolution of electric hoisting has been confined practically to Europe, where, due to the poorer natural resources, it has been necessary to introduce all technical advancement possible in order to meet competition. As far as electric hoisting for main shafts is concerned the beginning of serious work

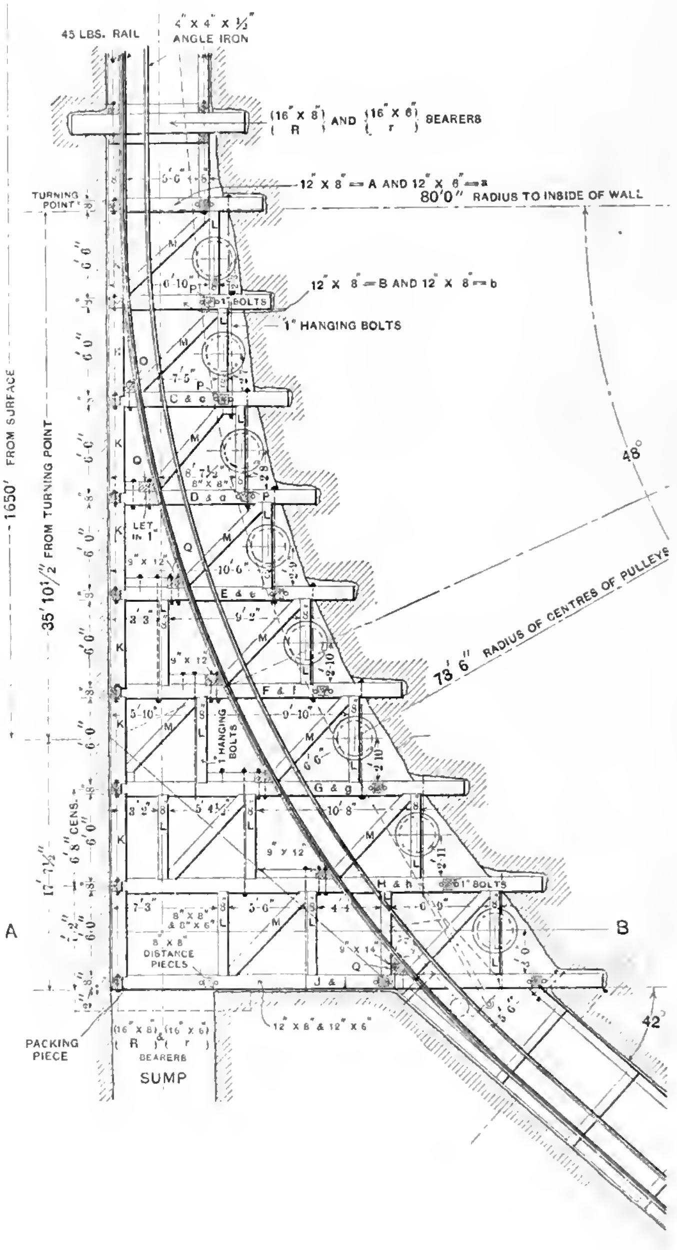


FIG. 42

dates from 1902. Before this time there were various causes operating for and against the introduction of electric hoisting. The mine operators who had electrified their properties were naturally desirous of using electricity for hoisting as well as for

*Written for MINES AND MINERALS by Wilfred Sykes, Elec. Engineer, Westinghouse Electric & Mfg. Co.

other operations. On the other hand, there was a feeling that electrical equipments had not proved to be sufficiently reliable or convenient in operation. The uneconomical performance of the then existing steam hoisting engines was also a very serious argument in favor of the introduction of electrical hoists, and the manufacturers of electrical apparatus were, of course, desirous of extending their fields of application. They were, however, reluctant to attempt this problem without considerable investigation as to the risk involved. From an engineering standpoint, the question of control was probably one of the most serious features to be overcome.

In previous articles on this subject, reference has been made to the local conditions with various types of hoisting plants. It will be appreciated that with steam-operated hoists, the acceleration and running speed are limited only by the size of the cylinders and drums, there being a great deal of latitude possible in the operating characteristics by varying the cut-off of the engines. In connection with electric hoisting plants, the thermal capacity of the motors makes it necessary to consider the maximum conditions when laying out a plant, and the

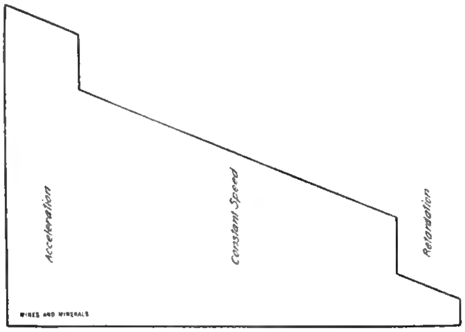


FIG. 43

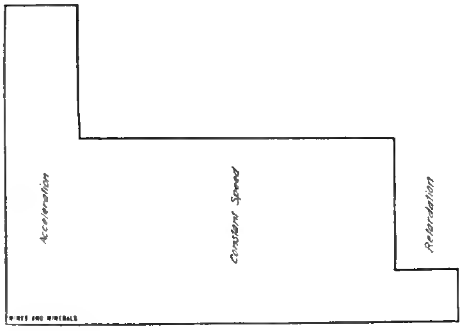


FIG. 44

question of acceleration is very much more important than is the case with steam plants. As the heating of machines varies approximately as the square of the load, it is important in order to reduce the size of the equipment to make the load during the operating period as uniform as possible. For this reason the development of electrically-driven hoists has also affected to some extent the mechanical design of hoisting plants. In Europe, the Koepe has been used to a very great extent with the object of reducing the inertia of the moving parts to a minimum, it being possible to do this on account of the even turning moment of the electric motor. With a plant of this kind, driven by steam engines it is necessary to counteract to a great extent the advantages by using sufficient flywheel effect to keep the angular irregularity to such a value that the rope will not slip.

The effects of the characteristics of the various types of hoists on the size of motors has led to a more careful study of load conditions than is usually made in connection with steam plants, and a brief statement of the characteristics of the various types of equipments may not be out of place.

Fig. 43 shows a characteristic load diagram for a double, cylindrical-drum, balanced hoist. The high peak load is due to the acceleration of the moving parts. After they have been brought to full speed the load gradually decreases, due to the variation in the weight of the suspended rope on the two sides. At the end of the hoisting period the energy stored in the moving parts becomes available during retardation and may be used in overcoming the static load or it may be necessary to

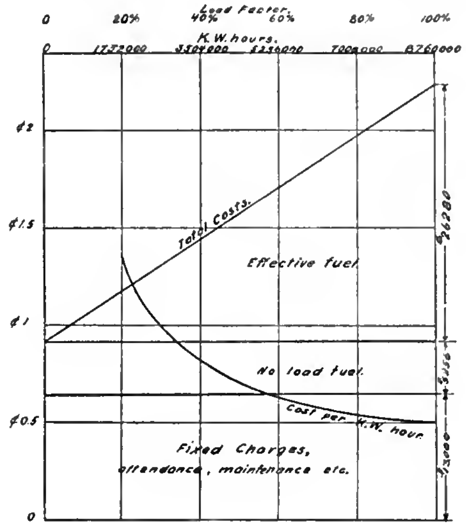


FIG. 45

absorb a portion of it with the brakes. With mines of great depth, the variation in load due to the varying weight of the suspended rope becomes very important and in certain cases it may be found necessary to use the brakes even before the beginning of the retarding period. The difference between balanced and unbalanced winding is that the static load when working unbalanced is greater than when working balanced and the change of load due to the rope is only half as great.

The second type of diagram covers the Koepe pulley, or a cylindrical drum using a tail-rope. It will be seen from Fig. 44 that the static load is constant, the accelerating peak being the prominent feature as before. This type of diagram also holds good for conical drum hoists providing that the drums are so dimensioned as to completely compensate for the unbalanced rope load. The reel hoist using flat rope also has the same characteristics providing the diameter of the hub is correctly

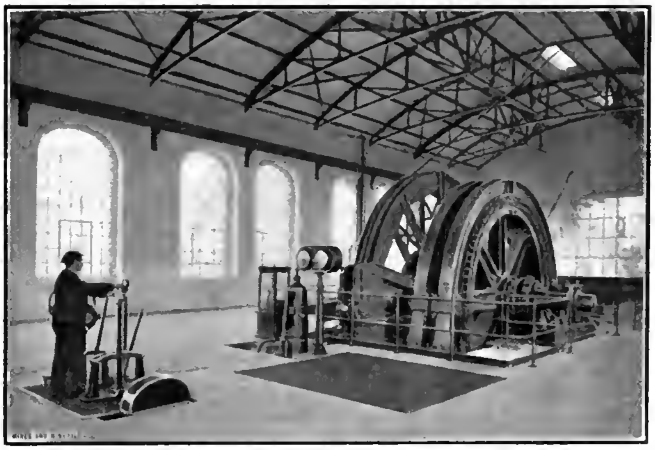


FIG 46. ELECTRIC HOIST, KOEPE TYPE

chosen with regard to the depth and the thickness of the rope used.

The effect of such loads upon the power house is obviously very severe when motors of several thousand horsepower are required to perform the work. The regulation of the power house, and consequently the operation of the other machinery

supplied with power from it, is very seriously affected by such fluctuations when the hoisting load bears a large proportion to the total capacity, as is generally the case. Under exceptional circumstances, where very large power hoists are used solely for industrial purposes, such fluctuations may not be of very great importance. The most serious feature of such a fluctuating load, however, is the effect upon the costs of producing

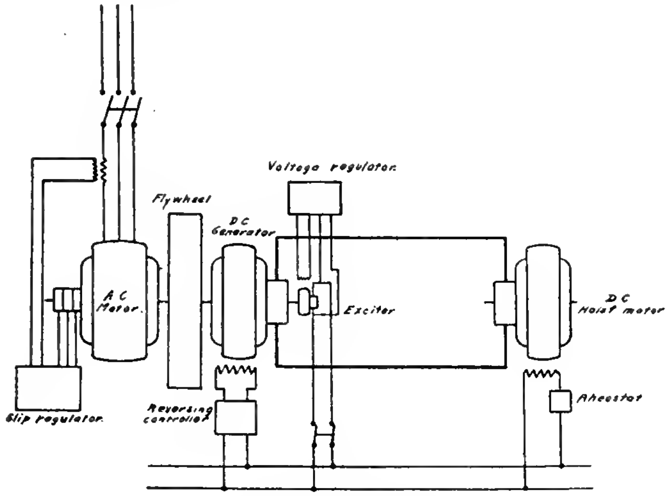


Fig. 47. Ilgner System

power. The average load in relation to peak loads may be comparatively small, and in this case the generating equipment must be run underloaded most of the time. When it is considered that the fixed charges of electrical plants very often exceed 50 per cent. of the total costs, the effects of running at light loads will be readily appreciated.

Fig. 45 shows the effect of varying loads upon a small industrial plant of 1,000 kilowatts capacity, this, of course, being applicable only for a certain case, but it indicates the effect of high-load factor. From this diagram it will be seen that the fixed charges, attendance, maintenance, etc. amount to $13,000 per year. The fuel and water required to run the plant light

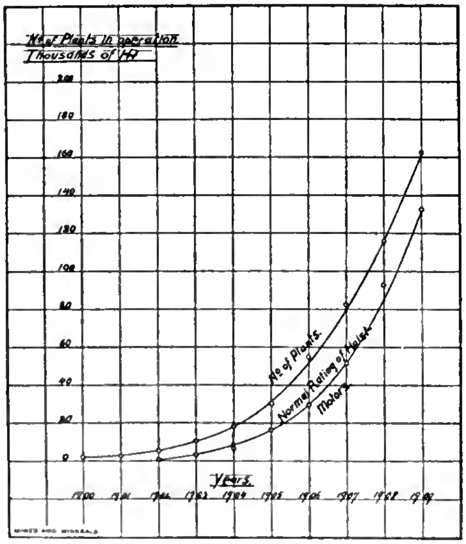


Fig. 48

loaded are taken as $5,256, so that if the plant ran without a load the total cost would be $18,256. The additional fuel required to operate a plant when generating power will vary practically directly with the load. In the case of the plant under consideration, the cost of fuel when running at 100 per cent. load continuously would be $26,280 per year. Coal has been figured as averaging $3 per ton delivered. From the curve drawn on this diagram showing the cost per kilowatt-horsepower, the effect of the load factor is very plainly shown—the power cost at 20 per cent. load being approximately 1.3 cents per kilowatt-horsepower, and at 100 per cent. load .5 cent per kilowatt-horsepower. The load factor of the ordinary hoisting plant seldom exceeds 20 per cent. so that the cost of power is high in relation to what it would be with the same continuous load.

In 1902 the Harpener Bergbau Aktien Gesellschaft installed the first large electric hoist, which is illustrated in Fig. 46. This hoist had a capacity of 4,800 pounds, and a full speed of 3,150 feet per minute, the motor having a maximum capacity of about 1,400 horsepower. It will be seen from the illustration that the hoist is of the Koepe type, being direct coupled to a three-phase induction motor with a wound rotor. The control apparatus for this motor is mounted below the floor level and operated by the levers shown in the illustration. The effect of the intermittent loads upon the generating station, which is of moderate capacity, is such that it is generally found necessary to run the hoist on separate generators, otherwise the regulation is so much affected that the operation of the other apparatus is seriously interfered with. This hoist has been working very satisfactorily since the installation, but the effects of the intermittent load upon the generating plant rather hampered the developments along these lines. The question of control was overcome by using a special liquid starter of very large capacity.

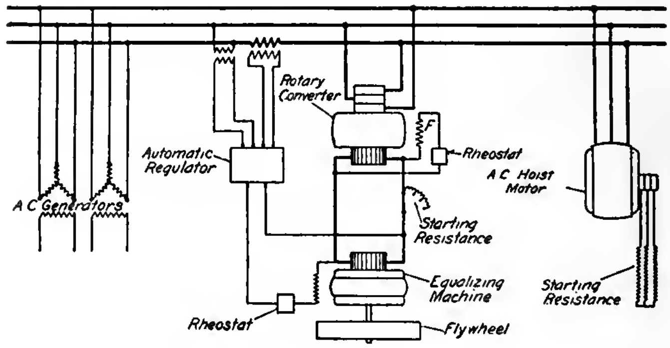


Fig. 49

About the same time, another hoisting system began to receive attention. This system employs a motor generator for supplying power to the hoist, the hoist motor being a direct-current, shunt-wound machine with separately excited field. The operation of the hoist is controlled by varying the voltage of the generator, to which it is directly connected electrically. By reversing the excitation of the generator the direction of rotation of the motor is also reversed. It will be seen that the control problem is very simple with such a system, as it is only necessary to handle a very small current. At the same time rheostatic losses are obviated, it being possible to run the hoist at any speed that may be desired. The characteristics of the shunt-wound hoist motor are such that when lowering materials it acts as a generator and returns energy through the motor generator to the line. This may also take place when the hoist is being retarded so that the mechanical brakes are only used to hold the load. This system was first used for hoisting purposes in the plant of the Gewerkschaft Hollertszug, which began operation in 1895. This arrangement does not obviate the intermittent nature of the hoisting load, but it eliminates the control problem, which is one of the most serious in connection with electrical-hoisting plants and at the same time it is somewhat easier on the generating plant, as the load is applied more gradually than is the case with direct rheostat control.

It was obvious, however, that electric hoisting plants could hardly be expected to compete with large steam-operated equipments unless some method was found to equalize the input so as to improve the load conditions on the generating plants.

In the beginning of 1903 the Ilgner system of hoisting plant was introduced. This system employs a motor generator set for supplying power to the hoist motor, which is of the shunt-wound type. The control is similar to the system previously supplied. Fig. 47 shows diagrammatically the essential features of the scheme. In addition, however, a flywheel is connected to the motor-generator set and arrangements are made to automatically vary the speed so that during peak-load periods the speed of the set is decreased, and part of the energy in the flywheel is used to assist the motor in driving the generator. When the load drops below a certain value the speed of the set is gradually increased and energy is again stored in the flywheel. By properly proportioning the flywheel, assuming that the cycle of operation remains constant, it is possible to keep the input to the hoisting plant within a few per cent. of the average load. It will be seen that this system overcame the principal objection to electric hoisting equipments and opened the way for the development of the largest equipments likely to be required in practice. The question of control, which was very troublesome, is also eliminated in the Ilgner system of operation. During the period from 1903 to 1906 a number of plants were installed working on this system, for maximum capacities up to about 2,000 horsepower. This time may be considered more or less as an experimental period, as the mine operators were anxious to obtain reliable data as to the operation of the various plants installed. In Fig. 48 the total number of plants and the normal continuous rating of the hoist motors used is shown for the various years. It will be seen that from 1906 onwards the rate of increase of electrically operated hoisting plants has been very great. The curves shown cover the plants in operation in Europe, and they represent an investment for electrical equipments of approximately $6,000,000. The curves show only equipments used for main-shaft hoisting and do not take into consideration haulage plants or small auxiliary hoists.

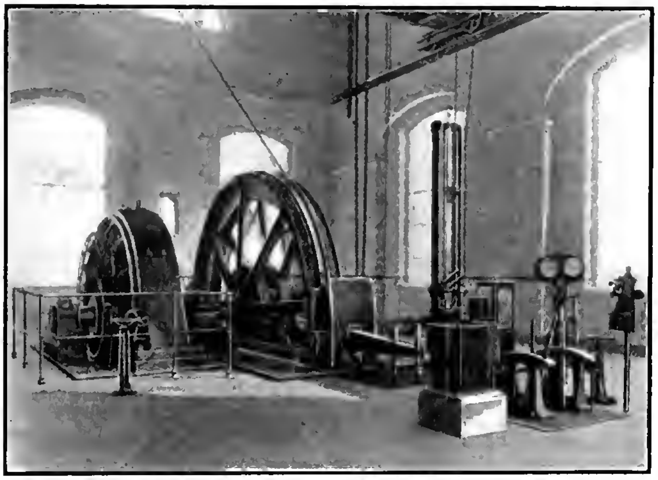


Fig. 50. Electric Hoist, Koepe Type

Fig. 51. Ilgner Type, Flywheel Motor Generator Set

Among the plants in operation may be mentioned that at the Gelsenkirchener Bergwerke A. G. which has a maximum capacity of 19,300 pounds and a full speed of 4,000 feet per minute, the motors having a maximum rating of 3,560 horsepower. One objection to the Ilgner system of hoisting is the expensive nature of the plant, the addition of a motor generator and flywheel increasing the cost considerably. To overcome this feature and still to provide for the equalization of the input, a system has been introduced by the British Westinghouse Co. which may be used under certain circumstances. This scheme is shown diagrammatically in Fig. 49. The hoist motor in this system may be either direct current or alternating current, depending upon the source of supply. The diagram of connections, Fig. 49, shows the arrangement of the plant with an alternating current source of supply. In parallel with the generators, an equalizing outfit is arranged, which consists of the direct-current machine coupled to the flywheel which is connected to the alternating-current system through a rotary converter.

This equalizing equipment can be located anywhere that may be convenient, it not being necessary to have it near the hoist. The operation is as follows: When the hoist load exceeds the value for which the regulator is set, the field of the equalizing machine is automatically strengthened, so that the speed tends to drop and the machine is driven as a generator by the flywheel and delivers energy through the rotary converters to the alternating-current system, the rate at which the energy is delivered being dependent upon the operation of the regulator. When the demand drops below the value for which the regulator is set, the field of the equalizing machine is automatically weakened and it then runs as a motor, absorbing energy from the alternating-current system through the rotary converter and speeding up the flywheel. In this way the demand on the alternating-current system is kept practically constant. When this system is used with a direct current source of supply the rotary converter is omitted and the equalizing machine connected directly to the line. It will be seen that this arrangement does not provide for controlling the hoist motor, as is the case with the Ilgner system, but it has the advantage that the equalizing machine has only to deal with the loads in excess of the mean value for which the regulator is set, and the cost is considerably reduced compared with the former system. It also has the advantage that, should any part of the equalizing equipment fail, it does not interfere with the operation of the hoist motor. The application of this system is practically confined to cases where the rheostatic control of the hoist motors offers no difficulties and where equalization of input is all that is required. In case of very large plants the control question is of such importance that the Ilgner system is used almost exclusively.

One important feature in connection with electric hoisting is the ease with which safety devices can be arranged to prevent overwinding or overloading. In connection with systems using either a motor generator flywheel or the Ilgner system of control, automatic devices have been arranged so that the rate of acceleration is limited and the hoist is automatically retarded independent of the operator, and as these devices are used every time the hoist is operated they are necessarily kept in order. With such arrangements, overwinding or starting up the hoist in the wrong direction is absolutely impossible, and in view of these features the German mining authorities have allowed the rate at which men may be hoisted to be increased from 1,200 feet per minute, which is the maximum with steam-operated hoists with the best safety gears, to 2,000 feet per minute with electric hoists, and the question has been under consideration for some time of increasing this limit to 3,000 feet per minute.

The value of such a feature will be particularly appreciated in connection with deep mines where the time required for getting the men to and from their work is of considerable importance.

One important feature in connection with electrical hoisting plants is their flexibility. In Fig. 50 an electric hoisting plant of the Koepe type with direct-current motor is illustrated. The capacity of this plant is 7,900 pounds at a speed of 3,150 feet per minute, the motor having a maximum rating of 1,250 horsepower.

At some later date, when the mine conditions are such that a greater output is required, a second motor will be added and the load increased to 15,800 pounds. It will be possible to do this without disturbing in any way the operation of the plant, a second generator being added to the Ilgner flywheel set which supplies energy to the hoist to take care of the additional load.

Fig. 51 illustrates a flywheel motor-generator set of the Ilgner type, and it will be noticed that the three-phase driving motor is considerably smaller than the generator, this being due to the fact that the motor is designed for the mean load and the generator must be capable of taking care of the peak loads. In large equipments of this type it is usual to mount the flywheel so that it may be disconnected from the motor generator set. This has the advantage that when only a few trips are required during periods of shut-down, the motor generator can be run alone and the hoist worked at slow speed so that the losses on the generating station are not excessive.

The experience obtained from the operation of the various plants in Europe has shown that electrical hoisting plants, when properly designed, are thoroughly reliable, the maintenance being practically negligible, plants having been in operation for several years, the only renewals required being brushes for the generators. Experience has been such that the operators state that they are not in a position to form any estimate as to the ultimate life of the plant, as indications are that it is indefinite.

(To be continued)

GAS BLOWERS IN COYOTE MINE

Written for Mines and Minerals, by A. A. Galloway, C. E.

The Coyote Mine of the Mexican Coal and Coke Co., Las Esperanzas, Mexico, develops a coal seam ranging from 3 to 6 feet in thickness, dipping 45 degrees from the horizontal,

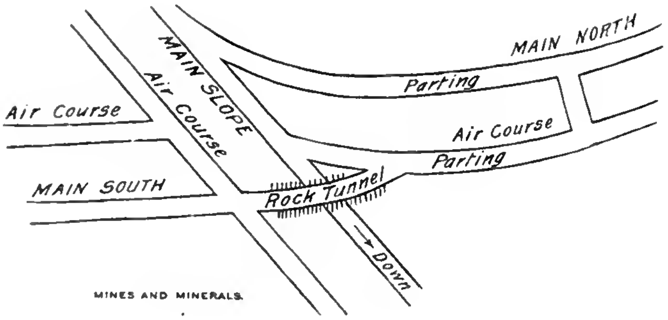


Fig. 1

and containing much gas. It is the same field as the ill-fated Palau Mine which, within the past year, has suffered two disastrous explosions.

Gas occluded in crevices of this coal seam and adjoining strata lets go with tremendous force when the entries approach near the pockets. During April, 1909, the miners working in the third north entry were several times forced to leave the mine because of an unusual quantity of gas suddenly entering the entry. One April morning, about 9 o'clock, gas, which had been occluded in the coal about 70 feet above the main entry broke out of its confinement and converted 40 tons of the intervening coal into slack, with which the entry was completely blocked. The two Mexican miners working in this entry were instantly killed by the force of the released gas. One miner was found stretched along the lower rib, the other in the middle of the track, and both were covered by tons of slack.

The ventilating current of the mine was not split, so that immediately after the blow-out so much gas was liberated that the entire mine was filled to the danger point, necessitating a close-down for that day. The next day all the workings were clear and no more trouble was experienced until the following June, when another pocket was encountered near the face of the main slope. Here ten more lives were snuffed out when the gas broke forth. Being two lifts lower than the third north entry, much more pressure was exerted on the restraining coal than the April release above described, and the increased number of fatalities is explained by the fact that two pairs of levels were being turned off the main slope at this point, hence more miners were within the danger zone.

Coyote Mine is developed by slopes driven on one-half of the pitch of the seam. This method of development permits of a larger trip, greater speed of trip, better track, and greater ease of ingress and egress for the miners, but the greatest advantage is the improvement in landing the trips on the partings, the one-half pitch slope permitting an easy curve and gradual grade changes in landing on the north side. For the south entries the empties are landed on the north partings and then passed to the south side through rock tunnels which pass over the main slope through the rock, reentering the coal seam after having passed the main slope. The method is well shown by the accompanying sketch, Fig. 1.

This method of driving slopes on the half pitch of heavily pitching seams was found so satisfactory that the engineer who had initiated it here introduced it into some Oklahoma mines in which he was interested.

CEMENT CONCRETE VATS AND TANKS

By Albert Moyer, Asso. Am. Soc. C. E.

Impervious, odorless, tasteless, and sanitary vats and tanks can be constructed of reinforced concrete, the reinforcing to be designed by a competent engineer, provided the interior surfaces are treated as follows:

After the forms are removed, grind off with a carborundum stone any projections due to the concrete seeping through the joints between the boards. Keep the surface damp for 2 weeks from the placing of the concrete. Wash the surface thoroughly and allow to dry. Mix up a solution of one part water glass (sodium silicate) 40° B., with four to six parts water, total five to seven parts, according to the density of the concrete surface treated. The denser the surface the weaker should be the solution.

Apply the water-glass solution with a brush. After 4 hours and within 24 hours, wash off the surface with clear water. Again allow the surface to dry. When dry apply another coat of the water-glass solution. After 4 hours and within 24 hours, again wash off the surface with clear water and allow to dry. Repeat this process for three or four coats, which should be sufficient to close up all the pores.

The water glass (sodium silicate) which has penetrated the pores has come in contact with the alkalies in the cement and concrete and formed into an insoluble hard material, causing the surface to become very hard to a depth of ⅛ to ½ inch, according to the density of the concrete. The excess sodium silicate which has remained on the surface, not having come in contact with the alkalies, is soluble, therefore easily washed off with water. The reason for washing off the surface between each coat and allowing the surface to dry, is to obtain a more thorough penetration of the sodium silicate.

TWO-STAGE AIR LOCOMOTIVES

Written for Mines and Minerals

Increased Economy of Two-Stage Compared With Single-Stage Compressed Air Locomotives

Single-expansion compressed-air locomotives are comparatively well known. In this type, air at a pressure of from 600 to 1,000 pounds is stored in a large reservoir from which the air is drawn through a reducing valve set to maintain a pressure of about 150 pounds at the throttle valve. The substitution of a reservoir for a boiler and the use of a reducing valve to regulate the pressure were the only features essentially different from a single-expansion steam locomotive.

From 1873 to 1908, these locomotives were built in gradually increasing numbers for operation in mines where the absence of fire, smoke, sparks, and electric wires was particularly desired. The handicap of a rather poor efficiency, in spite of which the single-expansion compressed-air locomotive has been found the best motive power for a variety of purposes, is to a large extent removed by the two-stage type, which will do from 40 to 60 per cent. more work with the same quantity of air, thus reducing the cost for power about 30 per cent.; reducing the size of the compressors and boilers 30 per cent.; reducing the first cost of installation approximately 15 per cent.; and increasing by 30 per cent. the distance which the locomotive will travel on one charge of air.

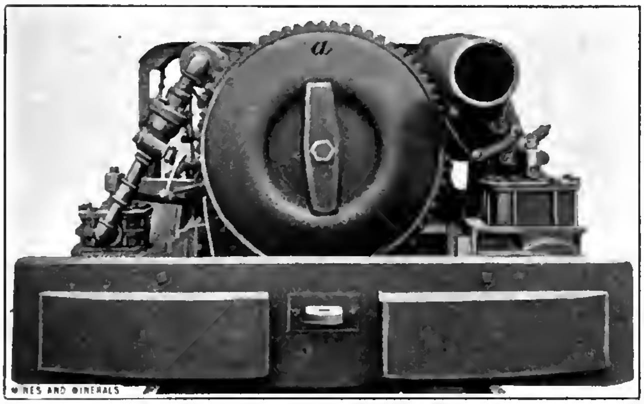


FIG. 1. FRONT OF TWO-STAGE LOCOMOTIVE

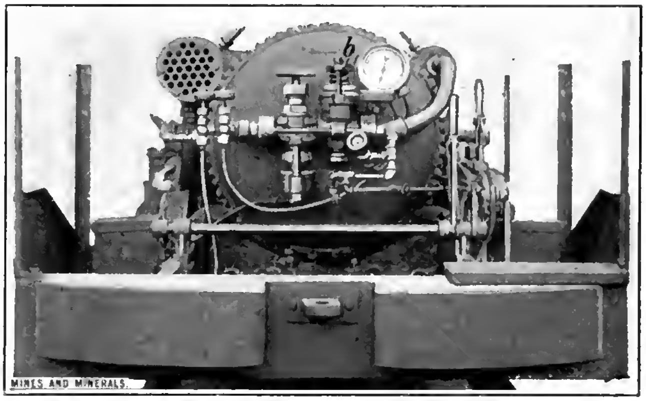


FIG. 2. REAR OF TWO-STAGE LOCOMOTIVE

In the two-stage locomotive the compressed air is expanded in two successive cylinders, providing for a wider range of expansion, with higher initial and lower terminal pressures, thereby securing many of the advantages which compound cylinders give where steam is used, except that in the case of air there can be no cylinder condensation. The effects of a wide range of expansion in a single cylinder, with the resultant great difference in temperature, are, none the less, of serious consequence, causing as they do a temperature of the cylinders and port walls considerably below that of the incoming air, with initial refrigeration and a serious shrinkage in volume. In addition to this, which is the principal reason for compounding with steam, there are various other considerations which render the expansion of compressed air in successive cylinders either vastly more advantageous or an utter failure, depending upon whether advantage is taken of the radically different conditions under which the two fluids operate.

These differences in conditions are due to the steam engine being a primary machine for the conversion of the heat derived from coal or other fuel into work. By application of heat, water or any other fluid may be vaporized, and if confined, will exert a pressure which can move a piston in a cylinder. To obtain the best results, all losses of heat must be avoided, as the pressure and vaporized condition has been caused by heating the water, and when the excess heat above that of the surrounding materials has been lost, the water again becomes inert and incapable of work. The excess heat of the working fluid above that of the surrounding natural objects is, then, the real source of power.

With air the sensible heat of compression and the sensible refrigeration of expansion are unavoidable evils which must be reduced to a minimum. If the heat of compression could be taken away fast enough to keep the air down to its natural temperature during the entire process of compression, and if heat could be supplied during expansion fast enough to prevent any cooling below that of surrounding objects, or if all of the heat of compression could be retained until the air reached the cylinders in which it is to be expanded and do work, compressed air would be 100 per cent. efficient, barring the losses due to friction of the compressing and expanding engines. Unfortunately, both plans are impossible, more particularly in the case of the locomotives. The air must be compressed and expanded so rapidly that there is no time either to extract the heat of compression while the air is in the compressor cylinders, or to prevent refrigeration while expanding in the cylinders of the locomotive; and any heat remaining in the compressed air after it leaves the compressing cylinders is unavoidably lost before it reaches the working cylinders of the locomotive, because of the necessity for storing the air for considerable periods of time in tanks and pipe lines in order that the locomotive may be charged promptly and at various stations frequently spread over an area of several thousand acres.

In compressing air, the first suggestion—compression without heating—has been approximated by performing the operation in successive cylinders of decreasing size, with intercoolers in which the heat due to the preceding compression is removed. The intercooler consists of a casing filled with small tubes through which cold water is circulated. The air passing through the casing and around the tubes is thoroughly cooled before it enters the next cylinder.

This rational, and therefore successful, method of preventing losses has been in general use since 1890, and all compressors for charging locomotives are of either the three-stage or four-stage type, with either two or three intercoolers, reducing the heat losses from 96 per cent. to 17 per cent., based on isothermal compression. With more stages the heat losses could be further reduced, but only at the expense of so much friction and complication due to the multiplication of cylinders and intercoolers that the reduction in heat losses would be more than balanced.

A great saving in the power required to compress air having been effected by intercooling with water at atmospheric temperature, it was only a short step to the reversal of this process—interheating with water after partial expansion in a high-pressure cylinder, before the air entered the low-pressure cylinder. This method of utilizing compressed air for driving pumps has

been adopted with marked success and greatly increased efficiency, as the water which the pump was moving could readily be circulated through the interheater. It was a longer step to the reversal of the process and the substitution of atmospheric air, the only available heating medium which could be drawn on continuously while the locomotive was in motion.

The problem was not satisfactorily solved until 1904, when the H. K. Porter Co. undertook a series of experiments leading up to the atmospheric interheater, with which their two-stage locomotives are equipped.

This appliance also made possible a number of incidental improvements. It provided for a higher ratio of expansion, with higher initial and lower terminal pressures, without unmanageable refrigeration, just as stage compression made possible higher pressures and higher ratio of compression without unmanageable heating.

In a steam engine, because heat is the immediate source of power, the steam should enter the working cylinders at the highest practicable temperature, be exhausted at the lowest, and all losses due to radiation or convection guarded against, in order that the largest possible percentage of heat may be converted into work. In a compressed-air locomotive plant, because the heat of compression is lost before the compressed air can enter the working cylinders of the locomotive, the best results are obtained by the closest possible approximation to compression without heating, and to expansion without cooling, thereby reducing to a minimum the extra work of compression due to heating, and the work lost due to the cooling of the air during expansion.

Because the performance of the compound steam locomotive has been thoroughly tested by many competent investigators, and the increased efficiency, as compared with the single-expansion locomotive, has been established at about 20 per cent., and as it has been previously stated that the two-stage compressed-air locomotive using many of the same appliances will be from 40 per cent. to 60 per cent. more efficient than a single-expansion compressed-air locomotive, a brief statement in parallel columns is given of the differences which justify another name for a similar apparatus, and in explanation of the greater gain in efficiency.

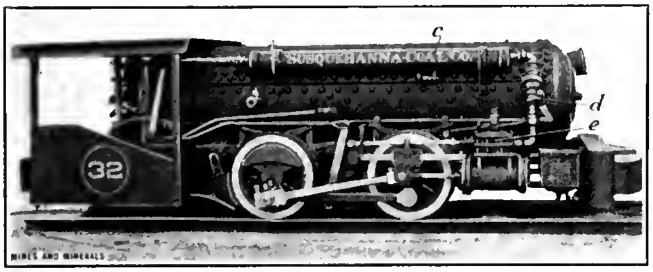


Fig. 3. Right-Hand Side of Two-Stage Air Locomotive

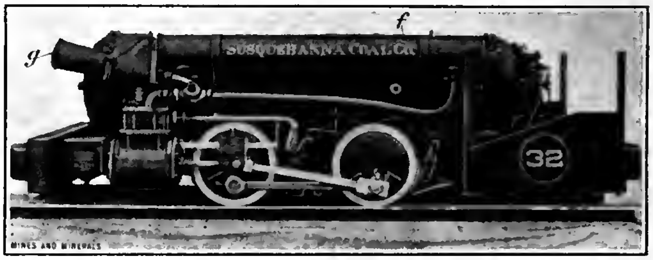


Fig. 4. Left-Hand Side of Two-Stage Air Locomotive

HIGHER PRESSURES TO OBTAIN THE ADVANTAGES OF COMPOUNDING

TWO-STAGE COMPRESSED-AIR LOCOMOTIVE

In order to store compressed air in sufficient quantities to avoid too frequent charging of the locomotive, the air must be compressed to about one-sixtieth of its normal volume and to 800 or 900-pounds pressure. This fundamental requisite provides for a higher pressure in the valve chest of the high-pressure cylinder without additional cost or other compensating disadvantages to obtain it.

COMPOUND STEAM LOCOMOTIVE

Higher steam pressures require a heavier and more expensive boiler; more coal to evaporate a given amount of water into steam at the higher pressure and temperature; more expense for repairs to keep the boiler in working order. The higher temperature of steam at a higher pressure requires better and more expensive lagging for boiler and cylinders to prevent undue losses due to radiation, and better and more expensive packing and lubricants.

REHEATING BETWEEN HIGH- AND LOW-PRESSURE CYLINDERS

The compressed air stored in the main reservoir is at atmospheric temperature because the heat of compression is necessarily lost before the air reaches the locomotive reservoir. For this reason the loss of heat due to the partial expansion of the air in the high-pressure cylinder carries it far below the temperature of the surrounding atmosphere (usually about 140° F. below), thereby rendering the atmospheric air an efficient and inexhaustible heating medium which costs nothing.

The steam in a locomotive boiler is at a temperature of from 350° F. to 375° F., and even after partial expansion in the high-pressure cylinder, is still at a temperature far above that of the atmosphere. It can only be reheated by burning more coal—and coal costs money.

EFFECT OF INITIAL TEMPERATURES

Compressed air enters the high-pressure cylinder at approximately atmospheric temperature, or below, and in expanding and doing work, becomes greatly refrigerated. If a high-grade ratio of expansion with high initial and low terminal pressures is attempted without restoring the lost heat while the operation is going on, refrigeration becomes so great that proper lubrication is impossible. This lost heat can only be properly restored by dividing the work into two stages and expanding the air in successive cylinders with a properly designed interheater between them. A high ratio of expansion and the resultant economies therefrom are therefore effectively possible in connection with two-stage or multiple-expansion cylinders only

Steam enters the high-pressure cylinder at a relatively high temperature, and as it loses heat due to expansion and work done, becomes cooler. But it is never cold enough to render lubrication more difficult at the lower temperature than the higher. Steam is expanded in two or more successive cylinders in order to reduce the range of temperature in any one cylinder, and to prevent cylinder condensation. There is no other reason why a compound or triple-expansion steam engine should be more economical than an engine in which the same ratio of expansion is obtained in a single cylinder.

The front of a two-stage compressed-air locomotive is shown in Fig. 1, the rear in Fig. 2, and both sides in Figs. 3 and 4. The supply of air is carried in the large cylindrical tank *a*, Fig. 1, which, when first charged, contains air at a pressure of about 800 pounds per square inch. From this tank

air is drawn through a reducing valve b, Fig. 2, which is adjusted to maintain a pressure of about 250 pounds in the small auxiliary reservoir c, Fig. 3. From this auxiliary reservoir the air passes through a throttle valve d, Fig. 3, located in a pipe leading from the front end of the auxiliary reservoir to the valve chest e, Fig. 3, of the high-pressure cylinder. The exhaust from the high-pressure cylinder is piped to the front end of the atmospheric interheater f, Fig. 4. From the rear end of the inter-

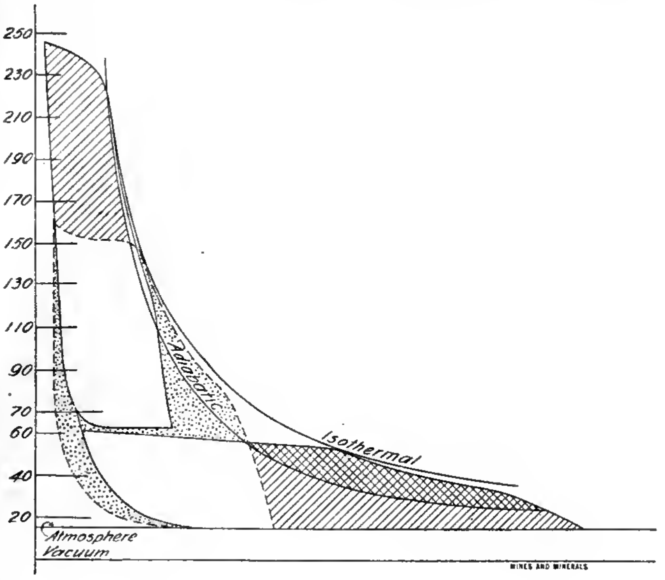


FIG. 5

heater another pipe leads to the valve chest of the low-pressure cylinder (on the left side of the locomotive). From the exhaust port of the low-pressure cylinder, still another pipe leads to an exhaust nozzle, located in and concentric with a slanting locomotive draught stack g, Fig. 4. For ordinary working conditions a cylinder ratio of 4 to 1 has been found most advantageous. During the partial expansion in the high-pressure cylinder, from 250 pounds down to 50 pounds, the temperature of the air ordinarily drops about 140° F., or from about 60 degrees above to 80 degrees below zero, so that on leaving the high-pressure cylinder the volume of the air is only $\frac{380 \times 100}{520} = 73$ per cent. as great as it would be if no refrigeration has occurred in the high-pressure cylinder. Without the interheater the volume of the low-pressure cylinder would only be $4 \times .73 = 2.92$ times the volume of the high-pressure cylinder if the initial pressure of 50 pounds per square inch in the low-pressure cylinder is to be maintained. The atmospheric interheating between the high- and low-pressure cylinders expands the air in its passage to make it fill a cylinder $4 \div 2.92 = 1.36$, or 36 per cent. larger than it could otherwise fill with air at a pressure of 50 pounds per square inch, and in so doing adds 36 per cent. to one-half of the work done by the air, or 18 per cent. to the total amount of work done by the air, due to the interheating alone.

The interheater consists of a cylindrical casing filled with small tubes. The partly expanded and refrigerated air on its way from the high-pressure to the low-pressure cylinder is passed through the casing and around and between the small tubes, while a violent current of relatively warm atmospheric air is being drawn through the small tubes by the partial vacuum created in the base of a slanting draught stack by the ejector action of the exhaust from the low-pressure cylinder.

With an interheater of the size shown in Fig. 4, practically all of the heat is restored and the air enters the low-pressure cylinder at a temperature within about 15 degrees of that of the surrounding atmosphere. The temperature of the surrounding atmosphere has no appreciable effect upon the relative economy of this type as compared with the single-expansion locomotive, because the air in the main tank is always at approximately atmospheric temperature, however low that temperature may be. The partly expanded air from the high-pressure cylinder is therefore always relatively cold as compared with atmospheric air which renders the latter, under all conditions, an efficient heating medium.

The preceding varied explanations of the cause and effect of the fluctuating temperatures of the compressed and expanding air in the operation of compressed-air locomotives are intended partly for the lay reader who ordinarily thinks that pressure does the work in steam or air engines and forgets that the pressure is derived from heat, and partly to assist the engineer in adjusting his thoughts to fit a rather unusual set of conditions.

Something over one hundred two-stage compressed-air locomotives have been built during the past 2 years, and their superior efficiency, desirable simplicity, and reliability, thoroughly demonstrated. In coal and metal mines the two-stage locomotive has been operated in comparison with single-expansion locomotives, and by the simple test of having first one locomotive and then the other haul the same train over the same piece of track, the added endurance on one charge of air and the superior efficiency of the two-stage machine has been established.

Operators who have been using single-expansion compressed-air locomotives for 15 years are having their single-expansion locomotives converted into two-stage machines, at an expense far in excess of the difference in the first cost of the two types, because there is a continuous saving in operating expenses, and because the first cost of converting the locomotives is less than the expense of additional compressors and boilers to provide for increasing tonnage and longer haul.

The results of two of the simple tests mentioned above, together with combined indicator diagrams, Fig. 5, of the two-stage with superimposed diagram of the single-expansion locomotive, are given in order to show the actual results which have been obtained. Both tests were made in the same way and without special preparations. Many others could be cited, but these two are sufficient when taken in conjunction with the manifest possibilities of the apparatus and the indicator diagrams. The two long curved lines are the isothermal and the adiabatic. The isothermal curve shows the relations of pressure and volume if the air could be expanded without any reduction in temperature; the adiabatic curve shows the same relations if the air were expanded without obtaining heat from any source except itself. The double-cross-hatched area is the gain due to the interheater alone. The single-cross-hatched

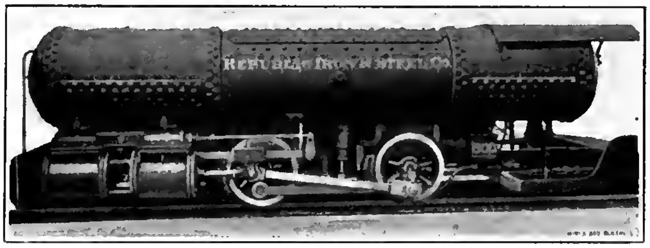


FIG. 6. COMPRESSED AIR LOCOMOTIVE, WEIGHT 34,000 POUNDS

area plus the double-cross-hatched area, minus the stippled area, equals the total gain due to the higher initial and lower terminal pressures, with a wider range of expansion in two successive cylinders, only possible in conjunction with the use of an interheater to prevent unmanageable low temperatures in the low-pressure cylinder.

An unavoidable loss occurs in the use of compressed-air locomotives, due to the high pressure to which the air must be compressed in order to reduce the bulk of the storage tanks

on the locomotive and to enable the locomotive to maintain its maximum tractive force for relatively long distances. In practice, the air is usually compressed to 1,000 pounds per square inch for storage in the stationary reservoir from which the locomotive draws its supply. The locomotive is usually charged to a pressure of 800 pounds per square inch. In the newer, two-stage type this pressure is reduced to 250 pounds before it enters the high-pressure cylinder; in the older, single-expansion locomotive it was reduced to 150 or 160 pounds for use in the cylinders. The consequent losses are not in direct proportion to this reduction in pressure, as might be inferred if the subject were given only superficial attention. To compress 100 cubic feet of free air in two stages to 150 pounds pressure requires 19 horsepower; 100 cubic feet in two stages to 250 pounds, 22½ horsepower; 100 cubic feet in three or four stages to 1,000 pounds, about 33 horsepower. There is, therefore, a loss of only one-third of the total power in reducing from 1,000 to 250 pounds, although the gauge pressure is reduced three-fourths. The reason for this is apparent when we consider that as the pressure increases the volume diminishes. In a three-stage compressor designed to compress the air to 1,000 pounds, the second-stage cylinder has about one-fourth of the volume of the intake cylinder, while the average pressure is four times as great, which makes the work in the second stage the same as in the first. In the third and final stage the average pressure is 16 times as great and the volume one-sixteenth of the intake cylinder, and the total work is again the same; so that only one-third of the total work done in compressing the air to 1,000 pounds pressure is thrown away in reducing the pressure from 1,000 to 250 pounds. It is this feature of air compression which makes the compressed-air locomotive a commercial success.

In addition to Figs. 3 and 4 which show the small compressed-air locomotive used by the Susquehanna Coal Co., Fig. 6 shows a 34,000-pound locomotive built for main haulage in the coal mine of the Republic Coke Works, Republic Iron and Steel Co.

Trials for Comparative Air Consumption.—The following method of making trials between single-expansion and two-stage compressed-air locomotives was employed in all cases: The same train was hauled by the single-expansion and two-stage locomotives the same distance over the same piece of track, with the same operator, giving the locomotives as nearly as possible the same work to do under the same conditions. In all cases the trains were started from a given point and were allowed to come to rest as near as possible to another given point without the use of brakes. Air consumed was determined by the difference in pressure recorded at the beginning and end of the trip by the gauge on the main reservoir. Running conditions were the same as would obtain in the regular hauling of coal in the mine.

Trial 1 was conducted at the Susquehanna Coal Co.'s No. 7 colliery, Nanticoke, Pa., in the presence of Mr. McMahon, Chief Engineer Susquehanna Coal Co., and Mr. C. B. Hodges, of the H. K. Porter Co.

Trial 1. Locomotive Data

	Single Expansion	Two-Stage
Weight	10,000 pounds	10,600 pounds
Cylinders, high-pressure, diameter	6 inches	5 inches
Cylinders, low-pressure, diameter		10 inches
Cylinders, stroke	10 inches	10 inches
Driving wheels, number and diameter	4—23 inches	4—23 inches
Working pressure, high-pressure cylinder	150 pounds	250 pounds
Maximum charging pressure	900 pounds	900 pounds
Capacity of main reservoir	41 cu. ft.	41 cu. ft.
Age of locomotive	8 months	1 month

Train and Road Data

Length of trial run, feet	2,200
Average grade, per cent	.96
Maximum grade, per cent	2.00
Track gauge, inches	42

Train consisted of loaded coal cars weighing about 9,500 pounds each.

REMARKS

The excessively high efficiency indicated on lines Nos. 4 and 6, hauling four and five cars down grade, may be due to the fact that considerably more air is wasted in the single-expansion locomotive when giving the train just a little assistance when the grade is nearly steep enough to cause it to run down by itself.

Lines Nos. 7 and 8 show additional trips up grade with the two-stage with five-car trips. The trip with the two-stage (line 5) was made with the reverse lever "in the corner," using the air full stroke all the way. The superior efficiency achieved on trips shown on lines Nos. 7 and 8 was the result of using the air as expansively as possible.

Taking the total pressure reduction of the three round trips with the single-expansion locomotive and two-stage locomotive (lines 1, 2, 3, 4, 5, and 6), we find that the two-stage locomotive used $\frac{1,245}{2,225}=56$ per cent. of the air used by the single-expansion engine under exactly the same conditions of service, this average per cent. for the entire test showing a saving of 44 per cent. of the air used by the single-expansion machine.

In making the test every care was used to obtain reliable results. The pressure gauge on the two-stage locomotive was removed and placed on the single-expansion locomotive during the test of this locomotive, and then shifted back to the two-stage for testing it. This gauge was a comparatively new one, and presumably correct, and if any error did exist, it would have been the same for both locomotives. The tanks were exactly of the same capacity. The same engineer operated both locomotives alternately during the trials.

Log of Runs

With Pressure Reductions and Percentage Used by Two-Stage Locomotive

Line	Single Expansion					Two-Stage					Percentage Used By Two-Stage	
	Number Cars	Gauge Pressure		Pressure Reduction	Time	Number Cars	Gauge Pressure		Pressure Reduction	Time		
		Start	Stop				Start	Stop				
1	3	690	280	410	3:15	3	680	400	280		68.3	Up grade
2	3	280	140	140		3	400	320	80		57.1	Down grade
3	4	770	180	590	3:30	4	570	250	320	2:40	54.2	Up grade
4	4	760	540	220		4	250	175	75		34.1	Down grade
5	5	805	200	605	2:55	5	710	290	420	2:50	69.5	Up grade
6	5	730	470	260		5	290	220	70		26.9	Down grade
				2,252					1,245			
	Total pressure reduction 2,225 pounds—3 round trips					Total pressure reduction 1,245 pounds—3 round trips						
7						5	610	220	390			Up grade
8						5	735	360	375	3:15		Up grade

Trial 2 was conducted at Orient Mine of the Orient Coke Co., Orient, Fayette County, Pennsylvania, in the presence of Mr. Chas. Opperman, of the Orient Coke Co.; Mr. G. E. Huttelmaier, of the H. C. Frick Coke Co.; Mr. C. B. Hodges, of the H. K. Porter Co.

TRIAL 2. LOCOMOTIVE DATA

	Single Expansion	Two-Stage
Weight	9,600 pounds	10,500 pounds
Cylinders, high-pressure, diameter	6 inches	5½ inches
Cylinders, low-pressure, diameter		11 inches
Cylinders, stroke	10 inches	10 inches
Driving wheels, number and diameter	4-23 inches	4-23 inches
Working pressure, high-pressure cylinder	150 pounds	250 pounds
Maximum charging pressure	800 pounds	800 pounds
Storage tanks	1	1
Capacity of main reservoir	40.26 cu. ft.	40.26 cu. ft.
Age of locomotive	6 months	2 months

TRAIN AND ROAD DATA

Length of trial run, feet	2,500
Average grade, per cent	.52
Track gauge, inches	44

In this run there was a reverse curve in a chute leading from one heading to a parallel heading. The train consisted of four loaded wagons, each about 7,000 pounds; and six empty wagons, each about 2,200 pounds.

LOG OF TRIAL RUNS

No. of Run	Type of Locomotive	Tank Pressures At Start	At Finish	Am't of Drop	Time (P. M.) Start	Finish	Elapsed
1	Single expansion	705	265	440	7:23½	7:27½	.04
2	Two-stage	740	420	320	8:06½	8:12	.05½
3	Two-stage	685	385	300	8:48	8:52¾	.04¾

No. 1 run: Very satisfactory.
No. 2 run: Very irregular; operator not so familiar with two-stage machine, hence decided to rerun.
No. 3 run: Much better and smoother than second.

DEDUCTIONS
Calculated by Mr. G. E. Huttelmaier, H. C. Frick Coke Co.

	Trial No. 1	Trial No. 2
Free air consumed, cubic feet	1,206.92	877.76
Drawbar effort {Loco. 97.92 / Trip 832.24}	930.16	{107.10 / 832.24} 939.34
Total work performed in ft.-lbs	2,325,400.00	2,348,350.00
Ft.-lbs performed per minute	581,350.00	426,973.00
Average speed {Feet per minute	625.00	454.50
Average speed Miles per hour}	7.10	5.17
Average horsepower developed	17.61	12.94
Ft.-lb. work per cu. ft. free air	1,926.00	2,676.00
Free air consumed per minute	301.70	159.59
Air per minute per horsepower	17.13	12.33
Percentage of air consumed as compared with trial No. 1	100 per cent.	72 per cent.
Amount of work per unit of air compared to trial No. 1	100 per cent.	1.38 per cent.
Saving of air effected over trial No. 1		28 per cent.

EXPLOSIBILITY OF COAL DUST

Geological Survey Bulletin 425, on the "Explosibility of Coal Dust," can be had by writing to the Bureau of Mines, Washington, D. C. It is by George C. Rice, with chapters by J. C. W. Frazer, Axel Larsen, Frank Haas, and Carl Scholz.

Mr. Rice reviews the experiments into the explosibility of coal dust in foreign countries and dwells at considerable length upon the attitude taken in France by the engineers, who, until the great disaster at Courrieres in 1906, which cost 1,000 lives, did not believe that coal dust would explode without the presence of firedamp. Since that terrible disaster a testing station has been established in France and now the French engineers are fully convinced of the dangers of coal dust.

"The coal-dust question in this country," continues Mr. Rice, "cannot be said to have awakened widespread interest among mining men until the terrible disasters of December, 1907, which resulted in the death of 648 men. In response to a demand by those interested in coal mining throughout the country, Congress, in 1908, made an appropriation for the investigation of mine explosions. The United States Geological Survey was charged with the investigation. A testing station was at once decided upon and was established at Pittsburg, Pa.

"While it is probable that for several years the leading mining men in the country have believed in the explosibility of coal dust without the presence of firedamp, yet until the public demonstrations were given at the testing station at Pittsburg, during 1908-09, and reports were received of similar tests made abroad, a large proportion disbelieved. These tests were so convincing to those who saw them, and such general publicity has been given to them, that it is now exceptional to find a mining man who does not accept the evidence of the explosibility of coal dust. The question of the day no longer is 'Will coal dust explode?' but 'What is the best method of preventing coal-dust explosions?'"

The following are some of the tentative conclusions of Mr. Rice on the dust problem:

"That coal dust will explode under some circumstances, both in the presence of firedamp and without it, is now generally accepted by mining men. The writer fully agrees with this and takes the following views of the explosibility of dust and the conditions necessary for explosion.

"Experiments at Pittsburg indicate that under ordinary conditions the dust must be from coal having at least about 10 per cent. of volatile combustible matter, though in certain foreign experiments it is claimed explosions were obtained with charcoal dust.

"Dusts with higher percentages of volatile combustible matter are more sensitive, ash, moisture contents, and size, being constant. This view is based partly on the preliminary experiments at Pittsburg and on the results of experiments of Mr. Taffanel and other foreign investigators. Where there is a large amount of dry coal dust, judging from the Pittsburg experiments, a humid atmosphere has little effect on ignition of dust or propagation of an explosion. A long continuance of the humid conditions renders the coal dust moist and inert, but the presence of moisture in the air at the moment of explosion is not sufficient to prevent an explosion; that is, not enough moisture is carried by the mine air to reduce materially the temperature of the flame. Fully saturated vapor at 65° F., an ordinary mine temperature in this country, weighs 6.78 grains per cubic foot (15.5 grams per cubic meter). Coal dust suspended in such a saturated atmosphere in a cloud of moderate density weighs, say, 200 grams per cubic meter. At the figures given, the weight of vapor is but 7.8 per cent. of the weight of dust. The Pittsburg experiments with wetted dust showed that several times this percentage of moisture in the dust, in addition to a nearly saturated atmosphere, was required to prevent propagation.

"Probably with a low dust density, the relative humidity of the air would be an important factor in tending to prevent the initiation of an explosion. However, the great purpose of artificially humidifying mine air is that it may serve as a vehicle for carrying water to the dust."

Mr. Rice concludes by reviewing the various remedies that are offered for the coal-dust problem, giving the good and bad points of each.

The world's production of calcium carbide in 1908 amounted to 200,000 tons, one-fifth of which was produced in the United States and Canada. The production in the other countries was as follows, in tons: Italy, 32,002; France, 27,000; Norway, 25,000; Switzerland and Austria-Hungary, 20,000 each; Sweden, 12,000; Germany, 9,000; England, 800. There were 70 factories employed in the manufacture of the article. The production has increased very much in recent years.

THE MINE LAW OF WEST VIRGINIA*

By P. A. Grady, Mine Inspector, Twelfth District of West Virginia

The purpose for which this paper is written is to show that the mine operators who are the most successful producers of coal in the state are the ones who do their utmost to observe every requirement of the mine law. They know that when their operations are in such shape that coal can be produced in the most successful and economical way, year in and year out, they are exceeding all the most important requirements made of them by the mine law.

Those most Successful in the Production of Coal Do Even Better than the Law Requires

Some of you have no doubt heard the statement made some time or another, by some so-called mining man, that he was put to an unnecessary expense by having to put his mine in such shape as to comply with the mine law, and that he considered such obligatory acts as unreasonable and a hardship upon the mining industry of the state. That such views are not entertained by many is attested by the great number who, in their zeal, have striven to exceed any lawful requirements that we can put on our statute books.

Among the very first laws on our statute books is the one calling for maps to be furnished every 6 months to the district mine inspector. Before the present requirement became a law, the writer, in glancing through some of the reports of the mine department, noticed in a few places where the maps of certain mines had not been extended for a period of a year and a half, and sometimes two years. No practical up-to-date manager could go for so long a time without having the workings extended on his maps, to show how the work was progressing under his charge.

How much different from this is the practice at our best mines, where the maps are extended quarterly and the inspector, instead of getting a copy every 6 months, can have it every 3 months if he so desires.

Some difficulty has been met with by the mine department in getting the second opening of some few mines in a condition so as to comply with the mine law. In such cases this second opening is generally used as an airway to the fan, and from the way it is blocked with roof falls and standing water, you would wonder how any circulation is maintained through it all.

Directly opposed to such a practice are the ones who have not only a second, but a third and fourth opening, kept in a clean condition for the men and mules to travel in from the interior of the mine. This third and fourth entry serves another excellent purpose in reducing the power that has to be applied to the ventilating fan.

One of the requirements in our state is to furnish no less than 100 cubic feet of air per minute for each and every person employed in a mine, and if it were not also required that there be as much more air furnished as the district mine inspector might require, that law would be considered farcical by those who have well-ventilated mines; for the standard they maintain requires at least 10 times as much as is required by law. By furnishing the workmen a plentiful supply of air they know that they can obtain more work than they could in a poorly ventilated mine.

To conduct this air-current the successful operator constructs permanent stoppings out of concrete, brick, or stone, and he prevents leakage of the air-current to such an extent that he has over 90 per cent. of the total quantity circulating around the head of the workings. Wherever he has an entry or a section of the mine that he thinks will outlive the life of a temporary stopping, he will have one constructed that he knows will be there until the mine is worked out.

The make-shift coal operator wants to construct his stoppings in such a way that you would think they were only supposed to last for a few months, and in the end he spends much more money in reconstructing them than would have been required to construct suitable ones in the first place. The amount of ventilation conducted around the head of the workings with such stoppings is only about 40 per cent. of the total volume.

In ventilating 11 mines which one of the largest coal companies in this state is operating, 1,285 horsepower in electric energy is expended in running the fans. The ventilating fans at these mines are of the best make, and every care is taken, by constructing the best stoppings, to prevent any loss of the air-current before it reaches the innermost workings. All the material necessary to conduct the air is furnished, with the result that they have nearly the whole volume of air in circulation around the working places.

If this company had constructed the poor stoppings mentioned above, the amount of electric energy expended on its fans in air wasted, would be about 60 per cent. of the total amount, or 771 horsepower, and the cost of generating this amount is $74,000 per year. This only represents one of the losses there is in constructing poor stoppings.

A state mine inspector, at a meeting of the Mine Inspectors' Institute, held in Chicago last June, was heard to remark that he was unable to get the operators of the mines in his state to make a proper division of the air-current, and to get away from the continuous-current system in ventilating their mines. If he had presented figures to them showing that an overcast, costing about $150 to construct, would do away with at least one trap door, with its attendant cost of about $20 per month, their knowledge of the amount to be saved in a few months would have made them decide to construct overcasts.

The successful coal operator of this state will not have a trap door on his main entries, but will split the air as much as the total quantity and the extent of his coal property will allow.

Mines generating firedamp or other dangerous gases are required by law to be examined by a competent fire boss before the men enter for work. The practice at some of our best mines is to have the workings examined before work commences, whether they generate dangerous gases or not. Such an examination is made for the purpose of discovering any loose overhanging top, electric wires that may be down, dusty working places in which shots should not be fired until made wet, and any disarrangement of the air-current that may have taken place.

Those who have such examinations conducted claim that it is worth much more than the amount of money paid for it.

Where gas is being generated in dangerous quantities, it is unlawful to use any mechanical power for ventilating, other than a fan. Whether the mine generates gas or not, there is no reason why it should be ventilated with an antiquated furnace, which will, in the labor expended on it, and the fuel burnt, cost more in 6 months than would pay for a fan that would give seven times the amount of air such a furnace will give. Surely the workmen in such mines could perform more labor and give better returns if furnished with a plentiful supply of air.

The law requiring two safety lamps to be kept on hand at every mine in the state, whether it generates explosive gas or not, is a good one The purpose of the law is to have such lamps on hand should occasion demand their use unexpectedly.

The operator who thinks he is complying with the law when he has two obsolete Davy lamps hung up in the blacksmith shop at the mine, in such condition that it would take weeks to get them in shape to be used should anything occur, is fooling himself very much.

Sir Humphrey Davy's invention for a safety lamp served the purpose well for a time, but today it should be replaced by the more up-to-date safety lamps, with which the leading operators have provided themselves, and which will stand the tests which they must undergo.

* Paper read at the Wheeling meeting of West Virginia Coal Mining Institute, December 6, 7, and 8, 1910.

When the law went into effect requiring mine foremen to measure the air-current of the mine, an educational step was taken which resulted somewhat in the betterment of the men engaged in such work.

When the Mine Department of West Virginia, with the cooperation of the leading operators of the state, inaugurated a system of examining mine foremen and fire bosses, and granting them certificates if found competent, a far greater educational step was taken; for it has fostered in the minds of such men a desire to study and learn more of their calling.

The best safeguard we can erect against mining accidents does not lie in excessive legislation, but in better training of miners and mine officials, which will produce a higher degree of intelligence and a more thorough state of discipline among them.

The encouragement given by the Y. M. C. A. to educational work among the miners in this state is to be heartily commended; and that it is appreciated is evidenced by the many reading rooms established at mines where the men can read and study and indulge in other wholesome pastimes.

Our leading operators in the state today not only provide the instruments required by law at the mines, but, to show that they are keenly alive to know conditions that may exist within the mines; they have hygrometers with which to determine the relative humidity of the atmosphere of the mine, and they have air samples taken in various places in the mine and analyzed, which will reveal to them any generation of explosive gas long before it can be detected with an ordinary safety lamp. Not a few have provided themselves with breathing apparatus which will, with the men trained in their use, prove invaluable in event of a fire or an explosion in their mines.

At some of our well-regulated mines, in addition to keeping stretchers and blankets, there is a first-aid hospital equipped, and the leading workmen are trained in rendering first aid to those who may be injured.

The impetus given to this humane movement by Dr. M. J. Shields, of the United States Army Medical Corps, is to be heartily commended by every mining man in the state. It is to be hoped that in the near future there will be organized classes at every mine under training in this work, which will prevent much unnecessary suffering, and sometimes death.

A system of checking the men in and out of the mine, and keeping a record of their names and the section in which they work, is in force at some of the best regulated mines. In case of a fire or mine explosion, it will prevent much confusion and uncertainty; for by means of it they will be able to tell when all the men are out of the mine, or show the sections in which they should be looked for.

Viewing this system from an economic standpoint, it reveals by a glance at the checking board on the outside any shortage of labor, such as a driver or trackman being off, which can be filled without having to chase all over the mine to discover it.

About one-half of the coal mined in the state is undercut with machines. The use of undercutting machines has a tendency to make the mining much safer; it does away with solid shooting, and does not make it more dangerous, as you sometimes hear some poorly-enlightened radicalist assert. Where the seam of coal undercut is thick, or has a heavy parting in it that would make the shooting dangerous, in so far as the ignition of coal dust is concerned, by firing too heavy charges of powder to bring it down, the best method, which is being practiced at some mines, is to shoot it down in two benches, using light charges of powder.

Regulations governing the shooting of coal so as to make it more safe have been put into effect by the Mine Department of the state. Those regulations, first of all, call for the adoption of one of the permissible explosives in use, or black powder must be used in such a way that the least possible danger will result from it.

To bring about such a result, it was required that shot bosses be employed, who would see that the coal was undercut, holes properly placed, the proper charge of powder used, and incombustible material used in tamping, and that the surrounding area is in a damp condition.

That these regulations have been proven to be valuable is shown by the results obtained, and if you would go to mines where the employment of shot bosses is required and ask the management their opinion of the requirement, they will no doubt tell you that under no consideration would they be willing to relinquish it; for as a result, they are securing a grade of coal that is not blown or shot into slack and dust as it was formerly, and the discipline obtained from the enforcement of such regulations tends toward the betterment of the mine in every way.

In conclusion the writer wishes to state that what has been outlined in this paper are observations made by himself, and he does not lay claim to knowing what the existing conditions are at every mine in the state. The assertion is made, however, that at the mines where the most money is being made by both the operator and workman, there you can find the mine law being complied with, and other safety precautions adopted which, if they had been embodied in our mine law a few years ago, would have been considered very radical indeed.

The practice obtaining at some of the mines in this state to protect the health and lives of persons working therein, and for the economical production of coal, would be a good guide to follow in drafting new laws that our legislative body might deem necessary to enact.

That there are many mines in this state whose condition and practice are better than what has been outlined in this paper, cannot be doubted. The Mine Department force of the state is doing its best to improve the methods at those mines that are not considered as good. To describe them all within the confines of this paper would be impossible.

THE ORGAN FEEDWATER HEATER

Written for Mines and Minerals

Originally feedwater heaters were intended to save fuel by using the exhaust from a steam engine to preheat the boiler feedwater. In ordinary practice there is a saving of .0909 per cent. for each degree of heat added to the feedwater.*

The saving for an assumed case where the boiler pressure is 100 pounds gauge, the original temperature of the feedwater 60° F., and its final temperature before entering the boiler 200° F., is calculated as follows: The total heat in steam, corresponding to a temperature due to 100 pounds gauge pressure, above 32° F., is 1,185 British thermal units.* The increase in heat units above 32° F., in the feedwater at 60° F. is 28, therefore the heat units in the steam above the original temperature of the feedwater are $1,185-28=1,157$. The increase in heat units in the feedwater heater is $200-60=140$ and the saving effected by the heater would be

$$\frac{140\times 100}{1,157}=12.1 \text{ per cent.}$$

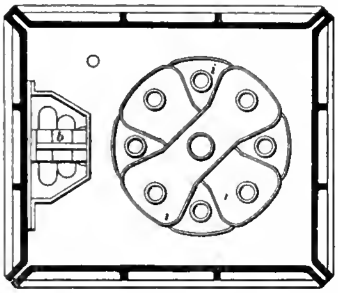
After the feedwater heater had been in use some time it was ascertained that where clear water was used for feeding the boiler, considerable sediment was precipitated. This was due to the change in temperature, and as this sediment was kept out of the boilers the name "purifier" was added to that of feedwater heater. Various experiments were next made to ascertain if it were possible, by means of feedwater heaters, to eliminate from feedwater chemical substances held in solution, as scale-forming chemical compounds were more injurious to boilers than sediments which could be blown off.

The Organ feedwater heater and purifier, shown in section, Fig. 1, is one of the results of such experiments and belongs to

* Kent's Mechanical Pocketbook.

the "open feedwater heater" class to distinguish it from the closed feedwater heaters in which the exhaust steam does not come in contact with the feedwater.

The construction of the heater is such as to interest engineers and steam users. In the Organ heater and purifier the exhaust steam enters the heater through the pipe *a* of the same size as the engine exhaust. The valve and cylinder oil used in steam engines gradually becomes vaporized, in which condition it travels from the cylinder with the exhaust steam. In order to economize in oil, also to keep oil from going into the feedwater, oil separators are used inside or outside the feedwater heaters of this type. As the steam enters the Organ heater



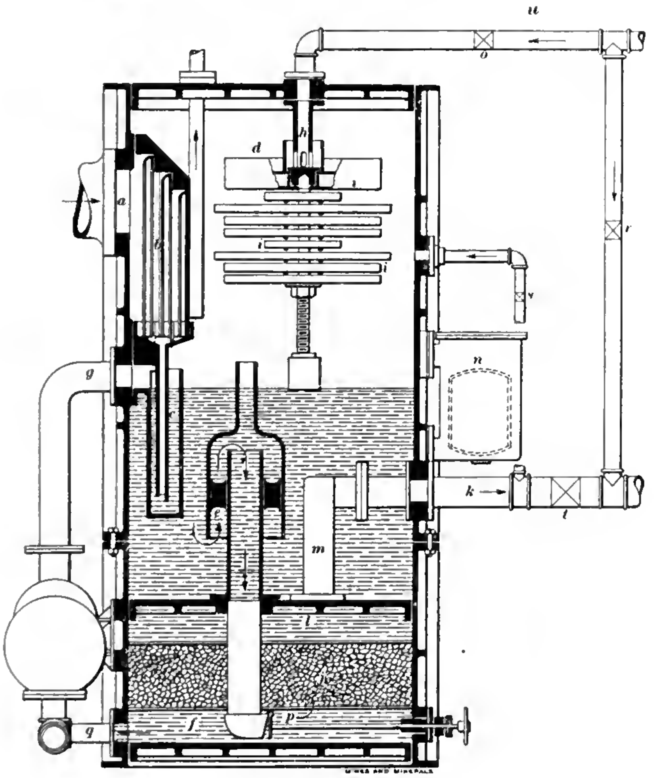


FIG. 1. THE ORGAN FEEDWATER HEATER

it impinges on the plates *b* whose edges are turned in to form channels. The oil held in mechanical suspension by the steam is deposited on the plates and flows down into the oil trap *c*, while the steam passes out to the heating chamber *d*.

Although this is an efficient arrangement it is augmented by a second trap or hood *e* which prevents any oil in the heating chambers going downwards. The water which accumulates in chamber *d* flows up through the trap *e* and then down a central pipe to the settling chamber *f*. Any oil that accumulates in the heating chamber will on account of its specific gravity float and be carried through the overflow *g* which acts as a skimmer. The feedwater enters the heating chamber through pipe *h* and enters the upper of a series of Maltese cross-shaped pans *i*, below the water level in the pan. The pans are arranged rosette fashion, as shown in Fig. 2, so that the water flowing over their rims falls on the pan below, and from the last pan to the tank.

The shape of the pans furnishes a large edge for the water to trickle over, thus bringing it as a thin sheet in contact with the exhaust steam and allowing it to absorb the heat quickly. This is particularly important when the engine is working under minimum load, and the boilers are worked to full capacity supplying live steam to the heating and drying systems. The amount of steam condensed and returned as pure water to the boilers is an important point to consider in the efficiency of any heater.

There is some settling of sediment in chamber *d* which communicates with lower settling chamber *f* by means of the hood and pipe *e*. The object of connecting the two chambers in this manner is to eliminate any disturbances that might prevent settlement of sediment. The water at this stage varies from 200° to 212° F. and much of the soluble salts and all sediment are deposited; thus, the scale-forming material is taken out before the water enters the filter *j*.

It will be noticed that filtering material is packed between two perforated plates and that the water is obliged to move upwards in order to reach the tail-pipe *k* of the feed-pump. From the top of the filtered heater chamber *l*, there are two uptakes *m*, which are in the water where its temperature is highest, and thus the heat amounting to from 2° to 6° F. lost in traveling down and then up is regained, a feature of importance, as every 11° F. added to feedwater saves approximately 1 per cent. In the cylinder *n* there is a float connected with the water-supply valve *o*, and with the shell of the heater below the water line, thus equalizing pressures and causing the float to respond quickly to the demands of the heater and automatically to operate the water-supply valve.

Cleaning doors are so placed that both the upper and lower settling chambers are easily accessible, and through them and hand holes the sediment is easily removed from the various places in which it settles. The trays may be loosened by slackening on the nuts, and each segment moved in front of the tray-cleaning door.

The filter is cleaned by closing the downtake tube from the upper to the lower settling chamber by means of valve *p* opening the blow-off *q* and the wash-out valve *r* and closing the vent valve *s*. This forces the water through the reheater down into the filtered-water chamber and filter into the lower settling chamber to the blow-off. The safety valve *t* relieves excessive pressure when cleaning the filter with reverse currents.

This is the only open feedwater heater and purifier in which the current of water under pressure is forced through the filter in an opposite direction to clear the filter bed.

All scum, oil, etc. collecting on the surface of the water in the upper filtering chamber is flushed off into the overflow by raising the water level in the chamber *d*.

Each part of the heater is provided with vents by which any air and gas liberated from the water may escape. The lower settling chamber *f* is vented by *e*; the steam chamber *d* by pipe *u*; and the filtered-water compartment *l* by the vent pipe and valve *s*.

These heaters are manufactured by the Exeter Machine Works, of Pittston, Pa., who guarantee them to a working pressure of 10 pounds per square inch, and test them to 15 pounds before shipment.

SWEDISH STEEL RAILS

The Swedish state railways have closed a contract with the steel combine of Germany for the delivery of 22,800 tons of rails during 1910 at certain seaport towns at \$29.48 per ton. They have also closed another contract for the delivery of the same quantity for the year 1911 at \$32.16 per ton, delivery to be made at certain seaports in the kingdom.

COAL MINING NOTES

The Pemberton Coal and Coke Co., of Affinity, W. Va., found it cheaper to install the alternating-current generator and the motor-generator set for converting the alternating into direct current for use at the company's mine 2 miles distant from Affinity, than to install a larger direct-current machine, and transmit the direct current to the other mine. The additional apparatus bought represents a smaller investment than that which would be incurred in maintaining the losses in transmitting the direct current to the mine 2 miles away.

The Avondale Colliery, historical from the fact that it compelled the first mine law to be enacted in the United States, is now being drowned out by an inrush of water, presumably coming from some cave. The Delaware, Lackawanna & Western Railroad are owners of the mine.

The long expected commission to revise the coal mining laws of the state of Colorado was appointed on November 14 by Governor John F. Shafroth. The membership consists of Dr. Victor C. Alderson, President of the State School of Mines; Prof. J. B. Ekeley, of the Colorado State University; R. D. George, State Geologist; and James Dalrymple, the newly-appointed chief mine inspector. Colorado has some very effective mining laws upon its statute books but they have not been enforced in the past, largely from lack of power on the part of the inspector so to do. It is the intention of the board to hold a series of public meetings at which all interested will have a chance to be heard, to be followed by visits of inspection to the various coal-mining centers of the state. After these public hearings and visits of inspection, during which the particular needs of Colorado are expected to be brought out, the codes of the older states will be studied and the results thus obtained embodied in a set of new laws which will be submitted to the next session of the legislature for its consideration. The attention of the commission is called to the fact that the chief difficulty to be overcome in Colorado coal-mining practice is that of handling dust in the prevailing dry climate. When the dust question is satisfactorily settled, practically all others will take care of themselves.

It is with sincere regret that Mines and Minerals has to announce the resignation of John D. Jones from the position of chief mine inspector of the state of Colorado, a place which he has filled with distinguished success since his appointment by Governor Peabody on February 6, 1903. While Mr. Jones' resignation was tendered during September, it was not publicly announced until the time of the Delagua accident when he was leading one of the exploring parties, and became effective on November 21. Mr. Jones leaves the service of the state to become General Manager of the Oakdale Fuel Co., at La Veta, Colo., a position which he will fill with the same signal success as he did that of chief inspector of mines. What is most to be admired about Mr. Jones, aside from his being a self-made man of the best type, is the fact that with him it was always "come on" and not "go in." Mines and Minerals joins all his many friends in wishing him a long and successful career in his new undertaking.

In a report prepared at the request of Governor Shafroth, Mr. John D. Jones, the retiring chief mine inspector, recommends that the state be divided into five instead of three districts as heretofore, and that each district should have its particular deputy inspector, all in charge of a chief, with headquarters at Denver. Mr. Jones' recommendation that the salaries of the chief be fixed at $5,000 and the deputies at $3,000 per year, with proper allowance for expense, seems a wise one.

Governor John F. Shafroth has appointed James Dalrymple as chief mine inspector of Colorado. Mr. Dalrymple is well known to the mining fraternity of the state and since leaving Irwin, Pa., 25 years ago, has filled responsible positions in and around mines on the Western Slope. He has appointed Harry Douthwaite, of Colorado Springs, as deputy mine inspector, to fill the position made vacant by his own promotion.

On November 22, what promised to be a greater tragedy than any that has yet overtaken the coal-mining industry in Colorado was averted through the coolness of David Griffiths, superintendent of the Bear Gulch or Fremont Mine of the Colorado Fuel and Iron Co., 2½ miles southeast of Florence, Colo. A fire broke out in the underground mule stables about 4 P. M., while Mr. Griffiths was at the surface. With a hastily-organized force he entered the mine and sent warning to each working place. The men were gathered at a point where the air was clear, and in squads of 10 and 12 were made to hold their breath long enough to pass through the smoke and reach the cage. All but two of the 173 men in the mine were brought up in this way and these lost their lives by refusing to stay with the party beyond the smoke zone.

Both the Colorado Fuel and Iron Co.'s rescue car and that of the Bureau of Mines were on the scene about 2 o'clock in the morning of the day after but their services were not required except to recover the bodies of the two dead. The value of telephones to mines was again clearly brought out in this case, where the engineer in charge of the pumping station, being in constant communication with the outside, was able to remain at his post until the last minute.

The experience of the past, that with the approach of winter explosions in coal mines increase in number, is borne out by the reports for November. On Saturday, November 5, 15 men were killed by what the evidence at hand shows to have been a dust explosion at the Lawson Mine, Black Diamond, Wash.; on Tuesday, 79 died from a dust explosion started by an underground fire at Delagua, Colo.; and on Monday, the 28th, 13 more were killed by what is said to have been the ignition of a pocket of gas at the Jumbo Mine, 20 miles from Antlers, Okla. On November 25, 11 miners were entombed in mine No. 3 of the Providence Mining Co., Providence, Ky. This makes a total of 118 for the month, to which total Colorado contributed 79.

Since the first of the year, 210 have died in Colorado, either from explosions directly attributable to dust or which were increased in intensity by this agent. Colorado coal mines are naturally not more dusty than those of other states, but the extreme dryness of the air causes the entering current to absorb the natural and acquired moisture of a mine more rapidly than elsewhere and points to the fact that more watering than heretofore done is necessary. This and the complete removal of all accumulated underground track refuse will almost entirely prevent a repetition of such accidents as have occurred recently.

Referring to the rescue work at the Providence, Ky., disaster, Chief Inspector of Mines Chas. J. Norwood states that the work was in charge of Thomas O. Long, inspector for the district. He first repaired the damaged fan by the use of canvas, and restored the air circulation. He then led the first party that entered the mine, using an oxygen helmet furnished from the United States Mine Rescue Station at Linton, Ind., as his new helmet just received from the manufacturer had not yet been adjusted. Assistant Inspector Jones, of the Central City district, was also present with his oxygen helmet and assisted in the work. In addition to the Kentucky inspectors, Barr and Jones, and volunteers from nearby, there arrived, quickly after the disaster, the car from the United States Mine Rescue Station, at Linton, Ind., in charge of Mr. R. Y. Williams, and accompanied by his assistants, A. A. Samms and John B. Shepherd, who with apparatus and advice furnished valuable aid.

Engineer Antonio Llambias de Olivar, the representative of the boring company subsidized by the government of Uruguay, informs the Minister of Industries, Labor and Public Instruction, that in the middle of August, in the fourth boring which the company is making in the Department of Cerro Largo, a bed of coal was found at a depth of 124 meters, more than a meter thick and of quality superior to that met in the boring of November, 1909, at a depth of 140 meters.

THE DELAGUA, COLO., EXPLOSION

Written for Mines and Minerals, by George F. Duck

An Account of the Conditions Existing at the Mine before and after the Explosion

The most disastrous accident in the history of Colorado coal mining, one in which 79 lives were lost, took place at the No. 3 mine of the Victor American Fuel Co., at Delagua (pronounced Del'-ah-wah), Las Animas County, Colo., about 2 P. M., Tuesday, November 8, 1910. About 1:30 to 1:40 P. M., a fire was reported at the abandoned door *a*, Fig. 1, some half-mile underground on the 4th north entry, and while preparations were being made to extinguish it the mine "let go."

Delagua is the terminus of a coal road 6 miles in length, running west from Ludlow on the Colorado & Southern Railway. Ludlow is 15 miles north of Trinidad, 82 miles south of Pueblo, and has an elevation of 6,295 feet above sea level. Delagua is about 20 miles north of the Starkville Mine, which exploded with the loss of 56 lives on October 8.

The seam worked, known as the Delagua, is in the upper part of the Laramie formation of the Cretaceous. It averages rather more than 6 feet in thickness with a persistent slate or bone parting near the middle from 1 inch to 6 inches thick.

The coal outcrops around the hillsides and has an average cover of 150 to 200 feet. The roof is poor and requires the use of much timber. The regular draw slate is not generally present but rather a soapstone, from 6 inches to 15 feet thick, full of slips which cause it to fall in irregular shaped pieces of no great size and with sharp edges. The coal is friable and will not bear stocking for any length of time. The composition is, essentially, 36 per cent. volatile matter, 8 per cent. ash, 2 per cent. moisture, and 54 per cent. fixed carbon. The sulphur is below 1 per cent., and when washed to remove fragments of the parting the coal makes a very satisfactory coke for furnace use.

It is stated that gob fires, blown-out shots and firedamp are unknown.

The surface plant, destroyed by fire on October 7, consisted of a wooden tipple of 2,500 to 3,000 tons daily capacity, a 600-ton washery, 600-ton storage slack bin, power plant, boiler houses, repair shops, etc. There are 160 standard beehive ovens in block. Five mines fed to this tipple, No. 3 producing about 1,250 to 1,500 tons daily. Since the fire, however, the production of No. 3 has been about 750 tons Delagua is known as the largest single coal producer in the state.

The system of working is the ordinary double-entry with rooms turned off both the main heading and air-course, or back heading as here known. There are four parallel headings, known as the 1st, 2d, 3d, and 4th north, respectively, driven on a course of 6° 45′ W, which were opened in the order given. No. 1 mine (not shown on the map) is 1,300 feet east of the No. 2 mine. It has its own fan and is independent of the other mines, but is connected with No. 2 at several points inside.

The No. 2 mine is 1,000 feet east of No. 3 mine and is also independent but is connected with No. 1 and No. 3 inside. Its fan is not in use, ventilation being secured through No. 3. The heading proper is 8,000 feet long.

The 3d and 4th north entries and their branches constitute No. 3 mine, the scene of the accident. They are driven from a slope *b* at present 2,600 feet long.

The 3d north entries, which leave the slope 675 feet inside, are 8,150 feet long, are advancing and have no rooms until those on the last pair of cross-entries are reached.

The 4th north entries are 650 feet west of the 3d, are 7,050 feet long to the face which is stopped in a fault or pinch. No rooms are working on these headings until near their end.

The main slope is not now advancing as the roof is being shot to bring the track to grade.

The 5th north entries start from the end of the slope and are working, headings being driven back to meet them from the 3d west and Rock Island, as shown in Fig. 1.

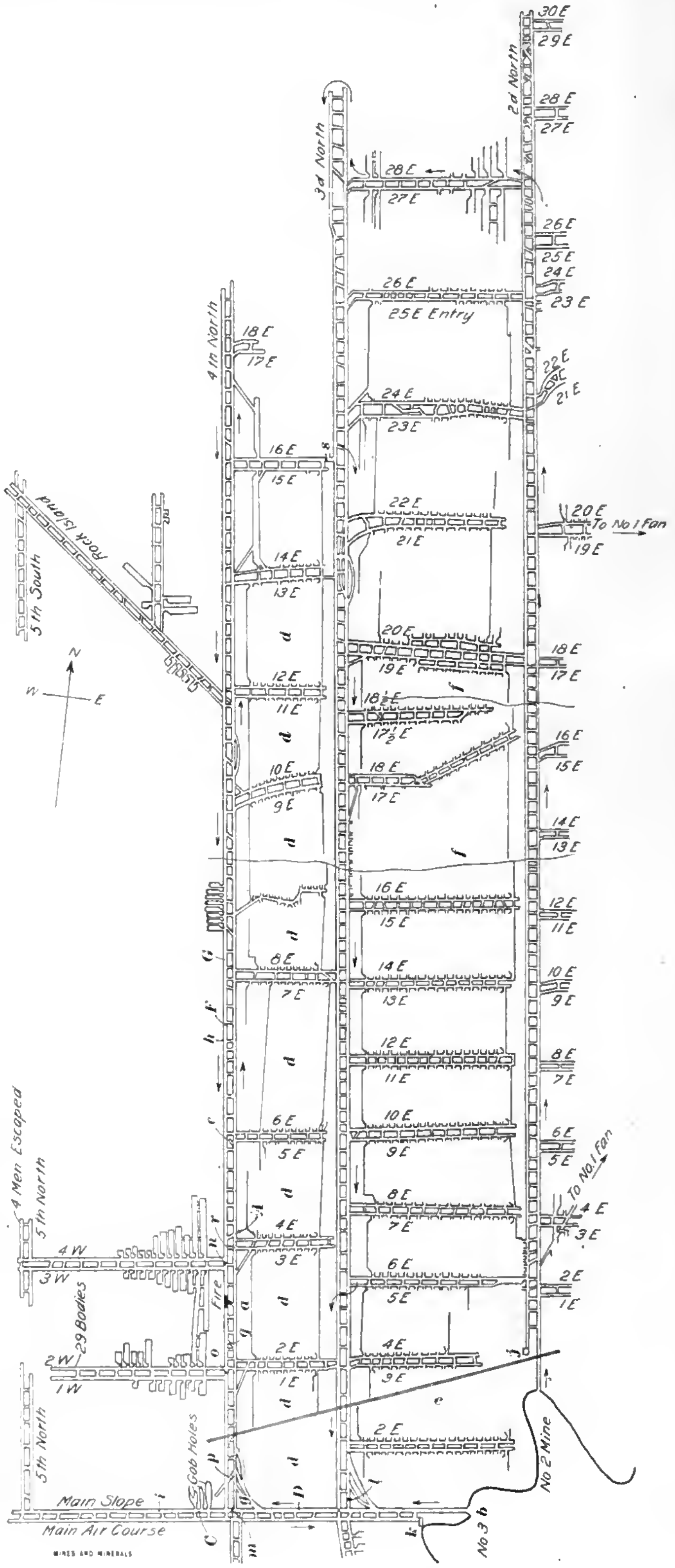


FIG. 1

A = slate forced into inby side of cap.
n = door blown outby.
C = slate forced into outby side of cap.
D = slate forced into inby side of cap.
l = door blown inby.
F = slate forced into outby side of cap.
G = slate forced into outby side of cap.

The bulk of the coal from the 4th north section comes from the rooms driven from the 1st, 2d, 3d, and 4th west entries,

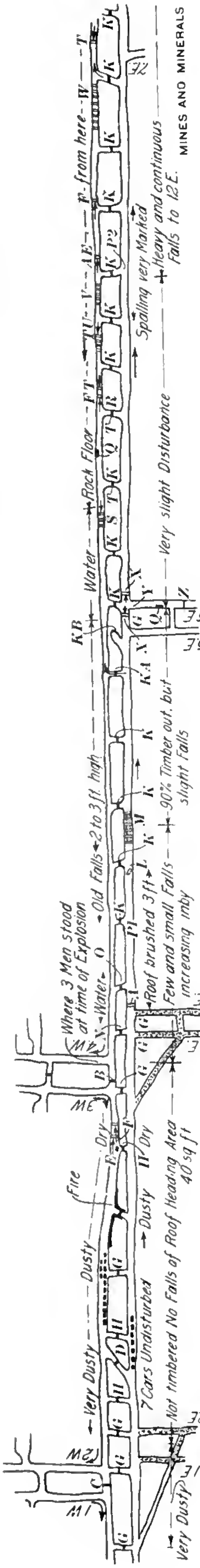


FIG. 2. ENLARGED SECTION OF ENTRIES AT FIRE AND COKING ZONE

A, slate forced into cap; *B*, door blown outby; *C*, curtain, unharmed; *D*, first west haulage door, slightly bulged into back heading; *E*, heavy coke on outby side; *F*, light coke on outby side; *G*, gobbed snut, not damaged; *H*, Bulged in toward back heading but not broken; *K*, entirely destroyed, blown into back heading; *L*, timber fallen outby; *M*, undisturbed timbering; *N*, brattice through which men escaped; *O*, upper ½ apparently out before fire; *P1*, electric pump not harmed but rheostat blown into back heading; *Q*, coke on rib 40 feet inby; *R*, the only stopping blown into haulage road; *P2*, electric pump blown 3 feet and rheostat 20 feet, very violent action here; *S*, three caps with center posts. Heavy coke on inby side of all members; *T*, coke on inby side; *U*, four sets caps, very heavy coke inby, some little outby; *V*, six caps unharmed; *W*, nine sets timber unharmed; *KA*, used as tool house by men working in 5th west; *KB*, fifth west started since November 1st on night turn; *X*, very heavy dust; *Y*, gob thrown into entry, fall, heavy spalling opposite mouth; curved arrows show direction of throw of heavy materials; small arrows show place of deposit of coke, slate or other materials; *Z*, Brattice blown inby; ▮▮ caps in place.

respectively. The 5th west was started at *c* opposite the mouth of the 5th east on November 1 and is worked nights.

The Rock Island entry, or more generally the Rock Island, is shown as starting 4,650 feet from the slope on the 4th north return air-course. Its name is derived from the fact that it is advancing into coal leased from the C. R. I. & P. R. R.

In Fig. 1 large areas marked *d* are caved, and although the pillars have not been drawn, the roadways and rooms are filled with falls of roof. It is said to be possible to walk or crawl over these old falls from the slope to the extremity of the workings. In this part of Colorado the dikes *e* and faults *f* as shown on the map, commonly give off marsh gas brought up from lower seams, so that it is by no means impossible that at intervals, at least, some little fire-damp is present in these abandoned workings.

The coal is undercut by the miner who also drills, charges, and tamps the hole, and after affixing a flag or marker, leaves the actual firing to the company's shot firers. While the introduction of "permissible" explosives is meeting with marked success among the more progressive men, the mine is largely a "black-powder mine." One-quarter of a keg of powder is allowed in one place at one time.

After being gathered to the various partings by mules the coal is hauled to the main slope by Jeffrey motors operated under 250 volts pressure, and up the slope by a rope The coal from the main slope below the 4th north is hoisted to the latter by an electric hoist at *g* in the cross-cut.

Some little water is met in the return air-courses which are to the dip of the haulage entries. This is handled by two small electrically driven pumps at *h* on the 4th north and a larger one at *i* on the slope.

As stated above, although connected with No. 2, No. 1 mine, the first pair of north entries, has its own fan. Also, No. 2 fan, at *j*, is not in use and has not been for some time.

The fan ventilating both No. 2 and No. 3 mines is that placed at *k*, the end of the return airway of No. 3. It is a Capel fan, 15 feet in diameter, driven by a motor on an independent circuit. At the time of the explosion 55,000 to 60,000 cubic feet of air a minute was circulating through the two pairs of entries.

The air is drawn in through No. 2 mine, passes up the 2d north its entire length, thence over to the 3d north entry of No. 3 mine, down it to near the main slope where it passes into the air course, across the slope in an overcast to the main return and out it to the fan. Thus, the 2d north is the intake and the 3d north the return for one split, the air traveling a total distance of 18,000 feet. Doors *l* are placed on the 3d north to separate it from the 4th north and also from the main slope.

The air for the 4th north comes in the main slope and is prevented from going down the same by a door *m* on the slope. It is thus drawn into the entry, to its face, back along the return to the 4th west cross-entries up which it is forced by a door *n*, a canvas curtain *o* forcing it up the 2d west, thence to the main air-course by way of the Rock Tunnel *p*, traveling a distance of 21,000 feet. The Rock Tunnel is shown in dotted lines between the 4th north entries. It is driven in rock and was intended for an overcast but the bridging at the slope has not been completed.

Sufficient air for the four men working in the 5th north entrles off the slope is thrown down by a regulator placed in the 4th north return between the slope and its air-course.

The Rock Island is ventilated by diffusion only, a comparatively few men being employed there.

Both the intake and return airways vary from 35 square feet to 100 square feet or more in area but average by several measurements about 50 square feet. This change in cross-section has the effect of rapidly increasing or decreasing the velocity of the air and had a marked influence upon the currents generated by the explosion.

Fig. 2 gives the details of the ventilation of the west entries.

The explosion at Delagua No. 3, whatever its origin, was unquestionably produced by coal dust, as at Starkville. Fire-damp undoubtedly did not figure in it at all, and it was not caused, independently of the coal dust, by the direct ignition of the hydrocarbons distilled from the burning coal. That the condition of the mine prior to the explosion may be understood certain definitions are permissible.

For the sake of clearness, a mine is defined as very dusty when the track and roadways are covered to the depth of at least an inch with fine, almost impalpable dust, most of which will float on water; as dusty, when the particles of dust are larger than in the above case, not so thickly deposited and when but a comparatively small percentage can be blown from the hand or will float; as dry, when the coal is in pieces up to ¼ inch in size with no evidence of moisture to the touch. The very dusty conditions are usually met at partings where the coal is churned into powder by the constant passage of men, mules, and motors; the dusty conditions along some well-traveled entry; and the dry, in rooms which have been advancing for some months. A dry mine is not quite safe, a dusty one is dangerous, and a very dusty one, highly so.

On the other side, a damp mine may be considered as one where the material in the roadways has no tendency to blow away when disturbed by the feet of men or mules and leaves a distinct impression of moisture to the touch; a wet mine, as one where the road cleanings may be moulded into balls by the hand, and where occasional patches of water are encountered; and a watery mine, as one where the track is deep in mud or

where standing water over an inch in depth is encountered generally throughout the workings. The damp mine is safe, and a watery one cannot explode. These definitions are not absolute and are modified by other conditions. Thus, a section of the mine marked as dry may be in a rock tunnel and hence absolutely safe.

According to the foregoing definitions the conditions in Delagua No. 3 mine as determined by observations made 36 hours after the accident were essentially as follows: All return air-courses were "safe," coming under the terms of damp and wet, and, in many cases, even watery. Standing water is frequently met in the air-courses which mostly have a rock floor from falls of roof.

No. 2 mine was not visited at the time of the accident, but was classified 10 days after the explosion as dry to dusty. The 3d north entry, from its face to the 8th east cross-entries, should be defined as damp to wet, having been so freely watered on Sunday that haulage was difficult on Monday. It was impossible to determine the exact conditions owing to the falls, but it is reasonable to suppose that this entry became dryer as it approached the slope where it was more constantly traveled.

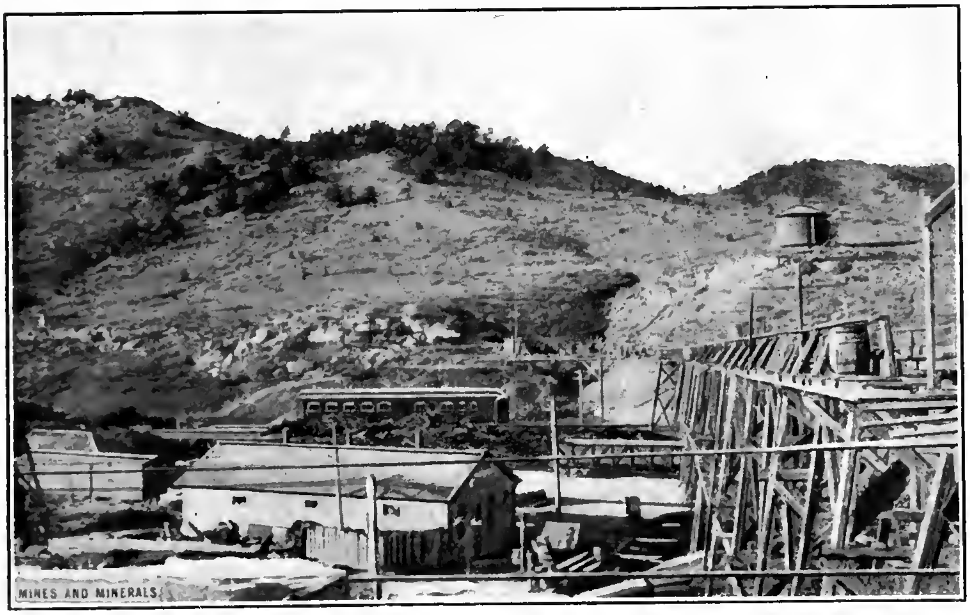


FIG. 3. DELAGUA NO. 3 MINE, SHOWING U. S. RESCUE MINE CAR, TEMPORARY TIPPLE, AND WATER TANK TO RIGHT OF DRIFT MOUTH

The 4th north entries, also watered on Sunday prior to the explosion, were dryer than the 3d north, and should be classified as damp to dry; that is to say, they were essentially safe. The Rock Island was damp and the two pairs of west cross-entries off the 4th north entries were the same. The partings on both the entries at the main slope, the 7th east and 10th east partings on the 4th north, some partings on the 3d north and the parting near the fire *a* on the return of the 4th north between the 2d and 3d west were dusty. In general, so far as the roadways were concerned the mine was safe.

The season has been an exceptionally dry one in Colorado. The Trinidad office of the Weather Bureau reports rain as follows: For August, .42 inch; September, .45 inch; October, 1.41 inches, and November, none. The October rain fell on the 16th, 18th, 19th, and 20th of the month, and except for a trace, less than .01 inch on the 26th, there was no rainfall for 19 days before the accident.

L. H. Dangerfield, local forecaster, in charge of the Weather Bureau at Pueblo, about 80 miles north of Delagua, and the nearest station thereto, has kindly supplied the observations for several days prior to the accident. As they are of considerable interest, the figures for 4 days are given.

Day	Temperature			Relative Humidity		Barometer		
	6 A. M.	Noon	6 P. M.	6 A. M.	6 P. M.	6 A. M.	Noon	6. P. M
5	36	48	48	80	41	25.48	25.46	25.42
6	23	69	61	82	10	25.42	25.31	25.30
7	30	67	58	34	17	25.31	25.23	25.20
8	54	70	65	13	6	25.13	25.08	25.02

Mr. Dangerfield gives as the mean of a 21-year period the following averages for November: Temperature, 43.9 degrees; barometer, 25.31 inches; and relative humidity for 6 A. M., 62.9 per cent., and for 6 P. M., 41.0 per cent., a daily mean of 52 per cent. It will be noted that prior to the explosion from 6 P. M. Sunday, the temperature was slowly rising and the barometer and relative humidity rapidly falling. The high temperature, which is about the annual mean of the place, while favorable to preserving the moisture in the mine was more than offset by the abnormally low humidity. The air was practically dry and greedily absorbed every drop of available moisture.

The accident at Delagua must be studied from the two propositions, that of the explosion, which happened about 2 P. M., and that of the fire which certainly was burning as early as 1:15 P. M.

Had some one not been careless, there would have been no fire; had the mine not been dusty, there would possibly have been no explosion. The two hypotheses, while undoubtedly connected, could have happened independently.

The fire occurred inside a disused door in a cross-cut between the 4th north entries. This door is 230 feet from the 1st west door *q* and 330 feet from the 2d east entry.

The door was built of ordinary 1-inch plank, had an area of about 30 square feet, and was placed about 20 feet in from the haulage road and 12 feet from the return, where the parting for the west entries was made. It was the custom of the drivers to eat their dinners on some benches just inside the cross-cut, and not infrequently other company hands used the opposite or the haulage side for the same purpose. There was probably the usual mine refuse in the place, chips, pieces of board, paper, and fat from dinner buckets and the like. This door was entirely destroyed by the fire, except the two side posts, and before the fire zone was cleaned up, from measurements made, it was seen that the fire burned 100 feet south and 80 feet north from this point on the return airway. The fire did not quite reach the haulway. It is evident that the fire started on the return airway side of this abandoned door. The effect of the fire was almost entirely on the rib nearest the main entry, which in places was burned and coked to a depth of 6 inches.

The time when the fire started and how it started, since no electric wires were near the door and the first observers of the trouble are dead, will probably never be solved, although there are a few fairly-well established times and a few apparent facts bearing upon its origin.

It is stated that Jos. Boyd, a fire boss of experience, and a competent man, passed the place at 12:05 and noticed nothing out of the way. The fire was undoubtedly not started at that time.

At 12:50 a miner from the 4th west went to the parting to inquire if more cars would be given after dinner, it being election day and the mine not running full. He found the two colored drivers, Sampson and another, eating their dinner, not at the usual place, but leaning against the cars 30 feet from the cross-cut and on the heading. If the drivers had fired the place at an earlier hour and supposed they had put it out they did not mention the fact to the miner. In any event, shortly before 1 o'clock, the miner, an intelligent Austrian, who subsequently saved himself and numerous others through his coolness, noticed nothing wrong. At this time, any previously started fire was by those possibly implicated supposed to be out. The miner returned to his place in room 6 on the 4th west and "in a few minutes" received a car which he and his buddy loaded and dropped down 150 to 200 feet. After talking over the possibility of more cars, one of them hunted up the driver on his heading, Sampson, and being advised that he would have none until 3:30 p. m. returned to his place and with his partner, gathered up their tools, clothing, and buckets, and prepared to go home. At the switch to their room they met the driver who asked if they had water in their buckets as there was a fire at the "old door."

All this took time and it was probably fully half an hour from when the miner left the parting where he inquired for cars to when, satisfied that no more would be received within a reasonable time, the two decided to go home and were then advised as to the existence of the fire. What happened in this 30 or 35 minutes is merely conjecture. The men on the 4th west cross-entries being on the intake side did not notice the smoke and of those on the 1st and 2d west cross-entries, where the smoke traveled, all but one, an Italian driver, are dead. The 3d and 4th west entry drivers were also killed.

The 2d west driver says he started to work at 12:35 (which appears a little early) and on the third trip out noticed smoke, etc. If he made four trips an hour, his mule would be handling over 80 tons a day, which is not bad work for the distance covered. This would make it about 1:20 when he noticed the smoke on the airway. It must be remembered that the men first referred to were inside the fire and each party knew nothing of the other's movements. Why none of the men on the 1st and 2d west entries when first noticing smoke, and it must have gone to their places in a very few minutes, did not attempt to escape, is as much of a problem as why the driver, Sampson, did not get water from the "water hole," but 80 feet away in the 4th west return, or why he did not notify the pit boss, Llewellyn Evans, who, in making his rounds by way of the 3d north, had supposedly reached the 10th east parting on the 4th north, 3,000 feet inside the fire.

The two miners inside the fire zone, after being notified, went in that direction and opening the door, noticed the "smoke rolling over and over and 'swishing' back and forth." After trying to get through under the smoke they returned to their places, notified the men in the 4th and 5th rooms, gathered up their outfits and with axes came back to the entry. They again looked through the door and found the fire was growing dull red and that the agitation of the smoke was more violent than at first. They again failed to get past under the smoke and after deciding not to try the return, proceeded to cut and scrape a hole through a brattice *r* just inside the 4th west door. As but one man could work at a time, they estimated that this would require 4 or 5 minutes. After relighting with difficulty the lights which, with the exception of an acetylene lamp, had all gone out, they crawled through the brattice and noticed the smoke within 10 feet of them on the main heading.

After trying to pass the fire on the haulage road side they turned into the mine and near the 10th east cross-entries met the pit boss and the nipper on their way toward the fire. Going further, they picked up the drivers from the Rock Island and warned many others, most of whom treated the matter with indifference, crossed over to the 3d north by way of the 16th east cross-entry, went out it to the 26th or 27th east cross-entry and to the 2d north entry and had just turned out of it for the No. 2 opening, when they felt a slight jar of the air which they afterward learned must have been caused by the explosion. As they were afraid of an explosion while on the 4th north, they probably ran as far as the 10th east, but from there in they walked in a more leisurely fashion. It is 2,750 feet to the 10th east from where they broke through the brattice and 4,000 feet by the usual route from there to the 2d north Making all allowances for running and for stopping to talk to men on the way, it must have been about 1:40 p. m. when they met the pit boss at the 10th east, about 1:35 when they broke through the brattice, about 1:30 when they started to do so, and 1:20 to 1:25 when they were notified of the fire by Sampson and after which they gathered up their companions, etc.

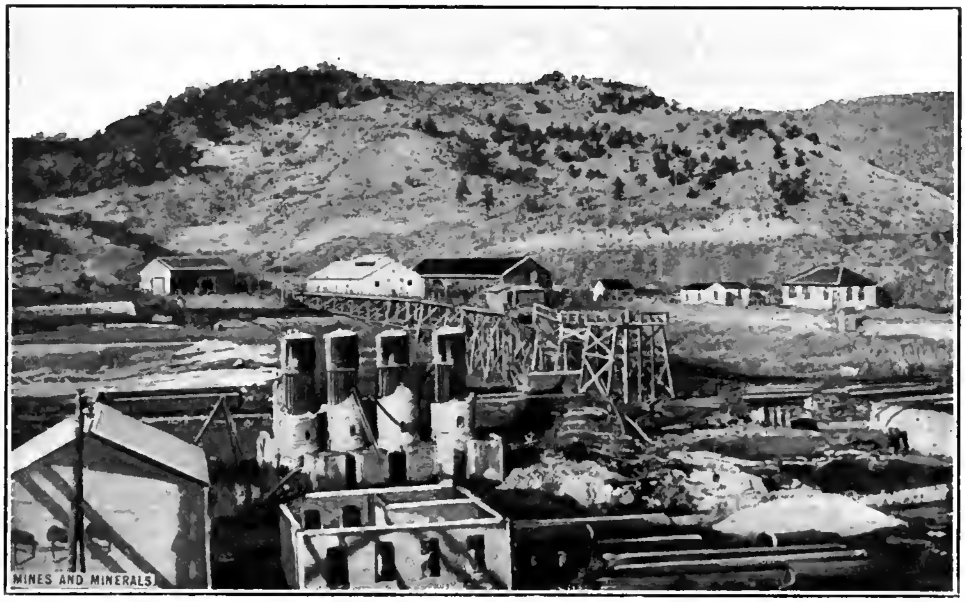


Fig. 4. Delagua No. 3 Mine From Hillside Opposite Fig. 3. Shows Temporary Tipple From No. 2 Mine. White Building is Blacksmith Shop. No. 2 Opening is to Left of Blacksmith Shop

On the return beyond the fire zone the time figures out fairly well. The 3d west driver, mentioned before, at an estimated time of 1:20 p. m. met the motor man, Bennett, at the 1st west door *q*. After some examination of the fire and a trip on the part of Bennett, it was decided to send the nipper, "Red," in to the 10th east to notify the pit boss. He must have been sent prior to 1:35 p. m. as by that time the miners on the inside could not get through the smoke on the haulage road, and also these miners met him and Evans at the 10th east parting about 1:40 p. m. It must then have been about 1:30 when Bennett, sent the boy back to inform the pit boss, whom he must have just left at the parting mentioned.

Outside the mine, at about 1:30 p. m. the smoke was noticed coming in thin clouds from the fan which was supposedly on fire. Thirty thousand feet of air a minute through an airway with a cross-section of about 50 square feet was traveling from the

fire to the fan a distance of 3,900 feet. This requires $6\frac{1}{2}$ minutes for the air to reach the fan and establishes the time at 1:23 P. M. when the fire was noticeable. This agrees with the figures obtained inside.

William Lewis, the superintendent, and his assistant, W. J. Evans, were in the office about 1:35 P. M. when notified of a supposed fire at the fan. After hastily investigating this and finding the fire to be inside and after making arrangements for a party to get water in barrels and for others to follow them, Lewis and Evans, started in "on the run." On the way they met the slope pumper, Smith, who reported a fire in the 4th north, "somewhere." Smith went back with them. It was some time after 1:40 P. M., perhaps nearer 1:45 P. M., that they reached the fire. On opening the 1st west door *q*, Evans remarked, "the 1st west is gone, we had better get out." Lewis, seeing that the fire could not be reached with buckets, sent the motorman out with his empty locomotive for hose. This must have been about 1:45 or a little later, for the motorman had time to drop his trip, set his switches, to stop twice on the way out, the distance being about 2,750 feet, and to help 3 to 5 minutes on the outside after his arrival there, before the mine "let go."

It seems more than a coincidence that the time. figured from such diverse starting points, both backwards and forwards, should agree so closely. It may be considered as settled that at any time between 1:20 and 1:25 P M. the fire was beyond control and that as late as 1 o'clock it had either not started, or having started was supposedly out.

After sending the motorman out, Lewis, who had supposedly been joined by several other of the officials, waited at the fire until something happened, perhaps the fall of the old door, to convince them that they were in immediate danger and that nothing further could be done. It may have been as late as 1:50 when the party hurriedly left the fire. Some believing they would be, safer from the impending explosion turned into the 3d north, but Mr. Lewis in company with several others continued out the slope and were overtaken by the blast about 400 feet from the mouth where they were as badly crushed and burned as were their companions in the side heading. Some comment has been made as to why the men were not warned in time to escape and the answer is obvious; no one knew of the fire until too late. Being on the outlet and near the fan it was not noticed by the only official inside until his attention was called to it by a boy, the nipper, "Red," who probably told him. "Bennett says there is a fire at the door and you had better come." Evans had only a child's statement for the fact and lost his life in the endeavor to get through. Lewis knew nothing definite of the fire until within 17 minutes of the explosion, when it is possible that those on the 1st and 2d west were already dead, and the others could not be reached through the fire. No one was killed on the 3d north entries except the officials mentioned as Lewis' companions, and three men who were asphyxiated in attempting to get through. It seems probable that, instead of attempting to escape, Lewis sent his companions through the 3d north in an effort to warn the men of the fire, while he and the others continued on out the slope to hurry up the hose, fire extinguishers, and other needed apparatus? This is the more charitable and, from the known courage of all the dead, the more probable view.

The explosion took place, as nearly as can be determined, at 1:57 P. M., and this time has been taken as correct in our estimates of the fire. The relative time alone is of importance.

A few seconds before the explosion an extremely dense cloud of heavy black smoke came out the fan, almost immediately turning to a light gray only to be followed by the shock. This was severe but not accompanied by any great noise. The fan was not damaged except by the blowing off of the roof and continued in motion until the current was turned off. This want of damage to the fan is undoubtedly due to the sudden increase in area of the return from about 50 to nearly 200 square feet near the mouth so that the gases had a chance to expand, to the roof being light and offering little resistance, and to the further fact that the airway being damp and having a rock floor afforded nothing for the explosive energy to feed upon.

At the drift mouth the blast came with great violence, the fall of the timbers and roof completely blocking the entrance. The casing of Bennett's motor was torn off and blown 350 or 400 feet and he and two others were killed by the flying stones and three others more or less injured.

In the mine the damage was confined to the blowing down of the timbers and consequent falls of roof on the main slope from the 3d north entries to daylight, on the 3d north from the 2d to 4th east cross-entries, with smaller and decreasing falls out to the 7th east, and on the 4th north entries from about the 7th east to the 12th east cross-entries. None of the air-courses were harmed, but with one exception, the air stops were all blown from the haulage roads into the return airways. Little or no damage was done to the slope below the 3d north entries, none at all to any of the four west entries or to the Rock Island, and the force was so slight at the ends of the 3d and 4th north, that many men continued at work until the rescue parties entering through No. 2 mine warned them of the accident.

Twenty-nine dead were found on the 2d west cross-entry at the last cross-cut and four more between that point and the fire on the 1st west entry. All these were asphyxiated, possibly by smoke prior to the explosion. Two were found on the slope at the 4th north; five on the slope between the 3d north and daylight; six on the 3d north not far from the slope; three on the 4th north back return not far from the fire; and others at various partings on the 4th north. Of the 135 miners and 22 company hands who entered the mine on the morning of the eighth, 88 either escaped after the explosion, between the time of the fire and the explosion, or had left before the fire started. The death list came to 79, of whom one was a visitor at the drift mouth, two who went inside out of curiosity, and Willis Evans, in charge of the C. F. & I. rescue car, who lost his life in bringing out 4 men imprisoned on the 4th west by the afterdamp. Thus, of the employes of No. 3 mine, 75 were killed, and as 6 of these can hardly be classed as company hands, 69 of the 157 going into the mine that morning are dead.

The evidence afforded by deposits of coke as to the direction taken by the explosion is conflicting and at best only confirmatory. The best evidence is afforded by the throw of heavy materials.

On the main slope several pieces of slate were found driven into the dip side of a cap about half-way between the 4th and 3d north entries and it is known from this that from the 4th north the explosion moved outwards to the mine mouth. Just below the 4th north, the door *g* across the slope was shattered and a piece found 195 feet down the slope. Also a short distance inside from this door slate was found forced into the pitch side of caps. Therefore, on the slope section, the force went both ways from the 4th north. On the 3d north, the door *l* across the entry was broken and blown 30 or more feet inwards. A timber further in was found blown inwards. Hence the explosion went into the 3d north from the slope.

At the mouth of the 4th north entry, a piece of timber has scarred the lower rib, while flying outward. At the Rock Tunnel, the gob was forced outwards from the haulage road. Just inside the 4th east cross-entry, a piece of slate was forced into the dip side of a cap. One hundred and twenty feet further in a heavy timber was plainly thrown outwards. From the 6th to the 10th east cross-entries, 45 posts are standing with their heads inwards and 22 with heads outwards along the left rib, and along the right rib 29 have the heads pointing outwards and 9 inwards. At the 7th east parting, 3 cars of a 7-car trip are off the track and "bumpered" in such a way that the force of the explosion must have been inward.

On the 3d north entry air-course no evidences of direction were apparent, but from the slope to the 12th east cross-entry,

all the brattices that were not gobbed shut, and even some that were, are forced from the haulway into the return airway.

The main slope air-course was not injured from the 4th north except that the brattices were blown in.

The 4th north entry air-course, however, does show marked evidences of direction. The door between the 3d and 4th west cross-entry was blown 30 feet toward the fire, and plastered with mud which could only have come from the water hole 80 feet farther inside. On this same air-course and 550 feet in the 6th east cross-entry, slate is found forced into the outby side of a cap. Slate is also found in the same position in another cap 450 feet farther in. As to brattices, all from the 4th west entry to some little distance beyond the Rock Island, with one exception, are blown in from the haulway to the air-course.

Starting somewhere on the 4th north main entry within a space of 610 feet from, and probably near, the 5th east cross-entry, the explosion traveled in two directions. That going north gathered up the dust at the 7th east cross-entry parting, probably resulting in a second explosion, and a third took place at the 10th east cross-entry parting. Meeting the wet and new section of the mine at the 12th east cross entry the explosion stopped for want of material. Entering the air-course at about the 5th east cross-entry the explosion also traveled north, forcing slate into the caps as noted, but doing no great harm as this heading is wet to watery and has generally a rock floor due to falls of roof.

From its origin, the explosion passed south to the slope, on its way forcing slate into a cap piece, blowing the brattices into the air-course as far as the fire, beyond which point none were destroyed and only a few bulged into the return. Passing the Rock Tunnel the explosive force threw several carloads of material into it. Coming to the network of dusty sidings at the slope, there was another explosion, but having room for expansion it did little harm. At this point the force divided, one arm turned down the slope, destroying the door, but was stopped at the "gob hole" by the water and by a large amount of rock dust on the floor. The other arm turned up the slope, forcing slate into the caps on the pitch side, but again, owing to the large cross-section, did little damage until it met the 3d north siding, where, meeting fresh supplies of dust, the final explosion occurred. The force extended but a short distance into the 3d north entry owing to the heading being wet. It was this last explosion, the one at the 3d north that did the most damage to the mine.

On the 4th north entry return air-course, the force entering near the 5th east cross-entry passed over the rock floor and water and knocked down the door between the 3d and 4th west. Why this branch did not explode in the presence of so much dust just outside the fire is a mystery, as deposits of coke are found between this door and the fire, indicating intense heat action.

These different points of probable explosion are clearly indicated, not only by the known condition of the mine at each place, but also by the presence of coke on caps and ribs. Coke is found in large quantities on the 3d and 4th north sidings and opposite the 7th east parting in the return. It is also found in the 6th east entry above an old fall 40 feet in from the 4th north. It is found on the return between the 3d west door and the fire in one place. All the places where the coke is found were either very dusty or near places that were. No coke was found near the 10th east cross-entry, probably because it was not looked for and it may possibly be found if careful examination is made.

While there were undoubtedly a series of explosions, the time interval between them was but a minute fraction of a second, so that to observers the action appeared instantaneous.

Several theories have been advanced as to the cause of the explosion and its place.

The first supposes that it was caused by the ignition of the heavy hydrocarbon gases, resulting from the burning of the coal in insufficient air and that it was done by the lamps of one of the two men whose bodies were found on the slope at the at the 4th north on opposite sides of the door. In any case, the man outside could not have lighted it, as a heavy door was between him and the return, and had it been ignited by the man inside, the door would have been blown out and not down the slope. For every reason this theory is untenable.

Another theory suggests that the gases at the fire, meeting sufficient fresh air when the burning door fell, combined with the increased supply of oxygen with explosive violence. It may be admitted that such a *minor* explosion is possible and that it may have been the cause or starting point of a dust or other wave of disturbance, but that it, and it alone, without extraneous aid was the cause of the loss of life and wrecking the mine, is impossible. Aside from the fact that we have no knowledge to prove that the coal was not burning to CO_2 direct, the explosion, the actual detonation, did not and could not have taken place at the fire or near it.

Explosive energy radiates in all directions from its point of origin along lines of least resistance. A study of the map proves that what little energy is encountered near the fire is all toward it and not away from it as it would be the case had the explosion taken place there. On the west, none of the headings are disturbed in the slightest degree. Even such light objects as cigarette papers are where left by the owners. On the north, the door at the third and fourth west entries is blown in toward the fire, but above all, three men standing 10 feet inside the mouth of the fourth west entry 240 feet from the fire, were merely knocked down by the concussion, and a companion 200 feet further up the fourth west entry was not jarred. On the east, the brattices were bulged in toward the return, but not broken, and on the slope side of the fire were not even harmed. On the south, the canvas curtain between the second and first west cross-entries was unharmed. The point of explosion must plainly be looked for elsewhere.

All indications point to the fourth north not very far from the mouth of the fifth and sixth east cross-entries as being the point of the initial, but by no means the most violent, of the series of explosions which occurred. Just beyond this point the energy is proven to have gone inward and just before it, outward. The brattices beyond the old falls 90 feet up the fifth and sixth east entries are shattered to pieces and blown inward, and intense heat is proven at this point by a small deposit of coke on the rib of the sixth east 40 feet in, as well as by the spalling action on the roof opposite the mouth of the fifth east. No coke is found in the air-course immediately opposite this point, but it begins a few feet farther in and is found in numerous places to a point beyond the seventh east cross-entry. Perhaps there was but one explosion at this point and not two, as the writer supposes, but indications point to one at the fifth east and another at the seventh east parting.

Because of lack of damage at the fire, it was first thought that it and the explosion were entirely independent. It seemed possible that a short-circuiting of the trolley wires brought about by a fall might have ignited the dust and that the fall, if not otherwise caused, might have been produced by the jar to the air caused by the sucking in of the door at the fire, the journey of the current being suddenly changed from one of 20,000 feet to less than 4,000. It also seemed possible that some stranger to the mine might have climbed into the 6th east entry with a naked light and ignited standing gas brought in there from the dikes mentioned. Another view was that the night shift working the newly started 5th west entries had left a full supply of powder behind, which was carelessly ignited by some passerby. Any one or all of these causes could have ignited the dust, and except the second, independently of the fire. But a thorough investigation into the gas question immediately after the accident, the accounting for all of the missing, and want of knowledge as to any powder being present, compel the abandonment of this view for one advocated, in part at least, by

some of those on the scene shortly after the explosion. It is freely admitted that there are weak points, largely ones of time, in the argument, but in want of definite knowledge it is given for what the reader may think it worth and the objections frankly stated. This is an attempt to connect a fire on the return with an explosion which took place 1,000 feet beyond it on the intake. The difficulties are obvious. This view supposes that the danger became imminent when the motorman left at top speed for outside, which we have shown was 10 to 12 minutes before the explosion. The hurrying motor stirred up vast clouds of dust at the slope parting 1,000 feet outwards from the fire. As the evidence shows that the fan was slowed down shortly before the explosion, the air was probably not traveling with its normal velocity of 600 feet per minute. Assuming 300 feet, it would require 3 to 4 minutes for the bulk of the dust to go up the 4th past the fire. By this time the door had burned partly through and the air was beginning to short-circuit. What dust passed through the openings was burned inside and the quantity of entering dust constantly increasing, it began to "backfire" into the haulage road. Perhaps it was this backfiring that convinced the party at the fire that danger was imminent.

The door, disintegrating more and more rapidly by reason of fire on both sides, finally burned down and the sudden short-circuiting of the entire air-current of this section of the mine threw still larger volumes of dust into the fire from the roadways, and possibly by suction from the gob holes and cross-cuts, thus producing more intense inflammation. While this was going on, the major portion of the dust was continuing up the 4th north, growing warmer and warmer until it finally began to distill gases. The falling of the door and the short-circuiting of the air caused increased volumes of dust to go out the 4th north, the air already having a tendency to go in. The gases were distilled more and more rapidly until when the 5th east was reached the action was so intense as to cause an explosion. Traveling inward the gases met fresh volumes of dust at the 7th east and 10th east partings and exploded, but stopped at the 12th east cross-entry for want of fuel. Going outward from the 5th east cross-entry the gases traveled with intense velocity through the rifle-like section of the entry, in some places of but 35-feet area, until, encountering the dust at the slope partings, another explosion occurred, but did little damage by reason of the very large cross-section of the mine at this point. The explosion did not travel down the slope, as it was wet, and did little damage between the 4th and 3d north because of the large area. At the 3d north sidings, where there were large volumes of dust, the final and largest explosion occurred. Going in along this entry, its cross-section diminishing rapidly, the place was completely wrecked, but for no great distance, the force being stopped in that direction by the watering received the Sunday before.

This view likens the affair to the ignition of a train of powder, the fire being the match and the points of explosion being occupied in this case by dusty places in the roadway.

The chief difficulty in accepting this view is finding an explanation why the explosion did not start as soon as the motor stirred up the dust on its way out. Part of the time may be accounted for by showing that it took several minutes for the dust to reach the fire, but there seems to be no explanation other than that already offered to account for the 5 minutes that presumably elapsed between the time Mr. Lewis and his party left the fire and the explosion. It would greatly simplify matters if it could be shown that the actual determining cause of the dust ignition was at the 5th east cross-entry.

The explosion killed all those in immediate authority, and while some little confusion naturally resulted, it was but 20 to 30 minutes until ventilation was restored in the mine. This was accomplished by placing a canvas brattice across the mouth of No. 2 mine, opening the doors between it and Nos. 1, 3, and 4, and covering the roof of the No. 3 fan. As soon as this was done a helmet party was able to enter with brattice cloth and close up the destroyed brattices along the main slope as well as to install a regulator below the 3d north entry. This forced the air up both the 3d and 4th north entries through No. 2 and out the No. 1 fan, which was speeded to its full capacity. By 6 P. M. the first rescue station was established at the upper cross-cut next to the 4th north entry on the slope, and by 6:30 P. M. the second was established at the fire which had been so dampened by extinguishers that it could be walked over.

About 6:30 P. M. the rescue car of the C. F. & I. Co., with a hastily gathered force, arrived and their trained helmet men went in. About 7, this party, noticing a light in the 5th north entry off 4th west cross-entry found four men behind a canvas brattice at the end of that entry, which they had built to keep out the firedamp, and it was by giving up his helmet to one of these men who was partly overcome, that Willis Evans, in charge of the C. F. & I. car, lost his life. Being left behind he was overlooked in the confusion, and when found was in practically a state of coma and died about 6:30 the next morning, despite the most vigorous efforts to resuscitate him. When the fire was quenched it was but a short time until the dead were reached and the bodies began to be brought out. The mine was thoroughly explored by Thursday noon and all but one of the dead definitely located. About 10:30 A. M. Wednesday 15 men came out of the 2d north mine opening who had been marooned on the Rock Island since the explosion and who had kept sufficient air in circulation by moving their jackets fan-fashion to sustain life. About 5 A. M. the morning after the explosion, the United States rescue car which had been at Golden, arrived, and rendered important aid with helmets and pulmoters. The work of restoring ventilation, by reason of there being a second fan, was simple and the rescue and recovery work after the air was restored proceeded with great rapidity.

In conclusion, while this was undoubtedly a dust explosion, the connection between it and the fire is not quite plain, by reason of the length of time from the origin of the probable impelling cause to that of the explosion.

The writer wishes to thank all the many officials of the Victor American Fuel Co. for unfailing courtesy; Jos. Boyd, the fire boss, for kindly accompanying him through the mine on many trips and for his patience in answering many questions. His thanks are especially due to J. C. Roberts, enginer in charge of United States mine rescue car No. 2; to Thos. W. Tweedale, car foreman; and to Thos. C. Harvey, first-aid miner, for many kindnesses, and for their kindly suggestions and criticisms.

TARIFF ON BRECCIA

In paragraph 112 of the existing tariff law breccia wholly or partly manufactured is put in the same class and subject to the same rate of duty as marble, onyx, and alabaster. In paragraph 111, which covers marble and onyx in blocks, rough or squared, there is no specific provision for breccia, but being expressly excepted from the provision covering building stones not dressed, hewn, or polished, subject only to 10 cents per cubic foot in paragraph 114, it would seem that Congress intended to put it in the same class with marble, especially since it is under paragraph 111, classed with and subject to the same rate of duty as marble manufactured.

There is ample evidence to show that breccia is similar in texture and uses to marble. It is crystalline and is used extensively in the form of columns, mantels, wainscoting, and general interior decoration. This is merely another perversion of geological terms by lawyers; breccia is a rock composed of angular pieces, and may be quartz breccia which has no resemblance to marble and could not be used as marble. The rock under consideration was probably "breccia marble," somewhat similar to Taconic marble.

ANSWERS TO EXAMINATION QUESTIONS

Written for Mines and Minerals, by J. T. Beard

Fire Bosses' and Hoisting Engineers' Examinations Held at Terre Haute, Indiana, August 17, 18, 1910

CORRECTION

Attention has been called to an evident error in the statement of Ques. 18, page 184, MINES AND MINERALS, October, 1910. The thickness of steel forming the boiler shell, as given in the printed copies of the examination sent us, is .045 inch. This is an evident error and should probably read .45 inch. In the solution of this question we have inadvertently omitted the factor of safety, which for mining practice should be 6. The solution of this question should be as follows:

$$d=\frac{2\,S\,t}{f\,p}=\frac{2\times 60{,}000\times .45}{6\times 150}=60 \text{ in.}$$

Then,

$$\frac{x}{.45}=\frac{22}{60};\ \text{or } x=\frac{11}{30}\,(.45)=.165 \text{ in.}$$

FIRE BOSSES' EXAMINATION*

QUES. 3.—(*a*) What is ventilation? (*b*) Why is ventilation necessary in coal mines? Points 6.

ANS.—(*a*) Ventilation in mining relates to the maintenance of a sufficient quantity of pure air so conducted throughout the mine by means of suitable doors, stoppings, brattices, and air-bridges as to render the mine atmosphere everywhere wholesome and safe. (*b*) Ventilation is necessary to insure the health and safety of the men and animals employed in the mine and the security of the mine with respect to the noxious and dangerous gases generated therein.

QUES. 4.—(*a*) Name the gases commonly met with in coal mines. (*b*) Give the symbol, chemical composition, and specific gravity of each gas. (*c*) State how each of these gases is formed. (*d*) Where is each to be sought? (*e*) What are the dangers to life of each of the gases mentioned? Points 20.

ANS.—These questions are all fully answered in reply to Ques. 4, 5, 6, and 7, page 350, MINES AND MINERALS, January, 1910.

QUES. 5.—(*a*) Describe how the different mine gases may be detected. (*b*) In what proportion in the air are they dangerous to life? Points 8.

ANS.—Marsh gas is detected in mine air by its effect in lengthening the flame of a safety lamp. When present to extent of from 2.5 to 5 or 6 per cent. in mine air, this gas produces a scarcely visible, pale-blue flame cap varying in height from ½ inch to 4 inches, according to the percentage and character of gases present and kind of lamp and illuminant used. Hydrogen sulphide is detected by its disagreeable odor. Carbon dioxide is detected by the dim burning of the lamps and their complete extinction when the gas is present in sufficient proportion. (*b*) More than 1 per cent. of marsh gas in the air of a dry and dusty mine or more than 2 per cent. in mines free from dust may be considered dangerous; but very much will depend on numerous other conditions that tend to increase or decrease the danger. One-half of 1 per cent. of carbon monoxide, or 1 per cent. hydrogen sulphide may prove fatal to life when breathed for some time. Carbon dioxide added to the air breathed becomes fatal when the gas forms 18 per cent. of the mixture, provided the oxygen of the atmosphere has not been depleted; but a much less percentage of this gas may prove fatal where a portion of the oxygen has been consumed.

QUES. 6.—Describe in full your method of examining a mine for firedamp, beginning with your preparation before entering the mine; and explain each step taken till the examination is completed and report made to the proper mine officials. Points 11.

ANS.—Clean, fill, trim, and light the lamp before going into the mine; examine the gauze and chimney carefully and observe that the lamp has been put together properly. If possible, test the security of the lamp by placing it in a box or chamber where it will be surrounded by an explosive atmosphere. Having carefully prepared the lamp, ascertain that the fan is running at its usual speed, and then enter the mine and proceed at once to the intake air-course in the district or portion of the mine in charge. Observe first that the usual quantity of air is passing at this point, and then proceed to examine carefully and in order each working place, following the course of the air throughout the district, from the intake to the return. Mark the date of the examination with chalk plainly on the coal or roof at the face of each place examined. If gas is found in quantity to endanger the workmen in any place, fence off the entrance and place a danger signal there to warn persons not to enter. If brattice at the face is down, replace it before proceeding further. Having completed the examination of the entire district, return to the shaft or mine entrance, remove the danger signal, which the fire boss should always leave there on entering the mine, enter the daily report of the examination on the book kept for that purpose, and report to the mine foreman the results of the same.

QUES. 7.—A heading 8 feet wide, in a 5-foot coal seam, is driven 50 feet past the last breakthrough. An examination shows 12 inches of gas (CH_4) at the face and tailing out to ½ inch at the inside edge of the breakthrough. (*a*) If this body of gas was thoroughly mixed with the air inside of the breakthrough what would be the explosive condition of the mixture? (*b*) What quantity of fresh air added to the mixture would render it harmless? Points 6.

ANS.—(*a*) Assuming (which, however, could never be true in practice) that the body of gas mentioned is pure marsh gas, unmixed with air or other gases, the volume of gas is $\frac{8\times 50\ (.5+12)}{2\times .12}=208\frac{1}{3}$ cubic feet. The volume of the mixture of gas and air would be equal to the volume of the entry inside of the breakthrough, or $8\times 5\times 50=2{,}000$ cubic feet. The percentage of gas in this mixture is then $208\frac{1}{3}\times 100\div 2{,}000=10.4$ per cent., or somewhat above the maximum explosive point. (*b*) The danger point will depend much on the character of the mine with respect to dust, the size of the workings, and manner of mining. In soft coal, when the coal is dry and the mine is highly inflammable, the percentage of gas in the mine air should not exceed 1 per cent. In this case, the quantity of air to be added to reduce the gas to 1 per cent. would be $208\frac{1}{3}\div .01-2{,}000=18{,}833+$ cubic feet. The conditions of this question are not practical; because, owing to diffusion, it would be impossible to have such a body of pure gas and pure air together at the head of an entry; or to measure the quantity of air required to dilute it below the danger point, whatever this may be.

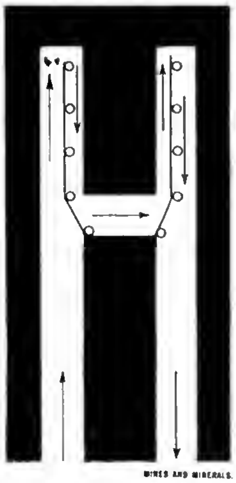


FIG. 1

QUES. 8.—Referring to the above question, state how you would proceed to remove this body of gas, and continue to advance the face till a breakthrough had been made. Points 4.

ANS.—It will be necessary to erect a temporary brattice of boards or canvas nailed to posts set in line about 2 feet from the rib. This line of brattice (Fig. 1) should start from the outby corner of the breakthrough and be extended toward the face only as fast as the gas is carried away. The intake airway must be cleared of gas first and then the return, the work always being performed on the fresh air side of the brattice.

*All miners and employes in and about mines should have a copy of the state mining laws, which they can obtain from the mine inspector's office, Indianapolis, Ind., upon request in writing.

Ques. 9.—What influence does coal dust exercise on mine explosions? Points 4.

Ans.—The fine dust of coal floating in the mine air adds fuel to the flame of a mine explosion. It intensifies and extends the explosion, which is fed by the gas distilled from the dust. As a result, an otherwise local explosion in a mine is extended throughout the entire workings by reason of the dust blown into the air by the force of the initial explosion.

Ques. 10.—If you were employed as fire boss in a mine where the coal is being blasted off the solid, the mine being very dusty and generating a small percentage of firedamp, what precautions would you advise to reduce the liability of explosion? Points 6.

Ans.—If practicable, abandon the practice of shooting off the solid and have all coal mined and, if necessary, sheared or side-cut before a shot is fired. In any case, before firing, have all shots inspected by competent men authorized and instructed to condemn any holes that would, in their judgment, be unsafe. Make and enforce with suitable penalties strict regulations in regard to the charging and firing of holes, and permit no shot to be fired till the place has been examined and pronounced safe. A system of spraying the entries, especially the haulways, and sprinkling the face before firing, should be adopted. All fine coal and slack should be loaded out and each working place regularly cleaned of such accumulations.

Ques. 11.—What dangers would you expect to encounter on entering a mine after an explosion? Points 5.

Ans.—The violence and extent of the explosion will determine the dangers to be encountered. The chief danger arises from the afterdamp that fills large portions of the workings if not the entire mine. The afterdamp consists of a variable mixture of poisonous, suffocating, and in some cases inflammable, and explosive gases. The mine atmosphere, as a result, is unbreathable and dangerous. In addition to this the roof supports have in many cases been displaced leaving the passageways and rooms blocked with heavy roof falls. Doors and stoppings have been generally destroyed and the circulation of air cut off from the workings.

Ques. 12.—What are the essential features of a safety lamp to be used (*a*) for testing purposes, (*b*) for general work? Points 10.

Ans.—(*a*) All safety lamps should be as simple in construction as their requirements will permit. For testing purposes: (1) The lamp should have a free upward circulation of air to insure the same gaseous condition within the combustion chamber as exists in the atmosphere surrounding the lamp. Otherwise the test, however accurate, does not truly indicate or gauge the gas in the mine air. (2) The lamp should be shielded, but not bonneted (except as may be required in special cases where the gas is particularly sharp or the current strong); but the upper portion of the chimney should be protected by a gauze cap, or a double gauze chimney should be used. (3) The lamp should not burn a volatile oil, owing to the danger of the lamp being extinguished and therefore requiring an igniter in the lamp; and further, because a volatile oil gives higher indications after remaining in gas a short time, owing to the effect of the gas in heating the lamp and vaporizing the oil more rapidly. (4) The lamp should be able to detect accurately as low as 1 per cent. of gas. The indication should not depend on the ability of the observer to see the flame cap. The old flame test does not meet present-day requirements. (*b*) For general use, the lamp should be (1) simple in construction to avoid arousing the curiosity of the user; (2) it should give a good light, burning a good quality of non-volatile oil; (3) should be well bonneted to protect the gauze from dust and dirt and the lamp against strong air-currents; (4) should be provided with a simple lead-plug lock, sealed, with the date stamped on the seal.

HOISTING ENGINEERS' EXAMINATION

(Selected Questions)

Ques. 2.—What do you consider the most essential qualifications of a hoisting engineer?

Ans.—He should be faithful in the performance of his duties; alert, closely observant, industrious, temperate in habits, self-reliant, and determined. He should be well informed on all that pertains to the mechanics and operation of his engine, the generation of steam, care of boilers and machinery in his charge, and master of every detail of his work.

Ques. 5.—Describe two ways of setting boilers and explain what provision is made for their expansion and contraction.

Ans.—Cylindrical flue or tubular boilers are either hung by hooks and rods from I beams that span the brick side walls, or are supported by cast-iron knees or angles riveted to the sides of the boiler. To allow for the expansion and contraction of the boiler, the rear angles rest on cast-iron rollers, which roll slightly forward and back over cast-iron plates resting on the side walls, while the angles at the front end of the boiler rest directly on the iron plate itself. In the same manner, when the boiler is hung the swing of the hooks allows sufficient freedom for expansion and contraction in the boilers.

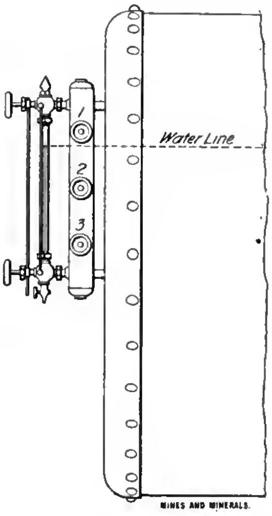


Fig. 2

Ques. 6.—Do the flues or tubes used in boilers increase or diminish their strength? Give reasons.

Ans.—The flues or tubes strengthen the boiler by the firm support they give the boiler ends. They act as rods to tie the two ends of the boiler together, and prevent their bulging outward under the great pressure of the steam.

Ques. 7.—(*a*) Where should water gauges be placed on a boiler? (*b*) Why are steam gauges necessary?

Ans.—(*a*) The water gauge should be at the front end of the boiler and may be mounted on the boiler end of a plain cylindrical boiler as shown in Fig. 2; or the connections may be made on the boiler shell of flue or return-tubular boilers. The position of the gauge must be such that the glass reaches above the high-water line and below the low-water line in the boiler. The upper end of the gauge is always connected with the steam space and the lower end with the water space and is supplied with a plug cock at each end to shut off the steam and water in case the glass is broken. In Fig. 2 the gauge is shown as mounted on a larger tube, called a water column, which in turn is fitted to the boiler in the same manner as explained above for an ordinary gauge. The purpose of this water column is to reduce the effect of the violent ebullition in the boiler to cause a wrong indication of the water level in the glass. Three try-cocks or gauge-cocks, *1*, *2*, *3* are attached to the water column to test the correctness of the water level as shown in the glass. (*b*) Steam gauges are necessary to show the steam pressure in the boiler, as a check upon the action of the safety valve, which might be stuck and refuse to open when the pressure has reached the safe limit.

Ques. 8.—(*a*) Describe three different kinds of grates with which you are familiar. (*b*) Describe the principle of the safety valve.

Ans.—(*a*) The common form of fire-grate consists of parallel cast-iron bars, made in the shape shown at (*a*), Fig. 3, the lugs cast on the sides of the bars, at the middle, and each end keep the bars a uniform distance apart. Cut (*b*) shows a form of rocking or shaking grate. The front and rear sections of the grate are operated separately by the levers shown in

front of the furnace. At (c) is shown a form of step grate for burning fine coal and slack, which is introduced into the furnace at the opening at the top of the grate, and gravitates slowly to the bottom.

(b) The most common form of safety valve is the ball and lever valve. In this valve, the upward pressure of the steam acting on the valve is transmitted by the valve stem to the lever. The moment of this pressure with respect to the fulcrum of the lever is balanced by the moment of the weight of the lever acting through its center of gravity and the moment of the ball. By sliding the ball out or in on the lever its moment is increased or decreased, and the steam pressure required to lift the valve is correspondingly increased or decreased.

QUES. 9.—The shell of a plain cylinder boiler is 30 inches in diameter and 20 feet long, and is made of single-riveted, wrought-iron boiler plate $\frac{3}{8}$ inch thick; what pressure will this boiler safely carry?

ANS.—Assuming the safe tensile strength of the iron as 8,000 pounds per square inch, the safe pressure for this boiler is $\frac{3}{8}\times 2\ (8{,}000) \div 30 = 200$ pounds per square inch. The length of the boiler is not considered in making this calculation.

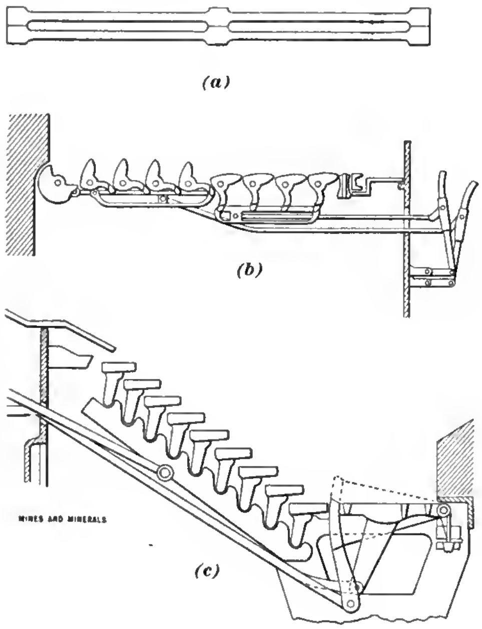


FIG. 3

QUES. 10.—(a) How is forced draft produced in a boiler furnace? (b) What determines the thickness of fire in a furnace?

ANS.—(a) Forced draft in boiler practice is produced by a steam jet, or small blower, or a blast from the air receiver of an air compressor. The greatest economy is obtained by heating the air before blowing it into the furnace. (b) The thickness of the fire is determined, in practice, by the draft, quality of the fuel, area of grate, and rate of combustion required. A forced draft increases the necessary thickness of fire, while a superior quality of fuel, large grate area, or demand for more rapid combustion decrease the same.

QUES. 11.—What should be the smallest rope used to hoist 2 tons of coal per hoist, the car, cage, and rope weighing 3,500 pounds?

ANS.—Adding $\frac{1}{10}$ of the gross load hoisted for friction, the total load on the rope is $1.1\ (4{,}000+3{,}500) = 8{,}250$ pounds. Then, assuming a six-strand, 19-wire, crucible cast-steel, hoisting rope, the required diameter of the rope for this load, using a factor of safety of 5, for a moderate depth of hoist, is

$$d = \sqrt{\frac{5\times 8{,}250}{39\times 2{,}000}} = .727, \text{ say } \tfrac{3}{4} \text{ in.}$$

QUES. 12.—An expansion joint is to be placed in a line of steam pipe that is 450 feet long; how much movement should it allow?

ANS.—The coefficient of expansion of wrought iron is .00000648 per unit of length and per degree Fahrenheit. The range of temperature to which a steam line may be exposed will depend on conditions not mentioned in the question but may be estimated as, say 400° F., which will allow for a fall of temperature below zero and a rise due to a steam pressure of 200 pounds per square inch, at sea level. This would necessitate an allowance in the expansion joint for a possible movement of $450\times 400\times .00000648 = 1.17$ feet, or say 14 inches. It would be well to be liberal and allow for a movement of say 18 inches.

QUES. 13.—(a) What different means are in use to fasten wheels and pulleys rigidly to shafts? (b) How does the reverse link of an engine produce the desired result?

ANS.—(a) Setscrews, keys, and feathers are in use, in a variety of forms and modifications adapted to different work. (b) By means of the reversing link both the direction and the amount of throw of the valve can be changed without any alteration of the eccentric or changing the length of the valve rod. As a consequence, not only is it possible to reverse the engine, but to alter the cut-off as may be desired.

QUES. 16.—What pressure per square inch is exerted on the plunger of a pump that is raising water 175 feet of vertical height?

ANS.—The static pressure due to this head is $.434\times 175 = 75.95$ pounds per square inch. When the pump is working, however, there is an additional pressure due to the friction of the flow of water in the discharge and suction pipes, which depends on the quantity of water handled and the diameter and length of the pipe. For a discharge of 400 gallons per minute through a 5-inch pipe, say 200 feet long, the friction would be practically

$$h = \frac{.01\times 200\times 400^2}{8\times 5^5} = 12.8, \text{ say 13 ft.}$$

which must be added to the actual head, making in this case, $175+13=188$ feet. The working pressure would then be $.434\times 188 = 81.59$ pounds per square inch, under the assumed conditions.

QUES. 17.—What steam pressure would be necessary to do this work if the diameter of the steam cylinder is 6 inches and that of the plunger 4 inches, making no allowance for friction or suction head?

ANS.—Ignoring the friction and assuming the discharge head is 175 feet, the pressure on the plunger when lifting this water column is, as previously found, 75.95 pounds per square inch. But in pumping, the pressure per square inch in the steam and water ends of the pump is inversely proportional to the areas, or the squares of the diameters. In other words, the pressure ratio is equal to the square of the inverse diameter ratio. Call the required steam pressure x; then,

$$\frac{x}{75.95} = \left(\frac{4}{6}\right)^2 = \left(\frac{2}{3}\right)^2 = \frac{4}{9}; \text{ and } x = 75.95\times\frac{4}{9} = 33.75 \text{ lb. per sq. in.}$$

In practice, the friction head and the suction head are both added to the discharge head and their sum multiplied by .434 to obtain the total pressure against which a double-acting pump operates. Thus, if the maximum suction head is 25 feet, the total head, for a discharge of 400 gallons per minute through a 5-inch pipe, is $175+25+13=213$ feet; and the pressure against which the pump, if double acting, operates, is then $213\times .434 = 92.44$ pounds per square inch. The required steam pressure, in this case, is

$$x = 92.44\times\tfrac{4}{9} = 41.08 \text{ lb. per sq. in.}$$

instead of that found above.

NOTE.—Ques. 18, 19, and 20 are fully answered in reply to Ques. 23, 15, and 18, pages 337 and 338, MINES AND MINERALS, February, 1909.

Ques. 21.—(a) What keeps the rod from running off the crankpin of an engine? (b) What should be the size of the steam pipe and what the size of the exhaust pipe for any size of cylinder?

Ans.—There is either a head turned on the crankpin, or a heavy washer having a diameter larger than that of the crankpin is secured to the latter by a setscrew; a dowel pin prevents the washer from turning. (b) The size of the steam pipe is usually designed for a velocity of steam, of say 6,000 feet per minute; and the exhaust pipe for a velocity of 4,000 feet per minute, the calculation being based on the volume of steam consumed, which will depend on the size of cylinder, piston speed, and cut-off.

Ques. 22.—In case the throttle valve should come loose from the steam and prevent steam from entering the valve chest, what should be done?

Ans.—Close the stop valve between the throttle and the boiler and then proceed to repair the throttle valve.

Ques. 23.—(a) What causes the wristpin in the crosshead and the crankpin to wear the way they do? (b) If the crosshead or crankpin brasses become "brass-bound" what should be done?

Ans.—(a) The reciprocating action of the crosshead causes the pressure to be thrown alternately on the two opposite sides of the wristpin each stroke of the engine, which causes this pin to wear in an oval shape. Owing however to the rotation of the crank or crank-disk the pressure due to the alternate push and pull of the driving bar is always exerted on one side of the crankpin, causing the pin to wear in an oval shape on one side only. Also, owing to a slight spring of this pin the wear is a trifle greater at the foot of the pin or close to the crank or disk than it is at the top of the pin. (b) If the brasses become "bound" or their edges touch each other owing to the wear of the pin or the brasses, they must be taken out and their edges cut or planed down.

Ques. 24.—(a) What should be done in case the eccentric slipped around on the shaft? (b) Is the principle of valve setting the same on all engines?

Ans.—(a) This question is fully answered in reply to Ques. 12, page 241, Mines and Minerals, November, 1909. (b) Yes, the principle involved in the setting of the valve is practically the same for all engines, but the method employed varies in different types of engines.

Ques. 25.—(a) How is the dead center of an engine found? (b) If it is necessary to run the engine in the opposite direction from that in which it is running, how can this be done? (c) Is the piston in the center of the cylinder when the centers of the crankpin and the crank-shaft lie in the same vertical plane?

Ans.—(a) Starting at about the middle of the forward stroke turn the flywheel till the piston is near the end of its stroke. In this position draw a fine line square across the crosshead and guide; also, before moving the engine, make a mark on the circumference of the crank-disk or on the rim of the flywheel, near the supposed dead center, by means of a scriber resting on a point of the engine frame which is marked. Now, from about the middle of the back stroke turn the flywheel backwards till the marks on the crosshead and guide again correspond; and, with the scriber in the same position on the engine bed, again mark the circumference of the crank-disk or rim of the flywheel. Bisect the distance from this mark to that previously made, and continue now to turn the flywheel till this bisecting mark reaches the scriber. In this position the engine is on dead center and a permanent mark should be made on crosshead and guide. (b) If there is no reversing link, the eccentric will have to be turned on the shaft so that it will lead or follow the crank an equal amount in the opposite direction. (c) It is not; because of the angularity of the driving bar.

New Inventions

Complete specifications and drawings for any of the following patents can be obtained from the Commissioner of Patents, Washington, D. C., at the following rates.

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
At entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED NOVEMBER 1 TO NOVEMBER 29, 1910, INCLUSIVE

No. 974,423. Coke oven, Walter Winstanley Peck, Manchester, England.

No. 974,690. Miner's lamp, Dominick Miglio, Calumet, Mich.

No. 974,632. Process of undercutting in mines, Frank Billings, Cleveland, Ohio.

No. 974,645. Mining machine, Alfred U. Davis, Lutherville, Md.

No. 974,395. Ore concentrator or separator, George Chandler Kidder, Salt Lake City, Utah.

No. 975,213. Mining measure, Anthony Cronwell, Sterling Run, Pa.

No. 975,084. Concrete timbering for tunnels, Thomas H. Stanley, Denver, Colo.

No. 975,468. Mining machine, Frank L. Sessions, Columbus, Ohio.

Nos. 976,043, 976,044, and 976,045. Treatment of precious metalliferous ores, John Collins Clancy, New York, N. Y.

No. 976,425. Apparatus for washing, grading, and classifying coal or other material, Frank D. Baker, Denver, Colo.

No. 976,785. Miner's lamp, Martin James Curtis, Billings, Mont.

No. 976,835. Mining machine-bit, Ralph E. Noble, Chicago, Ill.

No. 976,419. Ore jig, George H. Williams, Chicago, Ill.

No. 976,175. Ore-roasting furnace, John B. F. Herreshoff, New York, N. Y.

No. 976,769. Annular ore-roasting kiln, John Zellweger, St. Louis, Mo.

No. 977,119. Apparatus for handling coke, etc., William Clark, Mitchell, Sydney, Nova Scotia, Canada.

Nos. 976,931 and 976,935. Coke oven and door therefor, Frederic W. C. Schniewind, New York, N. Y.

No. 976,930. Coke oven and heating arrangement therefor, Frederic W. C. Schniewind, New York, N. Y.

No. 976,929. Burner for coke ovens, Frederic W. C. Schniewind, New York, N. Y.

No. 976,934. Coke-oven discharging apparatus, Frederic W. C. Schniewind, New York, N. Y.

No. 977,348. Regenerative coke oven, Emil Wagener, Dahlhausen-on-the-Ruhr, Germany.

No. 977,302. Waterpower mining drill, Carl A. Hanson, Seattle, Wash.

No. 977,275. Mining needle, Patrick F. Costello, Vintondale, Pa.

No. 977,087. Ore and coal washer, Henry W. Falkner, Ashland, and Franklin Schultz and John F. Wagner, Tamaqua, Pa.

No. 976,943. Ore screen, Philip R. Stanhope, Denver, Colo.

No. 977,096. Treatment of ores, Charles Morris Johnson, Avalon, Pa.

According to T. P. Bruce there is an undesirable conservation of natural resources in progress in the Western States.

"Our roads and some of our mining claims are being covered by agricultural filings. They wire up the exits and make us make a detour of 4 or 5 miles; build a shack worth \$20 to \$25, plow up 3 to 4 acres, take up a crop of less value than the seed and labor costs, get 160 acres of mineral land, where the hematite or 'bog iron' is looking at them out of the ground, while I, as I am patenting a mineral claim adjacent, have to fork out something in the neighborhood of \$165."

Between government reservations, corporation absorptions and farmers, the natural resources are aviating away from the prospector.

Mines and Minerals

Vol. XXXI—No. 7. SCRANTON, PA.—FEBRUARY, 1911—DENVER, COLO. Price, 25 Cents

CONSOLIDATED FUEL CO., UTAH

Written for Mines and Minerals, by R. J. Turner

Method of Gravity Haulage for Mine Cars With Standard Gauge Coal Cars on Same Road

The coal production of Utah has increased from 50,000 tons in 1877 to 1,967,651 tons in 1907, and even this latter tonnage did not cover the demand, as 979,152 tons were imported. Utah also produced 324,692 tons of coke in 1907. All coal mined in Utah with the exception of 151,488 tons came from Carbon County.

In 1907 the Utah Fuel Co., the Independent Coal and Coke Co., and the Union Pacific Coal Co. were about the only coal operators in Utah.

Since the construction of the Southern Utah Railroad, the Consolidated Coal Co. has opened mines at Hiawatha, Utah, 18 miles southwest from Price. These coal beds have been known long and favorably, and it was due to lack of transportation that they were not earlier developed. Located as they are 8,100 feet above sea level, and high above the present railroad, there is an abundant supply of good water and a fairly good supply of timber.

Carbon County coal beds belong geologically to the Laramie beds of the Upper Cretaceous, and while high in volatile hydrocarbons, are low in ash and sulphur.

The Consolidated Fuel Co.'s mines on Miller Creek, in the southwest edge of Carbon County, are opened on two beds of coal, one 18 feet and the other 6 feet 6 inches in thickness. These beds vary slightly in their volatile matter, ash, and sulphur contents in different parts of the field, as could be expected, the highest ash being 5.38 per cent., and the lowest 1 64 per cent. The analysis of Carbon County coal as a whole is as follows:* Moisture, 3.20; volatile matter, 45.67; fixed carbon, 47.22; ash, 3.35; sulphur, .56.

Fig. 1. Mine Mouth No. 1, Hiawatha Mine, Utah

Several features in connection with the Consolidated Coal Co.'s property make an account of the mines and the methods followed in moving the coal to the railroad cars interesting. The Consolidated Fuel Co.'s mines have been developed so rapidly that few people realize how important they have become. The Southern Utah Railroad was completed on January 19, 1910, when the first car of coal was shipped. Previous to this, little development work had been done on account of lack of transportation. At this writing the mines are capable of producing 2,000 tons of coal daily.

Owing to the steep grade where the mines are located, it was found necessary to convey the coal 2 miles from the mine to the tipple by means of a self-acting incline. This was accomplished by the usual gravity inclines, as the grade was an ideal one of from 6 to 9 per cent., but another feature had to be worked out. On account of the camp being at the mines it was found necessary to establish a standard-gauge road up to the mines so that supplies could be more economically handled. On account of the narrowness of the cañon it was decided to run a standard-gauge track between the mine-car tracks and use the same rails. This would have been feasible had the incline been a straight one, but it comprised three long curves of from 7 to 10 degrees each, and to run up the standard-gauge railroad cars meant keeping the rope in the middle of each track at the curves. This was only accomplished after a long series of trials and troubles, when a spring curve roller was finally designed that successfully fulfilled its purpose.

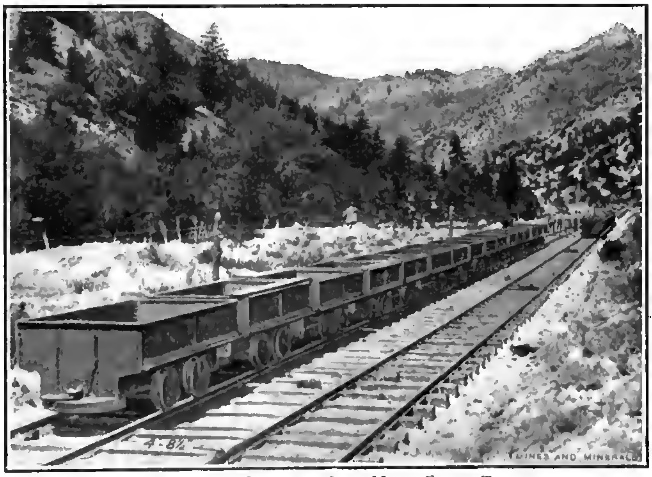


Fig. 2. Trips Passing One Mile From Tipple

The mine car adopted is of steel with Hyatt roller-bearing wheels and a capacity of 3½ tons of coal. These cars are made up in trips of 16, which in going down the plane attain an average speed of 1,000 feet per minute.

One of the two mine openings is shown in Fig. 1. It is like mine No. 2 in the lower coal seam which averages 18 feet thick, as shown by the portal.

Fig. 2 shows two trains of mine cars midway between the

* Utah Conservation Commission Report, page 86.

mine and the tipple. The curve in the background shows one of the difficulties to be negotiated in this country.

The apparatus for handling the trips on the incline consists of two 6-inch drums set tandem and coupled together with suitable gearing and friction clutch to an 82-horsepower General Electric induction motor. By means of the clutch the motor can be thrown in and out of gear at will. It is only necessary to use the motor when starting or stopping, and this only to have better control of the trip and be able to stop and start on the smallest margin possible. Each drum is equipped with two powerful brakes that have 12-inch face rims and are operated by hand by a worm-feed arrangement that gradually brings the drums and the trip to rest, thus relieving the rope and machinery of any severe strain due to the sudden stopping of the moving trip. It further gives full control of the trips as they near each end, the speed being kept down by the brakes until it corresponds to the speed of the motor, which is 400 feet per minute. The friction clutch is then thrown in and the trips all landed by the use of the motor controller, which affords a more reliable method of landing.

Fig. 3. Tipple at Hiawatha, Utah

The rope has three full turns around each drum and runs in grooves cut in the drums about 3 inches apart. Each rope runs off the lower side of the front drum, and to keep the rope in place for its respective tracks a large horizontal sheave wheel is placed to receive the rope from the drums. The rope used is a 1-inch, 6-wire, 6-strand, plow-steel, Hercules, a little more than 2 miles long. As the rope feeds on the drums and feeds off

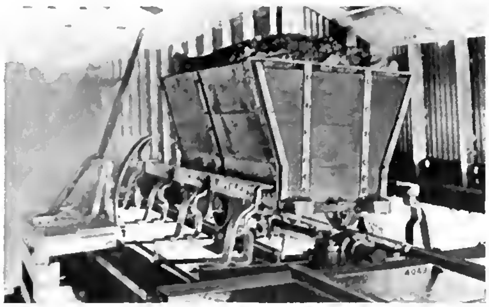


Fig. 4. Car Ready to Dump

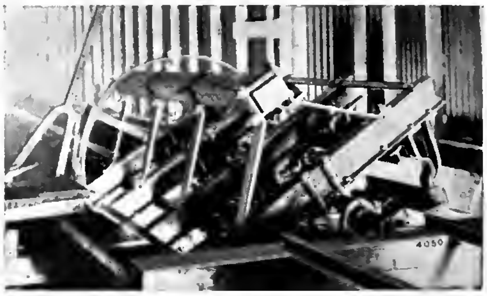


Fig. 5. Dumping

as fast, there is only a small piece of the rope on the drums at any time.

The tipple shown in Fig. 3 is shipping coal, although at the time the photograph was taken from which the cut was made the tipple was under construction. The tipple equipment was furnished by the Link-Belt Co., who also installed a rotary side dump, shown in Figs. 4 and 5, in order to do away with end doors on the coal cars. There is no question but that this was a wise move as a safety device in preventing wrecks on the plane, which are due every so often when end doors with latches are used. The mine cars discharge into a hopper shown in Fig. 3, from which the coal is fed uniformly on to the shaking screens by an automatic feeding device.

Five sizes of coal are made between the hopper and the cars, the coal all moving by gravity.

The tipple machinery is driven by a 75-horsepower General Electric induction motor. The power plant is near the tipple and comprises two direct-connected, alternating, three-phase units of 200 horsepower, carrying a voltage of 2,300. This current is carried to the mines by means of a pole line shown in Fig. 1, and there converted by means of a generating set into direct current of 250 volts, after which it is used for the various purposes of hoisting, pumping, hauling, etc., in and around the mines.

The mines are worked on the double-entry system, both to the rise and dip of the main entries, the rooms being turned from cross-entries in the usual way. Entries are driven 12 feet wide, 7 feet high, with 62-foot-thick pillars between. Rooms are driven 22 feet wide, 18 feet high, except the 30-foot room neck, which is driven entry width and height. The seam is clean throughout, has regularly defined cleavage, and has a smooth parting 7 feet from the floor, which makes an excellent roof for the entries. The roof above the coal is a heavy sandstone and no timber whatever is required in rooms or entries. The main dip of the seam is 2 per cent. in a southwesterly direction, consequently the haulage is done by electric locomotives. Two 12-ton Goodman locomotives, one in each mine, serve the purpose of hauling from the level-entry partings to the tramway, and one 5-ton Goodman gathering locomotive gathers all the coal cars in No. 2 mine. The coal cars in No. 1 mine are at present gathered by horses, which, on account of the light grades, is easily accomplished.

Ventilation is carried on by the aid of two Jeffrey reversible fans driven by electric motors. The ventilating current is split at each pair of cross-entries, and by means of an overcast each pair of entries have their own individual system of ventilation. All breakthroughs between entries are closed with sandstone masonry laid in cement, the sandstone for the purpose being brought into the mine. This arrangement forms a permanent

and effectual air stop which under ordinary circumstances should last so long as the mine.

As yet the mines give off no firedamp, but ample precaution has been taken in planning the workings to assure perfect safety if at any future time they should do so. An electric shooting system is being installed similar to the one used by the Utah Fuel Co., explained in these columns some months ago. A perfect sprinkling system is in operation to allay dust and make these mines as safe to work in as any in the state.

STEEL SUPPORTS IN COAL MINES

*Written for Mines and Minerals, by R. B. Wooodworth, M. Am. Soc. C. E.**

Durability of Steel Supports. Different Styles Used in Various Coal Mines

Within recent years a number of coal-mining companies have adopted the use of steel for supporting those underground excavations which are to remain open until the extraction of the coal is completed and the mine abandoned. Experience gained in the actual designing and use of steel mine timbers and steel shaft supports has convinced engineers who were once skeptical as to the use of steel within the mines, and has corroborated the correctness of the general conclusions advanced more than three years ago by the writer in the proposition that for permanent work steel is superior to wood in supporting underground excavations and is its logical successor.

Attention has frequently been called to the increasing cost of timbering mine excavations with wood, which is due to the rapidly decreasing supply of timber suitable for this purpose. For the past 40 years engineers have urged the importance of the antiseptic treatment of the timber which goes into the mines in order to prevent its destruction by disease. The suggestion, however, has gone almost unheeded, as only one or two companies in the anthracite region have adopted timber preservative methods on anything more than an experimental

FIG. 1. STEEL MINE TIMBERS, ROBY COAL CO. DRIFT NO. 2, ADENA, O.

stage. Timbers treated with tar products, though they may be immune from disease, are not increased in strength or made fireproof. In consequence, the logical engineer familiar with the real values of materials of construction turns to steel wherever permanence in mine supports is required.

The suitability of steel for supporting roofs in coal mines was demonstrated in England and in Europe years ago, and numbers of installations have been satisfactorily executed in the United States, some of them of many years standing, so that the adoption of steel for such purposes in this country has been merely a question of cost.

The substitution of steel for wood is simply the substitution of a material known to have great strength and longevity for a similar material of less strength and unknown endurance. The

FIG. 2. STEEL MINE TIMBERS, FLORENCE MINE, Y. & O. COAL CO., MARTIN'S FERRY, O.

forms of wood in timber sets that have proven suitable for mine timbering can exactly be duplicated in steel; a set of steel entry supports can be alternated with wooden sets; steel shaft frames may be alternated with wooden frames, or wooden timbering may be replaced by steel timbering, and vice versa. Wherever tensile and bending stresses are to be resisted, wood and steel find their true field of usefulness, with this difference: that the substitution of steel for wood avoids those elements of economic waste to which wood is liable, namely, the waste in sizes of timbers due to practical considerations which make legs and collars of the same cross-sectional area or diameter, waste in framing, and waste from destruction by decay.

The final collapse of mine timbering may be due to several causes:

1. Long continued stresses which may cause a permanent set near the elastic limit of the material so that slight increases in weight cause rupture.

2. Unequal distribution of the load due to the fracture of the strata above, unexpected side pressure or imperfect jointing and chocking.

3. Riding of the strata which produces a squeeze that nothing so far attempted has stopped.

4. Absolute weight of the overlying strata whose pressure is greater than the timber can sustain.

5. In the case of wood, final destruction of material of sufficient strength when first installed may be hastened by various diseases to which mine timber is subject if not antiseptically treated.

The load to which the timbering may be subjected in any particular mine varies with the pitch of the strata, the thickness of the cover, the density of the rock, and other causes, the total effect of which is hard to reduce to calculation. Experience in the use of mine timbers will alone determine what sizes should be used. The comparative values of the various species of timber used in mine supports to resist bending or compressive stresses may be obtained from the pamphlet entitled "Steel Mine Timbers," issued by the Carnegie Steel Co. and prepared by the present writer, and so also may the corresponding sizes of steel section. In the absence of such tables the strength may very readily be computed by the following simple methods:

1. *Rectangular Beams.*—Multiply the width by the square

* Engineer with Carnegie Steel Co.

of the height and divide the result by the length in feet. The quotient multiplied by 133, 89, 111, or 67, will be the uniformly distributed load in pounds for yellow pine, white pine, or spruce, white oak or hemlock, respectively.

2. *Round Beams.*—Divide the cube of the diameter by ten times the length and multiply the quotient by the same constants as for rectangular beams to obtain the uniformly distributed load.

3. The safe uniformly distributed load in tons on standard steel sections may be obtained by dividing the following constants by the length in feet: 15-inch 42-pound beam, 314; 12-inch 31.5-pound beam, 192; 10-inch 25-pound beam, 130; 9-inch 21-pound beam, 101; 8-inch 18-pound beam, 76; 7-inch 15-pound beam, 55; 6-inch 12.25-pound beam, 39; 5-inch 9.75-pound beam, 26.

4. The computation of loads in compression on the legs of 3-piece timber sets or the buntons of shaft sets is not so simple, as the strength of the material, whether iron or steel, decreases as the ratio of the length to the width (ratio of slenderness) increases; consequently a simple easily remembered formula cannot be put down. The general formula for the compressive unit stress in pounds per square inch of oak is

Fig. 3. H-Section Roof Supports Adrian Mines, Rochester & Pittsburg Coal and Iron Co., Punxsutawney, Pa.

$900 - 17\frac{L}{D}$, long-leaf yellow pine $1{,}000 - 18\frac{L}{D}$, and white pine or spruce $800 - 15\frac{L}{D}$, where L is the unsupported length of the leg in inches and D is the least diameter or width in inches. The steel **H** beams, which are the best fitted for use in mine timbering to resist compressive stresses, are generally employed where the length of the member is not more than 22 times its width. In such cases the safe load in tons may be obtained by multiplying the weight per foot of the section by 1.75. Where the length of the member exceeds this value, reference must be had to the general formulas.

So far as known to the writer, single props of steel have not been used in the United States, and where props remain in position until broken by the subsidence of the roof, there seems to be no economy in endeavoring to make a substitution of steel for the cheap wooden props now used in mining operations. With the introduction of longwall methods of mining, such as are more common in England and on the Continent, and to some extent in the middle Western States, with the consequent withdrawal and reuse of the props, it would seem that in such workings metallic props could be used in this country with as much economy and satisfaction as in European mines.

The numerous methods of placing steel mine timbers which experience has shown to be best suited for existing roof pressures are given in the following illustrations descriptive of typical installations:

The simplest use of single steel supports in mines is shown in Fig. 1 in the **I** beams spanning the roadway. The excavation is the Roby Coal Co.'s No. 1 drift at Adena, Ohio. "These mines have a very tender roof and at one time the haulways were considered the most dangerous in the district. Conditions were so uncertain that those in charge could never tell when they started in the morning the length of time the mine would be in operation, as a fall was liable to occur any time, suspending operations for the day. Under the able management of the present manager and his assistants conditions have changed, and these mines will now compare favorably with any in the district. At No. 1 drift the main entry is timbered with 833 steel **I** beams and 398 8″×10″ oak timbers; south face, 1,140 steel **I** beams and 594 8″×10″ oak timbers; north face, 160 steel **I** beams and 40 8″×10″ oak timbers, making a total in this mine of 2,133 steel **I** beams and 1,032 8″×10″ oak timbers. No. 2 drift, north and main entry, 818 steel **I** beams and 733 8″×10″ oak timbers; first south, 607 steel **I** beams and 198 8″×10″ oak timbers; second south, 65 steel **I** beams and 12 8″×10″ oak timbers; total, 1,490 steel **I** beams and 943 8″×10″ oak timbers. The management informs me that the cost of production is a great deal less than when they attempted to operate

Fig. 4. Three-Piece Steel Set Used by Susquehanna Coal Co.

without properly timbering the haulways and taking the risk of no one being there when a fall occurred." (35th Annual Report Chief Inspector of Mines, State of Ohio, page 430.)

Owing to the small bearing and lack of compressibility in steel it is customary in cases of this kind to rest the ends of the steel beams on wood. In breakthroughs or cross-roads the roof supports rest on **I** beams carried by short timbers, as shown to the left in the illustration.

The method of supporting a roadway through a territory where one side has been robbed and abandoned while the other side is a solid rib of coal is shown in Fig. 7. All pressure in this case is practically downward, hence the wooden legs are vertical while the steel beams are horizontal. In the mines of the Alden Coal Co., at Nanticoke, Pa., there are extensive installations of this kind where the 15-inch **I** beams used for collars rest at one end on the rib and at the other on a wooden post notched out to take the flange of the beam. In the mines of the Youghiogheny & Ohio Coal Co. unmortised wooden posts are used successfully as shown. At the Amsterdam Mine, 1,086 steel beams have been placed for roof supports from the first of June, 1908, up to December 31, 1909.

Another method of using one steel and one wooden support in a wide excavation is shown in Fig. 2, Florence Mine, Youghiogheny & Ohio Coal Co., where a branch road turns off to another portion of the mine. In this situation props are placed under the steel beam at the junction of the road, but out of the way. When a prop is so placed under a beam, it reduces the span

and thus materially aids in sustaining the pressure from a bad roof.

Where 3-stick supports are required, pressure may come on them from all three sides, or the sides of the excavation may be soft and might crumble under the weight of the collar. In the Adrian mines of the Rochester and Pittsburg Coal and Iron Co., at Punxsutawney, Pa., 4-inch **H** beams weighing

Fig. 5. Steel Gangway Supports, Allport Coal Co., Barnesboro, Pa.

13.6 pounds per foot replaced as many 8-inch square collars. These **H** beam collars were the first of their size rolled in the United States and were spaced 4½-foot centers, their ends resting on round wooden legs, as shown in Fig. 3. In the illustration a wedge will be observed driven between the top of the leg and the beam. This protects the leg by preventing splitting or fiber brooming, when the rock weight settles on the beam. It is of course as necessary to chock steel beams as wooden collars against the roof to prevent unequal distribution of load and bending at the ends over the posts.

The use of steel collars for roof supports in connection with wooden legs is strictly in line with good theory and practice, steel being relatively better adapted to resist bending stresses under transverse loads, while the compressive strength of wood is very much greater than its resistance to bending.

In Fig. 4 is shown the 3-piece all-steel set which was designed by Mr. R. V. Norris and installed in many collieries of the Susquehanna Coal Co. and its subsidiaries. These sets are spaced on centers by rods and gas pipe placed between them at the upper corners and at the sides. At the same time the arrangement prevents the sides from moving out of alinement. It will be noticed also that the sets are backed with lagging, which may be either plank or concrete slabs or old rails. The I-beam collar in this set rests on two pins to separate the two steel channels that go to make up the legs and which are also connected together by short pieces of gas pipe through which a bolt is passed and fastened by nuts on the outside.

In Fig. 5 is shown the main heading of the Allport Coal Co.'s mine at Barnesboro, Pa. There are 85 sets in this mine similar to those shown. Each set weighs 372 pounds, or about one-third the weight of the necessary 3-stick wooden set. These sets were lagged with wood and concreted up to car height. The installation was conducted by ordinary mine workers and complete sets erected in 8 minutes each by the watch. The construction is fireproof, fool-proof, and germ-proof.

Up to this point comparatively narrow single-track excavations have been shown, but the economy of steel as compared with wood increases much more rapidly than the length of the span. In Fig. 6, Oak Hill colliery, Minersville, Pa., there are two tracks with heavy steel roof beams supported at the ends by steel legs without central posts, thus making a longer span than was safe or economical with wooden beams. In the rear of the steel sets are shown the heavy 3-stick sets with intermediate posts to stiffen the collars, and the illustration shows very clearly the relative mass of the wood and its additional area of surface exposed to fire. The appearance, irrespective of the longer life and increased safety of the steel, is worth almost the difference in cost between the two.

The first use of steel in any form within the mines was doubtless the use of old rails at points of local danger. The scientific use of steel dates from about the year 1875 when I-beam shaped girders were used to support the roof at the bottom of the shaft in the Cambois colliery in Northumberland, England, in order that the great loads might be carried in the same way as had been possible theretofore with timber. For the collar of a square timber set there is no form of section rolled more economical than the standard **I** beam used in ordinary construction. For the legs and props, where loads have to be carried in compression, the highly economical integral section was not rolled in the United States four years ago. The **H** beam, which is the closest practical approximation in the form of a rolled shape to the section of the largest compressive resistance in proportion to its weight, has been found the section best fitted for use both as single props and for legs of the 3-piece gangway support, and its introduction can be said to mark an era in the history of American mine timbering as it did when introduced in England by the Darlington Iron and Steel Co., in 1885. The use of this section has been extended to include the framing of mine shafts. When used for wall plates of a mine shaft the **H** beam has the advantage of a large bearing surface as compared with beams or channels of the same depth, and for buntons and compartment separators the advantage of a large compressive strength combined with a comparatively high moment of resistance against bending.

The experience of three years indicates that a steel gangway support set need not be any more adjustable than the 3-piece set as framed in wood. A few adjustable sets were installed in one of the copper mines of the far West where the ground is heavy and the conditions severe. These sets failed under service, largely because of errors made in the substitution of steel for wood, the legs not being sufficiently heavy for the service they might reasonably have been expected to perform. It is significant that this is the only failure which has occurred

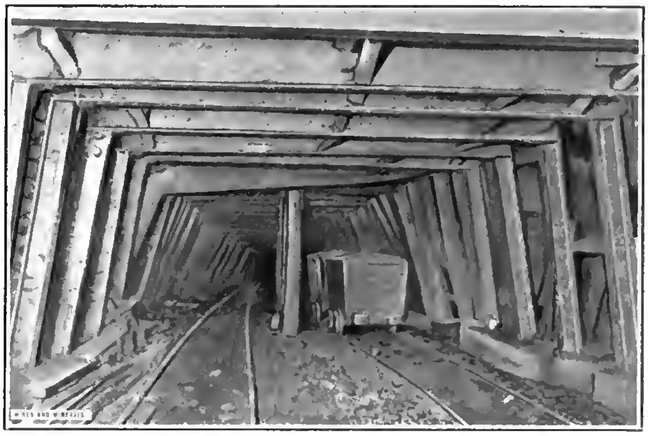


Fig. 6. Steel Gangway Support, Oak Hill Colliery, Minersville, Pa.

in three years; all the other installations, which have been quite numerous, have been entirely successful and satisfactory.

The use of steel in place of wood will not solve all the mine timbering problems. There are conditions encountered in mining with which the most careful foresight cannot entirely cope. The removal of the coal over wide areas necessarily entails an adjustment of the overlying strata to meet the new conditions, however well the timbering may be put in place.

The settlement of the superincumbent strata in level workings brings upon the timbers nothing but a direct bending or compression. When the strata are inclined, however, as in the West and in the anthracite regions, this movement causes squeezes and other disturbances of a similar character, which bring upon the timbering compression stresses in the horizontal members and bending stresses in the vertical or inclined members, together with crushing stresses on all, producing inevitable distortion and permanent deformation. Under such conditions steel will sustain its position better than wood, for the reason that its strength remains unimpaired over long periods of years unless its elastic limit is exceeded, whereas wood is continually deteriorating under such repeated stress and the influence of moisture and temperature conditions.

A careful series of experiments has been instituted to determine the extent of corrosion that may be expected and

Fig. 7. Steel Mine Timbers, Barton Mine, No. 3 Entry, Y. & O. Coal Co., Barton, Ohio

the best materials for the preservation of the steel against such corrosion. Samples of mine water were obtained from 24 mines in the bituminous and anthracite districts. They were carefully analyzed for total solids and for free sulphuric acid, and these analyses compared with a number of analyses made by coal companies and by the United States Geological Survey. The analyses showed that the acidity of mine water had been very much overstated, the worst sample showing only 3,320 parts of free acid in a million or $\frac{33}{100}$ of 1 per cent. It was found by further investigation that the worst condition is probably that in which a small percentage of ferrous sulphate is found in a water containing free sulphuric acid. On the basis of these investigations a series of laboratory experiments were conducted by immersing $\frac{1}{2}$-inch round rods painted with 50 various pigments in 120 combinations as to first and second coats in a standard solution containing 2.6 per cent. of free sulphuric acid and 4 per cent. of ferrous sulphate, or approximately 10 times worse than the worst sample of mine water. Further details of this investigation may be found in Cushman and Gardner's work entitled "The Corrosion and Preservation of Iron and Steel," page 224. These experiments were continued for 163 days, the results tabulated and compared and the pigments rated in accordance with their endurance under test and the condition of the material at the end of the period. These experiments indicate that only the simplest means are necessary for the absolute guarantee of an extremely long life for steel in underground mining conditions.

The steel should be painted before it is placed in service. If the paint is well chosen and applied with care, there need not be any fears as to its durability or as to its protective value. The first coat applied at the shop should consist of a practically inhibitive pigment to prevent the inception of corrosion in the steel, and the second coat should be put on in the field to protect the first from atmospheric and temperature conditions, to fill up thoroughly any voids which may occur therein and to cover surfaces abraded in shipment. To meet these requirements a good grade of red lead or a natural oxide of iron well applied will be sufficient for the first coat. For the second or excluding coat there seems to be nothing better today than a first-class graphite although the natural hydrocarbons, such as elaterite and asphaltum, have given excellent results. If used, these should be free from sulphur or other impurities.

After all, underground conditions are not nearly so severe on the steel as those above ground. Above ground the paint coatings are subjected to alternating conditions of high and low temperatures, various degrees of dryness and wetness, alternation exposures to light and darkness, abrasive action of winds, dust, etc., all of which are accelerative in their deterioration. Underground these conditions do not apply, and the coating is not exposed to any such severe service, consequently any first-class coating should show a high degree of endurance and insure ample protection. The fact that steel has been exposed 15 or more years in the anthracite region without renewal of the original preservative coating and without serious corrosion is a demonstration of the ease with which the life of steel may be indefinitely prolonged.

PERSONALS

Peter Haberstick, manager of the Elks Club, who planned the stage setting illustrative of a coal mine, and Arthur Langhan's florist, who did the decorating at the recent West Virginia Mining Institute meeting, in Wheeling, say that the members are jolly people and that they hope to see them again in Wheeling in 1913.

C. S. Herzig and W. Murdoch Wiley have resigned as president and vice-president, respectively, of the Constant-Herzig Co. C. S. Herzig has taken over the mining contracts and will continue the business heretofore carried on by the mining department of the company.

J. B. Tyrrell, Mining Engineer, of Toronto, has opened a branch office in Porcupine, in Northern Ontario.

Announcement is made by Charles G. Armstrong, the consulting engineer of New York City, that he has taken into partnership his son, Francis J. Armstrong, under the firm name of Charles G. Armstrong & Son. Francis J. Armstrong is a graduate of the Manual Training School of the Chicago University and of the Stevens Institute of Technology. He has been engaged with his father in consulting engineering work for the past year.

N. H. Darton, Geologist with the United States Geological Survey, has been appointed geologist of the United States Bureau of Mines, with headquarters at Washington, D. C. He states that he will continue to investigate the geological conditions under which explosive gases occur in coal beds.

Harry N. Taylor, President of the Wilmington Coal Co., has been elected president of the Illinois Coal Operators' Association.

W. A. Lathrope, President of the Lehigh Coal and Navigation Co., has been made a director in the Lehigh and Hudson Railroad.

Doctor Harold Osborn, of the Massachusetts Institute of Technology, was the first to receive the degree of Doctor of Engineering from the American Institute of Electrical Engineers.

Edward G. Locke has been appointed general manager of the Davis Coal and Coke Co., and the Western Maryland Railroad.

Richard Maize, who has been superintendent for the Pittsburg and Westmoreland Coal Co., at Hazel Kirk, has been appointed inspector for the First Bituminous Mining District of Pennsylvania, to temporarily replace Alexander McCanch who is ill.

THE MORAL OF STARKVILLE

Written for Mines and Minerals, by Sim Reynolds, Mine Superintendent

Questions Which All Having Responsible Positions at Mines Should Ask Themselves

During the night shift of October 8, 1910, it was the unenviable opportunity of the Starkville Mine, near Trinidad, Colo., to add to the long list of major mine explosions of late years, and, further, to give additional evidence that the number of combinations which can, and do, cause disaster in mines is not yet exhausted. Let us hope for our future's sake they are at least limited, and that the end will soon be reached.

This particular "accident" was a distinct surprise to a goodly number of people, although to one fully conversant with the results at Monongah it should not have been such. Both explosions were practically identical as to primary causes, differing slightly, of course, in detail. Evidently the surprise was greatest where it happened, among the managerial force of the Starkville Mine. This is certain; it was one of the few large explosions which was apparently totally unexpected. There have been disasters on a large scale heretofore that have been but confirmation of such rumor on the part of the employes and others, although not always arriving on schedule time. And among this class Pennsylvania and West Virginia have had a large share.

That at Starkville falls at once into a different category, and therein is the only feeble extenuation that we can see. To the writer the very idea of it at once conjures Cherry and Monongah. Neither was considered a dangerous mine, by which term we mean having within its operation and the winning of its coal bed possible death at one time for a large majority of its employes. Yet both were the scenes of manifold death and destruction of property. In spite of these obvious lessons, there are running today, as before these disasters happened, scores of just such "non-dangerous mines." Many of these latter have within them potential possibilities even graver than the holocausts mentioned. Many of them are today working under the exact duplicate of conditions at Starkville before the explosion. Many of them do use powder, dynamite, or other explosives of concentrated flame-producing power. Starkville even didn't have that, and, besides almost absolute freedom from gas, there was not even the usual dramatic setting of "a careless miner and a blown-out shot." The combination at the Colorado mine was contemptuously simple—merely a lot of black carbon settlings and an electric wire! It would seem as if our country were chosen of the gods on which to wreak undeserved vengeance when even this was chosen as a means of disaster, if—the same cause had not wrought the same result in France, in Belgium, in the little Staffordshire slaughter pits, and elsewhere, both since coal dust has attained to its modern dignity as an influence on mining mortality, and before men recognized its candidacy for a star role in this oft-recurring drama of the underground. But has it filled the mine official still operating under conditions similar to those that existed at Starkville with sufficient caution? I am afraid not. This happened at the other fellow's mine. His dust was undeniably inflammable, else the explosion would assuredly not have happened. But ours isn't! At least that's the way the continued methods seem to presuppose.

Fortunately in the Starkville case all the undeniable testimony did not die with the men, as is too often the case. From certain employes' evidence a vivid light is thrown on the utter disregard of danger characterizing that mine's operation previous to the blow-up. The superintendent knew the mine was very dusty, according to his own confession, but didn't consider it dangerously so until after it happened. He admits—now—that the mine was insufficiently watered. Doubtless he is also of the opinion shared by many other mining men that to allow the underlings having neither technical nor other knowledge of coal dust—except that it is a nuisance and black—to assume sole discretion regarding which part of the mine should be watered and which not, how much this part should get and how much that, was "going some" in the matter of managerial indifference. And all the mine superintendents and foremen do not live in Colorado nor at Starkville who would perforce have to admit guilt in this premise if cornered.

In view of recent events it seems to the casual observer that, scattered about this country, there are a considerable number of men in charge of mines unfitted by temperament for the positions they hold as guardians over human life. Neither is the indifferent temperament in a mine manager a good trait if one considers only the chances of property destruction, unless his employer's forethought has secured ample provision for anything that might turn up. Not because of insufficient practical or technical mining experience, mind you, for many of the most callously indifferent men have been raised in and about the mines. But there is lacking in their mentality that something fools define as cowardice, or fussiness, and wise men as philosophic wisdom. They lack that which is absolutely essential to the best management in all cases where human lives are the stake, an ever-present feeling of fear of what might happen; not largely, of course, but in that quantity of the golden mean which would insure their care regarding details as well as the large things; in short, a man who constantly labors under the impression that it is better policy, both from his own point of view, and from that pertaining to the company employing him, and for the not less vital ethical consideration involved in the lives in his care every day, to spend $10,000 to prevent an explosion than $50,000 in rehabilitation and law suits for damages. This of course does not preclude the possibility of his being efficient in other qualifications necessary according to circumstances and opportunity. And among the many mining men I have known they who have developed that instinct of wholesome caution were the bravest when it came to a pinch at the other fellow's mine. That's a fact regarding which I could fill many pages of Mines and Minerals with corroborative stories.

Of course I do not know the men actively engaged at all these mines which have blown up, and any expression made in this respect is general rather than specific. I refer to the mining industry as a whole. But what I decidedly want to convey to you fellows who are indifferent even now, after Monongah and Starkville, you men whose mines are dusty and have not much gas or none at all, but in which there are other means of communicating intense flame, is that a new era of management well besprinkled with plenty of water and a considerable quantity of caution and care of the little things, need not be defined by you wrongfully. Rather we would have you define it as essential precaution. You are doubtless a hustler or you wouldn't long hold your job; you are here and there and everywhere in a day. Doubtless you are fairly well educated, both generally and in mining. You have had all the practical experience necessary to make you proficient and an all-around mining man. You understand the possibilities that exist in all coal dust and gas, singly or individually, but—these things have always caused trouble at the other fellow's mine. You are the last to believe that your plant is prime for such an happening. Had this instinct of precaution been sufficiently developed, explosions would not have occurred. The coal dust accumulations of many years along the miles of roadway would have been thoroughly wetted and loaded out on Sundays and on idle days or nights. A hose would have washed down those trillions of carbon particles from their lodging places along the timber tops and the rib walls. To have obviated the necessity of having to take time to install new fans in event anything did happen, the ventilators would have been so situated that the worst explosion could not have destroyed them nor their operating apparatus. They would have been in operation in 10 or 15 minutes at most after the blow, pumping the fresh air so essential to rescue. Also, being so vital a part of the mine

machinery, they would have received their power direct, and not have been dependent on all kinds of contingencies along the main line of wiring. Had the superintendents been the logical men I assume them to be now, they could readily have convinced themselves and their companies that a few hundred dollars expended along those lines stood every possible chance of saving the companies many many thousands some unlucky day, and, should everything continue to run without accident of the character under discussion, the additional security and smooth-running qualities would amply repay the slight investment. That is logic that never fails to move any "tight wad" holding the purse of a coal or any other corporation and convinces him that each stitch put in the corporate cloth today stands to save nine tomorrow. If it does not, my advice is to hang on to your reasoning and find a new employer. Otherwise he's just as liable to leave you, as well as your men, wishing you had, some day you are knocking about in the mine. And if your own ideas calmly expressed fail to move things in your direction, gather, as your last card, the overwhelming mass of evidence along this line which has appeared in MINES AND MINERALS during the last 10 years; evidence which is as incontrovertible as that the earth is round, that such mines as yours do explode under the certain circumstances which it is your desire to forestall. If he still resists your effort to make the necessary changes it's up to you to make the next move, and meanwhile pity the blindness of some men whom fortune has given more money than brains. I absolutely refuse to believe it is the wilful, deliberate purpose of any mining man to toy with human life, even when it is of the lowest grade of labor. I cannot conceive of a man in his right senses deliberately placing himself before God, even if he manages to hoodwink and escape his state's law, as a predetermined slaughterer of his kind in cool calculation involving only dollars and cents. There is a vast difference between the burglar who is suddenly confronted with the man he is endeavoring to rob, and, fearing the other might shoot first, plugs him with a bit of lead, and the man who sits in a quiet office and does his killing. If the latter is obdurate he is blinded, let us hope, by the nearness of danger. To get a clear perspective of anything one must endeavor for a sufficiently long time to get away from it. And the effort is surely worth while if it removes from your life or from mine the possibilities of a life of regret and self-condemnation that no power on earth can remove from us once it shall be accomplished.

In mining, as in the broader affairs of men, there are more sins of omission than commission, but unfortunately they are just as potent in filling a graveyard and creating widespread poverty among the innocent. The fact that your mine gobs, your roadways, your timbers, like those at Starkville, have been covered with coal dust unremoved for upwards of a generation without incident does not preclude the possibility of it happening this month or next, this year or the year following. Yet that is a very common line of reasoning indulged in by many of us when we wish to delude ourselves in a false belief of safety. A very illogical method, I grant, not to say criminal, yet, again, pitifully commonplace. If you don't believe it after what has happened in the last decade from Mexico to British Columbia, from Pennsylvania to Utah, why you are beyond conviction, and this article is not for your kind.

And in closing let us examine a phase of the methods used at Starkville, not because of any suggestion we can make regarding the Colorado mine, not because of any criticism or condemnation for any of the officials concerned therewith, but because of the possibility that there is still at your mine, amenable to changing before anything really bad comes of it, just such a piece of misjudgment.

It is said the machinery driving the fan drew its power from the same lines as those furnishing power to the haulage motors. Here, in trying to kill two birds with one stone (a bit of economy, by the way, which can, and often does, result in more harm than good about mines) those in charge stood to have the men in the mine go to their death in several ways contingent on several kinds of accident liable at all large mines. Not only was the fan in each instance placed where any unusual force exerted on it from the interior could (and did) demolish it, but they trusted to everything going right along many miles of haulway to keep the ventilation going. A mine fire, small in itself, at any one of a thousand places in those vast ramifications could have been diverted by changing the current while efforts were made to remove the men from the interior, if all went well along the haulways. But, assuming it occurred, as it most likely would, along the main roads, and off went the ventilating power, the air in the mine at once would become stagnant, and the men in the workings perish. But, taking the same instance of possibility, and granting that previously there had been a little trouble and expense to carry the wires from the power house to those fans somewhere above or below ground, if in the latter case it were not along the haulways, or if so, so guarded as to be safe from ordinary short-circuiting and other accidents, and the last thing that could be done to safeguard the men's lives would have been done. Verily, as it was, in a mine of Starkville's character the management played against fate with long odds against them. And while they did not lose by fire, they lost in the other way. Obviously the officials there had not retained in their memories the lesson all learn in boyhood, that if one continues to play with fire sooner or later he is pretty sure to get burned. I'll warrant there's a better arrangement of the ventilating apparatus at the Starkville plant now, and a direct application of power. But why not before? Surely the management saw the possibilities of such an arrangement! Seems so obvious to the onlooker that one would not be surprised to hear of the trappers in a similar case making application to the state inspector for a change in methods or a change in heads, or refusing to work until the pernicious possibilities were removed. Surely men calling themselves superintendents, managers, foremen, or what not, could see these simple stumbling blocks which at any time might trip them, and at one blow undo all they had done in other directions. Surely their mining experience was sufficient to see beyond mere temporary necessity! But was it? Is yours? If they did why didn't they remove the difficulty? Why don't you remove yours?

What happened at Starkville tempts us to the magnanimity of believing they didn't, or this story I have just read of death in that Colorado mine would never have had to be written.

Do we need point the moral?

OUTBURSTS OF CARBON DIOXIDE IN FRENCH MINES

J. Loiret, of Clermond-Ferrand, read a paper before the International Mining Congress at Dusseldorf, on "Sudden Outbursts of Carbon Dioxide in Mines of the Central Plateau of France," and the dangers arising from this gas. Isolated cases had been known since 1856, but during the last 5 years the danger had much increased. The conditions which favored the outbursts were not understood, and outbursts could not be predicted. To guard against them, the use of the pick was forbidden, and blasting should be done by electricity from a safe distance. In a second paper M. Loiret referred to an interesting case in which the provision of a safety chamber, such as is used especially in Austria, in the collieries of the Kaiser Franz Nordbahn, had done good service. The chamber was provided in a cul de sac, which was entered through two doors, fixed about 9 feet apart; between these doors was the end of a special ventilator feed-pipe. It was the rule that the blasting should be directed from this chamber. The men did not observe this precaution, however, and were overtaken by a sudden rush of carbon dioxide. Two men fell down, so that the first door could not be closed; three others died, but five kept themselves alive by breathing the compressed air until they were rescued 10 hours later.

PRINCIPLES OF ELECTRIC BLASTING

By W. G. Hudson, M. D.

In Fig. 1 is shown an ordinary dry-cell battery, with two binding post connectors, *A B*. These are called, respectively, the positive + and negative − poles.

Elementary Electrical Principles as Applied to Electric Blasting

If the poles are connected with a piece of wire *C* an electric current will flow through it from *A* to *B*, and will continue to flow until the chemicals in the battery are exhausted.

The wire forms an unbroken circuit along which the current flows from the negative to the positive pole of the battery The current only flows when the circuit is complete; and a break in the wire stops the flow.

Non-conductors are those substances which do not carry the electric current. Conductors are those substances which carry it readily; poor conductors are those which carry it, but with difficulty. Strictly speaking, there are no perfect non-conductors, and no perfect conductors; but the terms are in common use, and are convenient and unobjectionable if it is borne in mind that they are relative. Metals are the best conductors; silver heads the list, followed in turn by copper, zinc, iron, platinum, lead, mercury, etc. Among the best non-conductors are glass, rubber, sulphur, silk, cotton, paraffin, tar, resinous materials, oils, etc. Water is a representative of poor

Fig. 1

Fig. 2

Fig. 3

conductors, but its conductivity is greatly increased when various salts, such as those likely to be derived from rocks in drilling, are dissolved in it. Acids also increase the conductivity of water.

The wire comprising the circuit can be much longer than shown in Fig. 1—even many miles in length—and still the current will follow it throughout its entire length, so long as some shorter or easier path between the two poles is not offered. If the wire is covered with some insulating material like silk, rubber, or cotton, so as to prevent the current from escaping from it and following some shorter or easier course, then the wire may be wound many times around other objects, or make any number of bends and twists, and still the current will follow it from one pole of the battery to the other, with almost as much ease as it did the short piece of wire.

Now, how is it known that a current is flowing through the wire in the manner described? It is known by its effects, and a few of these, which are of importance in understanding blasting by electricity, will answer for the present consideration.

First, if part of the ordinary thick copper wire used in Fig. 1, be replaced by a very fine piece of wire *G*, Fig. 2, the fine wire being preferably of iron, platinum, or German silver, then the difficulty which the current has in passing through this small piece of wire, or as electricians say "overcoming its resistance," will transform part of the current into heat. The fine wire will become red hot, and even melt if the current is strong. This is the principle made use of in firing electric fuses. Another familiar application is the incandescent electric light, where a fine carbon wire is forced to carry a large amount

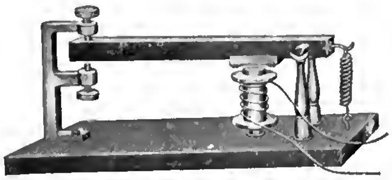


Fig. 4

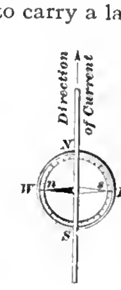


Fig. 5

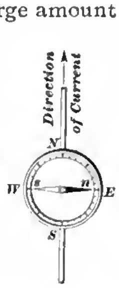


Fig. 6

of current, and becomes so intensely white hot (incandescent) that it gives out light.

Second, if wire insulated with cotton or silk be wound many times around a bar of ordinary soft iron as in Fig. 3, the ends of the wire being connected with a battery so that the current will flow through it, the iron bar will be found to become powerfully magnetic. As soon as the circuit is interrupted* at any point, whether by removing one or both wires from the battery, or breaking or cutting the wire anywhere throughout its length, nearly all the magnetism immediately departs from the iron. As soon as the circuit is closed, the magnetism returns, even though the opening and closing of the circuit is performed many times a second and the point of interruption is in a far distant part of the circuit. If another piece of iron† for the magnet to attract is balanced by a spring over the magnet, every time the circuit is closed it will be drawn toward the magnet, and when the circuit is opened the spring will draw it away. Such a piece of iron provided for the magnet to influence is called an armature. The telegraph sounder, shown in Fig. 4, works on this principle, the armature in its up and down movements causes a lever to strike resonant metal pegs, which give out the familiar "clicks" by the sound of which the operator reads the message. Many other electrical instruments also work on this same principle.

Third, that an electric current is flowing through the wire, *C*, Fig. 1, can be shown by crossing part of the wire over a compass needle, as shown in Fig. 5. Ordinarily, the needle of the compass will point north and south, and the wire above it should run in the same direction. But as soon as the connection with the battery is established the needle will be deflected, so that it will stand at right angles to the wire, or in other words point west and east. If the end of the wire that is connected with the positive pole of the battery be transformed to the negative pole, and vice versa (that is, if the "poles be changed") the needle will reverse its direction, so that the end which pointed east before will now point west.

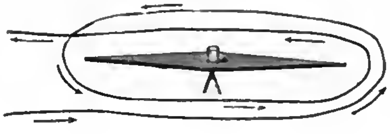


Fig. 7

The needle will also reverse its direction, if the wire be moved from its position above the needle to one below it, as shown in Fig. 6. In other words, the direction of the electric current affects the direction in which the magnetic

* Interrupting or breaking a circuit at any point is referred to by electricians as "opening" the circuit; reestablishing it so that the current can pass as "closing" the circuit.

† Soft iron readily loses nearly all its magnetism as soon as the current stops, and the softer the iron the more readily it becomes demagnetized; although it never loses all of it. Hard steel on the contrary retains a great deal of magnetism once it has been magnetized and on this principle depends a permanent magnet. The magnetism left in an electromagnet after the current has ceased is called "residual magnetism."

needle is deflected; and also deflects it one way when it passes above the needle, and in the opposite way when it passes below it. This fact enables us to greatly intensify the action of the current upon the magnetic needle, by putting a coil of insulated wire about a compass, as shown in Fig. 7. With such an arrangement all the strands of wire above the needle are carrying the current in one direction, and all those below in the opposite direction. They all, therefore, tend to deflect the needle in the same manner, and the effect is very greatly magnified, so much so, that such an instrument indicates the passage of currents that are too feeble to be detected by any other means. It is called a galvanometer.

Practical Magnets.—The electromagnet is much more powerful when, instead of winding the wire on a straight bar, as in Fig. 3, the bar is bent **U** shape, as shown at *A*, Fig. 8. For in this position both ends can be made to act at once upon the same piece of iron, and they can attract it with double, or more than double force. It is also found that the wire in the middle of the electromagnet does not have as much of an effect as that near the ends, and for this reason the wire is not generally wound on the middle part, but only on the ends, as shown at *B*, Fig. 8. Again, it is ordinarily advantageous, from the manufacturing standpoint, to make the iron core of a magnet in sections, afterwards fastening them together, as shown at *C*, Fig. 8. The section, as shown in Fig. 8 *D* at *H* can then be wound with the wire, just like thread is wound on a spool, securing great efficiency as well as ease in manufacture. Fig. 8 *D*, shows the section of a spool *I*.

Such electromagnets, as shown in Fig. 8 at *D*, when of large size and actuated by powerful currents, are of tremendous power, and will lift masses of iron weighing tons.

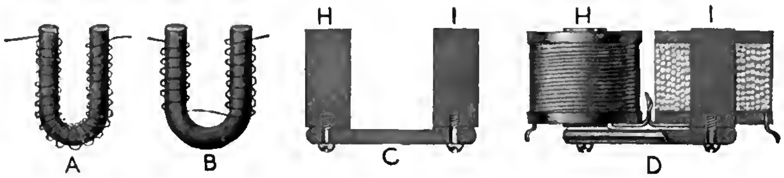


FIG. 8

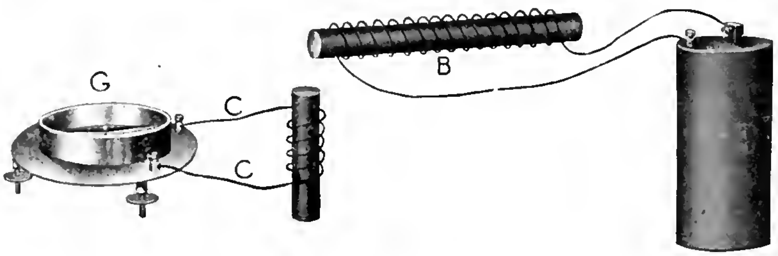


FIG. 9

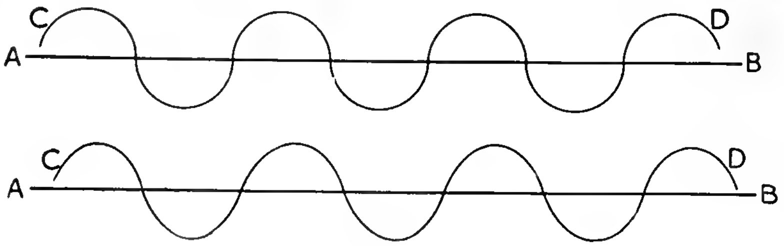


FIG. 10

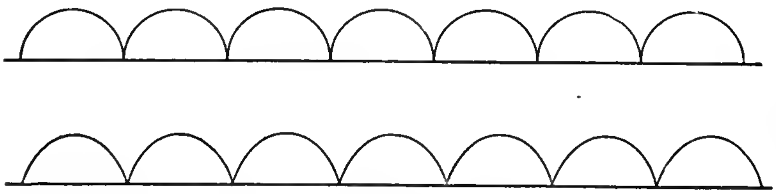


FIG. 11

If a piece of soft iron; i. e., an armature be placed between the poles of a conveniently-shaped magnet, as shown in Fig. 12, the piece of soft iron is also caused to become magnetic. North polarity is induced in the end near, or in contact with, the south pole, and vice versa. If the position of the armature is reversed, so that the end *A* is nearest the south, and *B* nearest the north pole of the magnet, then the armature reverses its polarity, so as to always present its south end to the north end of the controlling ("field") magnet. This it does, even though the reversal of ends is very rapid, such as would result from fixing a shaft into the armature, and rotating it rapidly in the direction shown by the arrows.

It has been shown how passing an electric current through wire wound upon a soft-iron bar is capable of causing it to become magnetic. The reverse of this proposition; namely, the induction of an electric current in the wire about an iron bar by causing the bar to become magnetic, is also true with certain limitations. Take such a bar wound with wire *C*, as in Fig. 9, and connect the ends with a galvanometer, the construction of which has been explained, to ascertain if a current passes. Now cause the end of the bar to approach the live electromagnet *B*. All the time it is approaching *B*, the galvanometer *G* shows that an electric current is passing. When the movement is stopped the current stops.

If the movement be reversed, that is, if the wire-bound bar be moved away from the magnet, the galvanometer will again show that a current is passing, but in the opposite direction.

The reason the current is induced in the wire *C* around the iron bar, is because the iron bar is caused to become magnetic as it approaches the live magnet *B* and loses this induced magnetism as it is withdrawn; and there is a natural law that any change in the magnetic condition of an iron core will induce an electric current in wire around it.

It may be argued that the result is not an electric current, but a series of electric pulsations. That is perfectly true; but if the pulsations are sufficiently frequent, through rapid rotation of the armature shaft, they produce similar enough effects to the steady flow of a battery current to be available for most purposes.

But the currents induced in the armature do more than pulsate. If the ends of the armature wire are connected with a galvanometer, and the armature slowly revolved so that the movements of the needle can be watched, the needle will be found to swing first to the east, then to the west, then east, then west again, changing direction with each half revolution of the armature. If the end of the galvanometer needle should be equipped with a pen, so that it could make a mark on a paper tape moved steadily beneath it by clockwork, the tracing that would be obtained by this experiment would look like Fig. 10.

In this illustration, the straight line *A-B* is the mark the pen would make, if the paper were moved forward with the needle at rest and pointing to zero, or north. The wavy line *C-D* is the mark made by the needle when such a current, continually reversing its direction, is sent through the apparatus. Such a current is called an alternating current.

But it is evident from what has already been said that such an alternating current would not do to energize the field magnet. In order to maintain the constant polarity of the field magnet, the current supplying it with energy must be in one direction, like the battery current—this is called a direct current. The alternating current induced in the armature is, therefore, rectified or changed into a direct current by means of the commutator, Fig. 13.

The commutator consists of a cylinder of hard fiber or rubber, covered with copper, and mounted on the same shaft which drives the armature. The copper is cut lengthwise into two sections A and B, Fig. 13. These are firmly attached to the surface of the hard fiber close together, but not touching; that is, they are insulated from each other by the fiber which carries them. One end of the armature wire is connected with section A, the other with section B.

The current is taken off of the commutator for use by C and D called brushes. Now, as the commutator revolves with

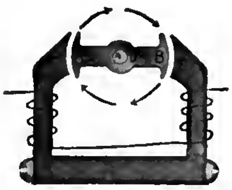


Fig. 12

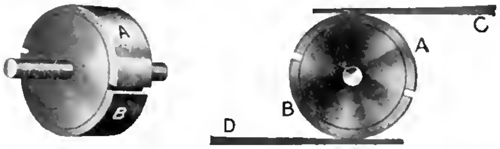


Fig. 13

the shaft, while the brushes remain stationary, the section A is in contact with brush C for half a revolution, then with brush D for the other half. The same thing happens to section B. Therefore, the effect of the commutator is to change the alternating armature current into a direct current, and the tracing which the recording galvanometer will make on the tape, when it receives the current from the dynamo equipped with the commutator, is shown in Fig. 11.

The alternating current would fire electric fuses as well as the direct current; but if it were not converted into a direct current as it is, we would either have to energize the field magnet with a battery, or use a permanent magnet. With the present arrangement, the rectified current is carried from the armature through the wire on the field magnet, which is thus energized by the current from its own armature. At the start, the very slight residual magnetism which is retained by the field magnet is sufficient to set up a feeble current in the armature; this in turn makes the field magnet stronger, and the stronger field magnet develops a stronger current in the armature; thus the machine "builds up," as it is called, until after a few revolutions it is working at its full power. This is noticed when one pushes down the rack bar of a "push-down" blasting machine. The first part of the stroke is easy, but after the armature has made a few revolutions, it pushes quite hard, because the magnet has become strong and pulls back on the armature, tending to resist the effort to turn it.

The dynamo just described, which is used in most American blasting machines, is one of the simplest and earliest forms of dynamo. Those used for generating powerful currents for electric lighting, etc., are more complicated, and more efficient electrically than those made on this simple design; that is, if the blasting machine dynamo were constructed on modern principles, it would take less power for the same output of current, or give greater output with the same amount of power, whichever way one chooses to look at it. But for the purpose to which a blasting machine is put, considerations of simplicity outweigh this kind of efficiency. Blasters would rather exert a little more muscle in operating the blasting machines than pay for the increased cost of repairs to a more modern dynamo, not to mention the increased first cost. Indeed, it is doubtful if a more satisfactory blasting machine could be reasonably asked for than the "push-down" machine, as it is now made. To show how the electrical principles so far explained are applied, the blasting machines shown in Figs. 14 and 15 are to be dissected. The field magnets F are shown enlarged in Fig. 16 by *8* and *9*. They are shown to be wound with insulated wire at each end and fastened in a way to form a **U**-shaped magnet. In both Figs. 14 and 15 is shown the brass end of the armature that revolves between the magnet ends F, but which is better shown by *16* in Fig. 16.

The brushes are shown in Figs. 14 and 15 bearing on the commutator, shown at *15*, Fig. 16. The ends of the armature wire are soldered to their respective commutator sections. In Fig. 16 the rack *1* attached to handle *36* is shown, the remaining numbers being *4* contact spring, *10* and *11* bearings, *81* connecting wire, *3* guide rod, *35* leather strap, *7* guide plate, and *45* shelf. These numbers represent repair parts and are not considered in this article. There is a pinion not shown which is on the armature shaft opposite the commutator, which is made to revolve as the rack is pushed down. This imparts a rotary movement to the armature.

There is one part of the blasting machine which has not yet been taken up, and that is the "shunt," sometimes called the "circuit breaker." This is the brass contrivance *s*, Figs. 14 and 15, placed in the bottom of the box. It is a brass spring which makes contact with the bridge above it at one end when in its normal position. The parts of the spring and bridge which come in contact are covered with platinum so that they will remain bright, and make a good electrical connection.

The function of this shunt is as follows: The spring *s*, Fig. 15, called the "contact spring" *4* is connected in Fig. 16 by means of a piece of heavy copper wire to one of the binding posts surmounted outside on top of the case with a wing-nut, representing one pole of the dynamo; the bridge is connected with the other binding post, representing the other pole of the dynamo, so that when the contact spring is up in contact with the bridge, a short easy circuit called a "shunt," of practically no resistance, is offered for the electric current to pass from one pole to the other—in the language of the electrician, the dynamo is "short-circuited." While the rack bar is being pushed down, the blasting machine is "building up," the current generated passing across the shunt, so that by the time the rack bar is near the bottom of the stroke the dynamo is

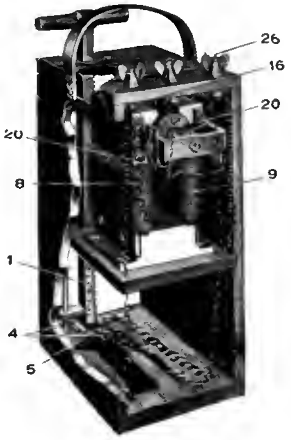


Fig. 14

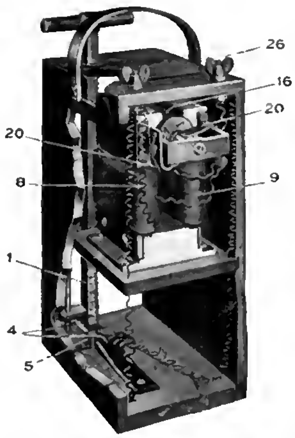


Fig. 15

working at its maximum. When the rack bar strikes the contact spring, however, separating it from the bridge, the short circuit is broken, and the current of the dynamo has now no other way to pass from one pole to the other except by flowing out through the electric fuse circuit, which it does just at the instant when it is at its maximum strength. The little piece of platinum wire within each electric fuse heats up almost instantly, causing them all to fire at practically the same time.

Were it not for the shunt, operating as just described, current from the dynamo would begin to flow through the

electric fuses, so soon as one started to push down the rack bar. It would be a very weak current at first, gradually increasing with the building up of the blasting machine. Such a current is not well adapted to fire a number of electric fuses simultaneously, because it is impossible to make all of exactly the same degree of sensitiveness, and with the gradually increasing current, the more sensitive electric fuses would fire first, breaking the circuit and causing the less sensitive ones to miss. By employing the shunt, on the other hand, no current is sent out from the blasting machine until there is ample power to fire even the least sensitive.

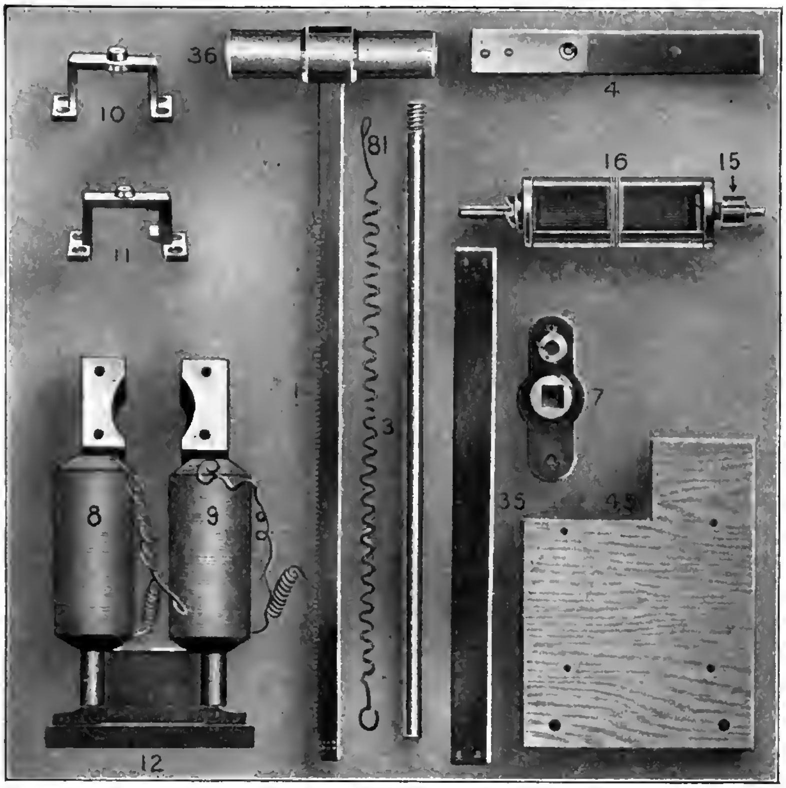


Fig. 16

The three-post blasting machine, Fig. 14, contains an ingenious device for increasing by nearly 50 per cent. the number of electric fuses that the two-post blasting machine would be able to fire. When it is desired to take advantage of the three-post feature of such a blasting machine, the electric fuse circuit is arranged as shown in Fig. 17. Three leading wires are used, which divide the blasting circuit into two separate circuits. The great increase in efficiency is brought about by equipping the contact spring with both an upper and a lower contact, and arranging it so that it will throw the full power of the dynamo first into one circuit and then into the other. While the two circuits do not really fire simultaneously, the interval between them is so extremely short that only one explosion is heard. If it were not very short the wires in one circuit would be broken by flying fragments of rock from those bore holes in which the charges detonated first.

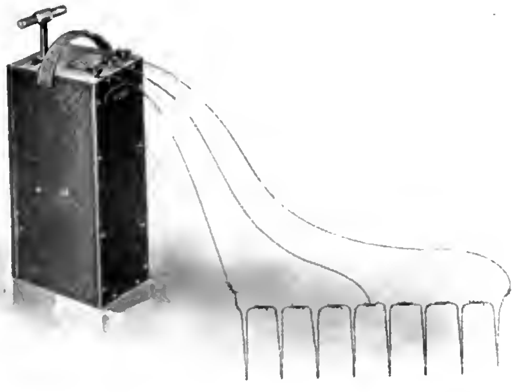


Fig. 17

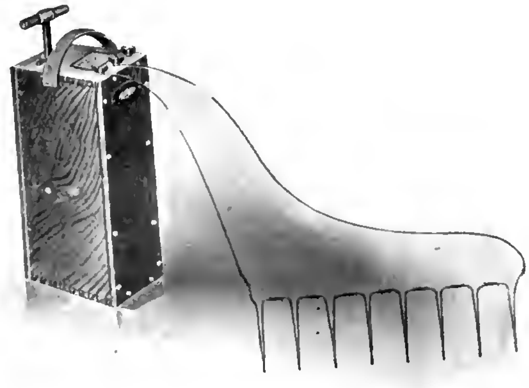


Fig. 18

By connecting to the middle binding post and either one of the outside ones, a three-post blasting machine may be used with two leading wires just like a two-post blasting machine, shown in Fig. 18, whenever the number of electric fuses to be fired does not exceed the capacity of a two-post blasting machine of that particular size. The great advantage in having a three-post blasting machine, is that when difficult conditions arise, which would ordinarily require a larger blasting machine, they can often be met in a very satisfactory manner, merely by laying another wire, and making use of the three-post feature.

⁂

[Eucalyptus trees are said to be quick growing and to thrive in localities and on soil where less hardy trees will not. It has been frequently suggested that these trees be planted in western mining sections where timber suitable for mining is scarce. The following complaint from the Forest Service is intended to correct the misstatements of boomers who are misusing Government reports to give their schemes plausibility.—Editor.]

The Department of Agriculture has been informed that certain of its publications dealing with eucalyptus have been misquoted by several companies interested in selling lands. For instance, Circular 97, of the Forest Service, has been misrepresented as saying that California will in a few years be the only source of hardwood supply in the United States. Such a statement has never been made in any of the Forest Service publications and is not considered a fact.

The Department experts believe that there is promise of considerable success in the cultivation of eucalyptus trees in many parts of California, but estimates of profit and of growth have been attributed to the Department which are unauthorized. There are many uncertainties connected with eucalyptus culture which the investor should take into account. In future any concern which attributes to the Forest Service unauthorized statements may expect the statements to be publicly disavowed.

COMMERCIAL CLASSIFICATION OF FUELS*

Fuels in their natural state may be classed as (1) gaseous, (2) liquid, and (3) solid. Under the third class, coal only will be considered in this paper.

The Qualities Necessary in Fuels According to the Purposes for Which they are to be Used

Coal is technically classified as anthracite, semibituminous, bituminous, and lignite.

In the commercial classification of fuels, coals are known as (1) steam, (2) by-product coking, (3) producer gas, (4) illuminating gas, (5) cement, and (6) domestic.

Steam coal constitutes more than 50 per cent. of all coals mined, and covers a wide range in quality. For steam production all kinds of coal are used, with the following range in analytical values:

Volatile matter, from 15 per cent. to 42 per cent.

Ash, from 4 per cent. to 18 per cent.

Sulphur, from .8 per cent. to 5 per cent.

British thermal units, from 9,000 to 14,800.

Anthracite is not included and will be eliminated from further consideration, as its use is confined principally to the large cities and densely populated districts along the eastern seaboard of the United States, where local conditions determine almost entirely the fuel to be used.

Steam coals may be subdivided into locomotive, steamship, and stationary power plant fuels.

Locomotive fuel is required to meet rapidly the maximum variations in demand for steam, and a coal which will deliver a considerable portion of its total heating value in the shortest possible time is the most satisfactory. This requirement is best met by gas coal, as it will give up about one-third of its total heat within 2 minutes after firing, due to the liberation and combustion of the volatile matter. The remaining fixed carbon maintains a constant temperature by its practically uniform rate of combustion.

That the manner of firing and the condition of the locomotive are the ruling factors in "smoky" or "smokeless" operation of locomotives with gas coal has been proved by some of the leading railroads. In such cases three-quarter gas coal is used with very satisfactory results, and almost smokeless combustion.

To avoid excessive smoke, low-volatile coals of high heating value are often used, but they do not burn as quickly as gas coals, a shortcoming compensated for by the reduction in smoke.

Aside from regions where low-volatile coal is the most available fuel and localities where strict smoke laws prevail, low-volatile coal is used by some railroads on certain passenger runs in order that the quantity of smoke may be more easily kept at a minimum.

While high-ash coal is not especially desirable in locomotive fuel it is used with very satisfactory results, due to the readiness with which the ashes can be removed from the firebox as well as to the increased grate surface with which recent locomotives are provided. For locomotive fuel, sulphur in coal need be given no serious consideration. The belief that locomotive coal must be lumpy or screened is gradually disappearing.

Owing to the strong draft in a locomotive firebox a large proportion of fine coal is objectionable, as considerable of it may be carried through the flues without combustion. The tendency to exaggerate the extent of such loss is due largely to prejudice against any but lumpy coals for locomotive use.

Steamship fuel is required to meet a steady demand for steam, as the boiler load remains practically constant once the vessel is under way. For such service a fuel high in heat units and having a practically uniform rate of delivery of its heat units is best adapted. Low-volatile coals are most desirable for steamship fuel, and nearly all vessels of the trans-Atlantic trade and of the United States navy have adopted such fuel.

Practically the entire tonnage of low-volatile coal reaches the market as run-of-mine, consequently the percentage of small coal is considerably in excess of that in a similar quantity of run-of-mine bituminous coals; but this is not objectionable in steamship operation, because, with the draft much less than in locomotive operation, the loss due to unburned coal is small.

On some freight vessels sailing the Great Lakes mechanical stokers have been installed that use slack of bituminous coals with satisfactory results, as far as can be learned.

The chemical analysis for steamship fuel would be within the range of percentage values given for locomotive fuel—specifications usually requiring ash under 10 per cent., sulphur about 2 per cent. or under, volatile matter approaching the lower limit, and British thermal units near the upper limits. Sulphur content is of no importance, and the requirement of 2 per cent. or under must be met to satisfy the purchaser rather than because of any detrimental effect the sulphur may have on the fuel value of the coal.

The load of large central power stations usually varies greatly, and to meet the peak loads the fuel must generate steam quickly. The advantages of meeting this demand by using a coal which will give up a large portion of its heat units in the least time after firing becomes more apparent as the boiler load and capacity approach an equality.

The selection of coal for large power plants in densely populated districts is influenced by the smoke laws operative in such localities, and to comply with the law low-volatile coals are given preference.

In boiler plants fired by mechanical stokers in the eastern United States, low-volatile run-of-mine coal is used, crushed so that no piece exceeds 3 inches in size.

In the central section the prevailing practice in large plants is stoker-fired boilers, using slack or screenings of bituminous coal. Fuel of this size avoids the necessity of crushing, and, for the greater portion of the time, affords practically smokeless operation. Bituminous slack is capable of meeting sudden changes in load more readily than low-volatile coal, and allows more uniform operation under widely varying plant loads.

Where freight is the larger part of fuel cost, the highest quality of coal is the best investment for the purchaser, as the cost of transporting coals of any quality for the same distance would be identical. For this reason principally, the eastern United States is the great market for the best grades of low-volatile coal. In the central section slack is the cheapest fuel obtainable.

If the use of low-volatile coal is advantageous in large boiler plants equipped with mechanical stokers, the peak loads are usually provided for by having ample boiler capacity. In small power plants with hand firing, the size of coal is considered of importance. In rare instances do such plants have the variations in load commonly found in large power stations. Taking into consideration steam fuels used in the various regions, it is found that they cover the entire range of chemical values, but in some small plants a coal low in ash is desirable on account of furnace-grate arrangement.

Smoke prevention is frequently of much importance. Care in manner of firing and selection of coal will usually permit of operation within the requirements of the smoke laws.

For by-product coking, requirements as to quality are far more strict than for steam coal, the range for sulphur being confined to rather narrow limits. In the by-product process coal is coked in such a manner that the gas, tar, and ammonia are recovered. Gas is sometimes the principal product and the ammonia, tar, and coke are the by-products.

A satisfactory by-product coal must be rich in gas, and meet certain requirements as to quantity of some of its constituent elements, the limits of which are determined largely by

* Abstracted from R. E. Rightmire's paper read at the Wheeling, W. Va., meeting of the West Virginia Mining Institute, December, 1910.

the purposes for which the coke is to be used. This means that gas coals are most desirable for such use. While many low volatile coals will yield a satisfactory coke, their yield of gas is low, and and bituminous coals other than gas coals are unsuited, because they are non-coking and their gas lacks the rich illuminant found in strictly "gas" coals.

Where illuminating gas is the principal feature and the coke is not intended for use in iron manufacture, limits of sulphur are not so closely drawn, although it is difficult to purify gas made from high-sulphur coal. Two per cent. or over of sulphur in coal has been used in such plants, the coke being used for steam and domestic purposes.

For by-product operation in connection with steel plants, sulphur limits in coal are ordinarily placed at 1.50 per cent. as a maximum, with an average value of about 1 per cent. to 1.25 per cent.

The standard range for coal for by-product use is 6 per cent. to 7.50 per cent. ash, which produces a coke ranging from 9 per cent. to 11 per cent. in ash. Ash is objectionable in coke for iron making, as its final appearance is in the form of slag, which requires heat for its production, representing so much loss of heating value and in part explaining why limits have been established.

In general, the range of analytical values for by-product coals would be represented by:

Volatile matter, from 28 per cent. to 38 per cent.

Ash, from 6 per cent. to 7.50 per cent.

Sulphur, from minimum to 1.50 per cent. for coke used for metallurgical purposes. From minimum to 2.50 per cent. for coke not intended for metallurgical purposes.

Fuel for producer gas covers a wide range in values, especially since gas engines are meeting with increased favor as prime movers.

Producer gas found favor by supplying the demand for cheaper fuel, and is used principally for metallurgical and power purposes, although it is applicable for kiln-burning of clay products, lime, cement, and firing steam boilers. The principal use of producer gas is in the manufacture of steel, which requires a coal ranging from a minimum to 1.5 per cent. in sulphur content.

All coals will produce about the same quantity of producer gas, but in metallurgical processes gas coal is preferred, for the reason that its gas burns with a long flame, giving more uniform distribution of heat at high temperature.

Other bituminous coal from which the demand could be supplied, and satisfactory temperatures obtained, have a prohibitive sulphur content. The selection of fuel for producer gas is limited by the use to be made of the gas and the type of producer.

Producers are classified according to their method of operation as, (1) suction, (2) pressure, and (3) down draft.

The suction producer has found application in small plants operated by gas engines, and the fuel has been limited almost exclusively to charcoal, anthracite, and coke, or fuels whose gas is free from tar, which is very objectionable in gas-engine operation.

Pressure producers are operated under a low pressure produced by a blast of steam and air. This is the type generally used for metallurgical purposes where the gas is used direct from the producers and tar is not objectionable.

For gas-engine use the gas is stored and the tar removed before passing to the engine, and lignite, peat, and bituminous coals, as well as charcoal, coke, and anthracite, may be satisfactorily used in this type of producer.

Tests with a pressure producer conducted by the United States Geological Survey at the various government testing plants have shown satisfactory results with fuels ranging widely in analytical values, as follows:

Moisture, 1.40 to 39.60 per cent.; volatile matter, from 9.70 to 42.50 per cent.; fixed carbon, from 23.80 to 73.70 per cent.; ash, from 2.70 to 23.40 per cent.; sulphur, from .30 to 7.40 per cent.

It is reported that bone coal containing 44 per cent. of ash produced a gas affording economical gas-engine operation, and that any coal of commercial value can be successfully used.

Down-draft producers made their appearance after the pressure producers. These fix the tar as a permanent gas, thus making use of all the volatile matter and preparing it for immediate gas-engine use.

Gas producers are utilizing more and more the inferior grades of fuel, and at present are operated with fuels covering the range from anthracite to charcoal and peat.

In illuminating gas manufacture, practically all plants specify a screened coal gas of low sulphur content.

Three-quarter-inch screened coal is specified for several reasons:

1. It can be gasified in less time than run-of-mine coal.
2. It is lower in sulphur than run-of-mine coal.
3. Charging of retorts by hand is readily accomplished.

Low sulphur is required that the gas may be purified to meet the requirements of the law.

The commonly accepted standard of analytical values for illuminating gas coal would be approximately represented by the following: Volatile matter, from 32 to 37 per cent.; ash, from 6 to 8 per cent.; sulphur, not to exceed 1.50 per cent.

The usual standard for yield of gas is an average of 10,000 cubic feet per ton of 2,000 pounds of coal, with an average candle power of 18.

Coal for cement burning is reduced to a powder before being used, hence slack coal is generally used because of its low price and the readiness with which it can be reduced. The essential requirements are that the fuel be: (1) Sufficiently high in volatile combustible matter to insure quick ignition; (2) sufficiently high in heating power.

While the maximum sulphur content of cement is definitely fixed, the sulphur in the coal has not been closely limited, as it does not enter into combination with the cement as long as proper kiln temperatures are maintained. As the impurities of some coals are approximately of the same composition as the cement, no strict limits have been placed on the amount of ash in such coals, but high ash percentages reduce the heating power, and, for this reason only, a low ash content is desired.

Gas coals have been found to give best results in cement burning, as they are high in volatile matter, which has a high heating value with long flame of quick ignition and maximum temperature at a short distance within the kiln.

If coal which ignites slowly is used, ignition takes place too far inside the kiln, giving the zone of maximum temperature at such a distance from the firing end that difficulties are encountered in securing proper clinkering of the cement, besides increasing loss of heat due to the waste gases leaving the kiln at a high temperature. While gas coals are considered the best for cement burning, other high-volatile bituminous coals are used satisfactorily.

Domestic coal must meet consumers' requirements as to size, and is prepared as lump, egg, and nut. Coal firm in structure and suffering the minimum amount of breakage in handling will meet domestic requirements most satisfactorily. It should be non-coking or free burning, so that its heat is given off readily. The ash should be small in quantity and free from clinker.

Export bituminous coal must be in large sizes and of firm, tough structure, that it may reach its destination in large sizes with minimum amount of breakage. This coal is usually passed over 1½-inch screens before shipment from mines.

With the low volatile or semibituminous coals which produce a large amount of fine coal in mining, size of coal for domestic purposes is obtained by screening. For export these coals are usually shipped as run of mine.

In general, the preparation of coal for market by the

removal of slate, bone, and sulphur balls, is of importance to the producer as well as the purchaser. It is of importance to the producer in extending his market by maintaining the reputation of the coal he produces at a high standard, and to the purchaser that he may receive the greatest fuel value for the money expended.

Sulphur balls produce about one-third as much heat in burning as the same weight of coal, and on this basis alone are not so objectionable as other impurities, which have no heating value whatever. They are objectionable on account of difficulty in handling the clinker they produce.

The impurities without heating value go to increase the amount of ash, and represent so much worthless material at the same price and freight rate as the coal. As transportation is usually the greater portion of cost of coal to the consumer, the importance of removing as much free impurities as possible is readily apparent.

The selection of coal to be used for any particular purpose is dependent upon, cost to the consumer, limitations imposed by smoke laws, and fuel value. The fuel value is greatly influenced by the readiness with which clinker is formed from the ash, the greater the freedom from clinker the greater is the recoverable "fuel value."

THE BELLEVUE EXPLOSION, ALBERTA

*Written for Mines and Minerals, by James Ashworth**

On the evening of the 31st of October last, the writer was called on to visit the Bellevue Mine, belonging to the West Canadian Collieries, Ltd., where an explosion had occurred about noon on that day. This mine has been developed by a main gangway driven in on the coal seam from the tipple level. The rooms driven up the pitch of the seam, which is from 60 degrees to vertical, have cross-cuts at various distances between them Before the coal reaches the surface an anticline throws it down to a syncline, from whence it again rises, and thus, by cross-cutting into the mountain above the anticline, the seam is again worked from a surface opening. This part of the seam has a separate ventilating current of air. The coal from this No. 2 working is brought in cars to the top of No. 45 chute, down which it is carried to the main gangway and again loaded into cars and taken by an air locomotive to the tipple. Nos. 81 and 82 chutes are the upcast proper, but there are two other openings through which some air is allowed to escape via No. 45, and one where the mine has caved to the surface. No. 109 is also in course of being driven through the rock cover to the surface. For ventilation there is a Sirocco fan 78 inches diameter by 72 inches wide, driven by an electric motor. This fan furnishes the mine with about 60,000 cubic feet of air per minute at 2-inch water gauge. A short time ago the mine was worked with open lights, but now all the miners use the Wolf double-gauze safety lamp.

Difficulties in Bringing a Large number of Men through a Poisonous Atmosphere

After the miners had quit work on Saturday, the 29th of October, the fan was stopped for about 2 hours, and on the Sunday from about 7:30 A. M. to 4:30 P. M. The explosion occurred at noon on Monday, when no one was in the mine. No effects were apparent at the mine entry, but a door on the fan was blown open without doing any serious damage, and was quickly reinstated. The only people who can be said to have seen the effect of the explosion were a party of electricians who were putting up an electric power line on the surface between the generating station and the upper mine on the mountain (No. 2). These men were scared out of their wits by a loud noise, and the issuing of what they thought was smoke and dust from the caved-in portion of the mine, and also by the same appearance accompanied by more force from No. 81 chute, the upcast, and from No. 45 chute. They at once rushed down to Bellevue and gave the alarm. The effect of the explosion was also felt by a dairyman, whose milk ranch is not far from the position of No. 36 chute. As soon as the officials could be assembled, an examination of the mine was made and a great deal of damage was found inside, chutes, brattices, and timbers being blown out. It was at first thought that there was a fire in one of the chutes but this was not the case and, as the main doors in the gangway were fortunately undamaged, men were sent to the top of No. 45 chute to cover it over so that the force of the ventilation could be concentrated on No. 81. All the outlets to the surface were examined, and the writer came to the conclusion that there was little or no fire underground, and that the disaster was not an explosion of firedamp so much as a huge caving of rock. On examining the mine it was found that there had been a perfect hurricane of "sludge" on the main gangway extending outwards from the neighborhood of 72 chute, and inwards from about the same point. The ends of the loaded cars were thickly coated with about 2 inches deep of sludge, the timbers likewise, and at several points inbye the props were coated on both sides with a conical streak of mud, showing great force inbye and an equal force outbye, caused by the reflex action of the compressed air. No fire was found in the mine, but some coal in No. 82 chute was decidedly warm. Further investigation showed that there had been a very large fall of rock where the pillars were being extracted, in the neighborhood of 71 and 75 chutes. There was no evidence of flame anywhere on the main gangway, which, as previously remarked, was soaking wet.

The damage to the mine was not so great as to seriously interfere with its reopening, and it was just getting back to its normal condition when about 7 P. M., on the 9th of December, the engineer of the power house found both the high- and low-pressure air lines had been broken, whereupon he stopped the high-pressure engines thinking that a joint had blown out. No damage was done to the fan. A horse driver on the surface tramway connecting No. 2 mine with the top of No. 45 chute of No. 1 mine was the first to report that he had seen smoke and dust thrown out of the chute, and was told that he was spinning a yarn. Probably if it had not been for this incident the fact of a disaster having occurred would not have been known in Bellevue sooner than when the few men who escaped alive had reached the mouth of the entry.

The writer believes that it was due to Mr. Brown, of the Hillcrest mines, that phones and wires were put into use to summon the trained men of the district, with the Draeger apparatus from the British Columbian Government station at Hosmer, also from the Crows Nest Pass Coal Co.'s mines at Fernie. All the men who could quickly be collected together were transported to Bellevue by special train from Fernie, calling at Hosmer and Michel. This party was composed of Mr. Strachan, British Columbian Inspector of Mines; Messrs. Wilson, Fuller, and Alderson, of the Hosmer mines; Mr. Norman Fraser, of the C. N. P. C. Co.'s mines at Michel; a miner named Birmingham, who had had experience with the Draeger apparatus in Nova Scotia, Messrs. A. J. Carter and W. Mackie (who had been in the Bellevue Mine the previous afternoon as late as 4:30 to 5 o'clock) and the writer. This party arrived at Bellevue shortly after 2 A. M. Later another train, the "Flyer," was stopped at Bellevue and landed a further contingent of Draeger and other experienced men amongst whom were Mr. Evan Evans, another British Columbia Inspector of Mines, who brought the Half-Hour Draeger outfit from the C. N. P. C. Co.'s Coal Creek mines, and Messrs. Spruston, Matusky, Huby, Mathers, and Musgrove. Later in the day, Mr. Williams, of Lille, and Quigley with others from Hillcrest, rendered assistance, also Messrs. Whiteside, Davidson, and others from Coleman. By the first train in the morning Mr. Shanks, of the C. N. P. C. Co.'s mines, at Coal Creek, arrived with all the company's Draeger apparatus and a band of willing and experienced officials. The

* General Manager Crow's Nest Pass Coal Co., Ltd.

latter apparatus was, however, too late to be of service. Before the arrival of the rescue party 6 men had come out of the mine alive and the bodies of 4 dead men had been recovered. Two of these men had been killed by violence in the neighborhood of No. 54 chute, and two had been suffocated by afterdamp. One man who escaped alive was up in No. 45 chute bucking coal when the disaster occurred. The driver of the air locomotive standing near No. 54 chute was inbye, and before he heard any noise he saw what he thought was a white cloud approaching and he was immediately knocked down by a hurricane of sludge. The loaded cars alongside of which he was passing at the time must have protected him from the direct force, and thus have saved his life.

The first rescue party under Mr. John Powell, the superintendent of the mine, and who had taken charge only 8 days previously, found a few men on the road alive, and also 21 dead all in one bunch arranged around a high-pressure air locomotive charging station, near No. 84 chute. About this time the rescue party arrived on the scene and commenced work. Messrs. Strachan and Alderson first went in with a full 2-hour helmet outfit, but after going inbye about 500 feet had to return, as Alderson said something was wrong with his helmet. On a second try they found a group of men ranged around the inside high-pressure air charging station at No. 124 chute. The problem now was how to bring these men out from No. 124

Surface Plant, Bellevue Mine, B. C.

chute to No. 84 chute, a distance of about 2,000 feet. It was agreed that Strachan should go out and take one man with him by the aid of Alderson's Draeger outfit, and thus Alderson was left inside with the men and without any rescue apparatus. Mr. Strachan then started in again with his 2-hour apparatus and carried another complete apparatus for Alderson. Alderson then put on the apparatus, and Mr. Strachan stripped off his own and put it on another man, whom Mr. Alderson then took out to No. 84. Alderson again started in in his outfit, and carrying another full suit for Mr. Strachan's use, the load was, however, too much for him, and he dropped it on the way. On reaching Mr. Strachan he had his chin valve open and said he thought his potash cartridge was played out and consequently he was himself very exhausted. He then took off his apparatus and remained inside with the men to be rescued, and who were still in good form, having the high-pressure air to keep them alive. Mr. Strachan then put on the apparatus and came out without experiencing any difficulty. This appears to show that Alderson must have hurried in, and made a greater demand on the apparatus for air than it was capable of supplying The Draeger men had in the meantime been sent out of the mine to bring in two sets of half-hour apparatus, and Messrs. Evans and Huby also went out for a further supply of 2-hour oxygen cylinders and potash cartridges. About this time Birmingham had returned with two $\frac{1}{2}$-hour apparatuses, and another Draeger man, Matusky, put on a 2-hour apparatus, and went in carrying a $\frac{1}{2}$-hour apparatus in his hand. When he arrived at the place where the men were grouped together, he was alarmed to find that all of them had collapsed, from some reason that was not apparent to him. One man, the fire boss, was standing, and could just speak, but was incapable of assisting Matusky in putting the $\frac{1}{2}$-hour Draeger apparatus on him. Matusky then left the $\frac{1}{2}$-hour set with the man and hastened out to report the serious state of affairs. A messenger was then despatched to the outside to bring in additional help, and a conference was held to determine what should be done to save the men at No. 124 chute. It was decided to make a dash and pull out the men without waiting for the extra oxygen or for a rope, which had also been sent for. This "forlorn hope" party then formed themselves into a string about 10 feet apart. Mr. Spruston led the way and carefully tested the air as he advanced, and the only gas he detected on his lamp was about a $\frac{3}{8}$ to $\frac{1}{2}$-inch cap of firedamp before he reached the unconscious men. None of this party appear to have perceived the effect of any gas until they commenced to exert themselves, and hence the whole of them were affected at about the same moment. It then became a *sauve qui peut*, some reaching No. 84 chute dizzy and almost unconscious, and others falling by the way. Another messenger then rushed out of the mine and announced that every one of the rescue party was lost, and the Draeger men in particular. Doctor Mackenzie, who formed one of the "forlorn hope," in his eagerness to be of service carried in the pulmotor apparatus, and probably fell at the same time, at No. 114 chute. Messrs. Evans and Huby and others who arrived on the scene about this time were the means of bringing out and reviving by artificial respiration the whole of the men found at No. 124, excepting only Alderson and another man, who were found much later side by side and too late to revive them. The last man to be rescued alive was the doctor, who along with two others was pulled out by men attached to a rope which had been brought in by the fresh band of rescuers. At least 10 men of the "forlorn hope" party were rendered helpless by a mixture of firedamp and carbon monoxide gas. That it was this latter insidious gas which rendered the work of this party so dangerous is clearly proved by the certificate of the doctor, who certified that the two men taken out dead had died from poisoning by this gas.

This experience with the Draeger apparatus is a practical proof of the great danger which is inseparable from attempts to do work which is outside its scope, and such experiences bring out the practical fact, that though it may be of inestimable value as the assistant to old-fashioned methods of life saving, such as the renewal of brattices, doors, stoppings, etc., as the party advances, it does not entitle a large rescue party to run the risk of being cut off by gas driven into the roads behind them through a mistaken notion that the use of the Draeger or other rescue apparatus entitles the rescuers to neglect the plain common-sense precautions of mine-rescue operations.

The public at large, if the editorials in the public press correctly represent their opinions, expect too much from the Draeger apparatus. Thus it it has been stated in connection with the Bellevue disaster that if such apparatus had been immediately available every man would have been gotten out alive. No statement could be more misleading, and this was clearly demonstrated by the disaster which overtook the rescue party, and which cost the life of one brave man, and that of one of the men whom he was attempting to save.

There is a great difference of opinion amongst mine officials as to which size of apparatus is the most useful, and the experience at Bellevue shows most distinctly (in the writer's opinion) that the $\frac{1}{2}$-hour apparatus is far more useful than the 2-hour—in the first place it is lighter; viz., about 14 pounds, as compared with about 42 pounds; secondly, the wearer is not muffled up in a helmet, he is cooler, has the full use of his eyes, and there is no complication arising from a proper fit of the

helmet to his face, and no chin valve to close. It is also quickly adjusted.

The Bellevue case afforded a demonstration of the fact that any of the Draeger apparatus can be used for bringing a man out through a considerable length of poisonous atmosphere, even when he has never seen or used the apparatus previously. That the great weight of the 2-hour apparatus is a serious drawback was exemplified in the case of Alderson, who failed on his second journey to carry in a full outfit in addition to the one he was himself wearing. Another lesson worth noting is that the apparatus ought never to be put on in a vitiated atmosphere excepting in cases of absolute necessity.

The ½-hour apparatus is handier than the 2-hour when a man is renewing stoppings, etc., and most important of all it compels the wearer to keep in closer touch with the main rescue party.

The pulmotor and oxygen trunk are most important adjuncts to a rescue party, and although the former is recommended for use on the surface, the writer would, after the Bellevue experience, recommend that it be taken as near to the rescue party as the state of the air will admit, and that the oxygen trunk be even closer to the active rescuers, and in fact actually part of their outfit. It is just the "stitch in time which saves nine," and the trunk may easily be that one stitch.

In conclusion, the writer would suggest that every one with rescue experience should fully discuss in MINES AND MINERALS the very important points in rescue work which he has to some extent adversely criticised.

CATALOGS RECEIVED

In writing for catalogs, please mention Mines and Minerals

THE ALDRICH PUMP DEPARTMENT, Allentown, Pa., Pump Data, No. 12A, The Aldrich Electric Sinking and Recovery Pump, 12 pages.

CONVEYING MACHINERY CO., 120 Liberty Street, New York, N. Y., Catalog No. 110, descriptive of their well-known coal handling machine, 32 pages.

BALDWIN LOCOMOTIVE WORKS, Philadelphia, Pa., Record No. 68, Mallet Articulated Locomotives, 36 pages.

COLORADO PORTLAND CEMENT CO., Denver, Colo., Alkali-Proof Portland Cement, 36 pages.

CHICAGO PNEUMATIC TOOL CO., Fisher Building, Chicago, Ill., Catalog No. 33, Franklin Air Compressors, 96 pages.

CHRISTY BOX CAR LOADER CO., Des Moines, Iowa, Pigeon Hole Information about Christy Box Car Loaders, 16 pages.

COLORADO IRON WORKS CO., Denver, Colo., Pamphlet No. 28, The Portland Continuous Filter, 8 pages.

E. I. DUPONT DE NEMOURS POWDER CO., Wilmington, Del., The "Ammonia" Dynamites and Their Uses, 8 pages; A Few Pointers on Charging and Tamping Bore Holes, 10 pages; Ditching With DuPont Dynamite, 16 pages.

DODGE MFG. CO., Mishawaka, Ind., A Remarkable Test, Five and One-Half Miles Per Minute, 12 pages.

DEAN BROS. STEAM PUMP WORKS, Indianapolis, Ind., Catalog No. 83, Special Boiler Feeders or Pressure Pumps, 28 pages.

ELECTRIC RAILWAY EQUIPMENT CO., Cincinnati, Ohio, Catalog No. 14, Electric Mine Haulage Supplies, 114 pages.

INGERSOLL-RAND CO., 11 Broadway, New York, N. Y., Class "A," Straight Line Steam Driven Single Stage Air Compressors, 20 pages; Class "PB," Duplex Power-Driven Air Compressors, 24 pages; "Sergeant" Rock Drills, 24 pages; "Radialaxe" Air-Driven Coal Cutters, 20 pages; Davis "Calyx Diamondless" Core Drills, 48 pages; Pneumatic Tamping Machines, 12 pages.

31—7—4

NATIONAL ELECTRIC LAMP ASSOCIATION, Cleveland, Ohio, Bulletin 9B, High Efficiency Lamps, 34 pages; The Engineering and Scientific Activities of the National Electric Lamp Association, 14 pages.

PRECISION INSTRUMENT CO., Detroit, Mich., Catalog D, Precision Simmance-Abady Combustion Recorder, 16 pages.

MCKIERNAN-TERRY DRILL CO., 115 Broadway, New York, N. Y., Core Drills for Prospecting, Testing, Blasting, etc., 60 pages.

VULCANITE PORTLAND CEMENT CO., Philadelphia, Pa., Pamphlet No. 10, Concrete Surface Finishes, 12 pages; Pamphlet No. 11, Concrete in the Country, 112 pages.

WESTERN ELECTRIC CO., New York, N. Y., Bulletin No. 5,500, Hawthorn Direct and Alternating Current Enclosed Arc Lamps, 20 pages; Condulet Talk, Series 2, No. 5, 12 pages.

B. F. STURTEVANT CO., Hyde Park, Mass., Bulletin No. 187, Economical Fire Room Methods, by F. R. Low, 23 pages.

GENERAL ELECTRIC CO., Schenectady, N. Y., Bulletin No. 4,790, Electric Mine Locomotives, 34 pages; Bulletin No. 4,796, Alternating Current Generators for Direct Connection to Reciprocating Engines, 12 pages.

THE BRISTOL CO., Waterbury, Conn., Bulletin No. 130, Wm. H. Bristol Indicating and Recording Pyrometers, 56 pages; Bristol's Recording Pressure Gauges, 8 pages.

STROMBERG-CARLSON TELEPHONE MFG. CO., Rochester, N. Y., Circulars covering various types of battery instruments.

TAYLOR INSTRUMENT COMPANIES, Rochester, N. Y., "Tycos"-Rochester, Vol. 1, No. 9, 12 pages; Winter Motoring Now Assured by the Pyrofreezometer, 4 pages.

JOHN DAVIS & SON (DERBY) LTD., 110 West Fayette Street, Baltimore, Md., Selections from Catalogs, 28 pages.

LINDFORTH & MILBURN, Hennessy Building, Butte, Mont., Engineering Models, 25 pages.

CALENDARS RECEIVED

The Roessler & Hasslacher Chemical Co., New York.
Keuffel & Esser Co., Hoboken, N. J.
General Electric Co., Schenectady, N. Y.
John A. Robling's Sons Co., Trenton, N. J.
Morris Machine Works, Baldwinsville, N. Y.
Lindsay Brothers, Philadelphia, Pa.
Ironsides Co., Columbus, Ohio.
Warren-Ehret Co., Philadelphia, Pa.
Watt Mining Car Wheel Co., Barnesville, Ohio.
Maloney Oil and Mfg. Co., Scranton, Pa.
Hazard Mfg. Co., Wilkes-Barre, Pa.
Weston Dodson & Co., Bethlehem, Pa.
Baldwin Locomotive Works, Philadelphia, Pa.
York Bridge Co., York, Pa.
Aldrich Pump Department, Allentown, Pa.

VALUE OF ALASKA'S COAL LANDS

In the hearings before the joint committee that investigated the Interior Department and the Forest Service, A. H. Brooks, of the United States Geological Survey, testified that the accessible coal of the best Alaskan fields—the Bering River and Matanuska—was worth half a cent a ton in the ground. To some persons this meant that the coal lands of Alaska had no value whatever, but the value stated by Mr. Brooks is higher than that of most coal lands in the Eastern States, notwithstanding their nearness to lines of transportation and to markets. In fact, good bituminous coal in some well-developed eastern fields has recently sold for one-thirteenth of a cent a ton in the ground.

Priced at the rate given the best Alaska coal lands are worth from $50 to $500 an acre, values far above the average price of bituminous coal lands in the United States.—*United States Geological Survey.*

THREE-WIRE MINE SERVICE

Written for Mines and Minerals, by J. M. Hunt

Method of Reducing Cost for Copper Wire in Mines Operated by 250 Volt Direct Current

Every coal operator when developing a new property which will use electricity for its source of power, is brought face to face with the problem relative to the system best suited to his wants.

This question in the case of the larger operator is taken care of by his consulting engineer, and so will not be discussed in this article, which purports only to suggest a method that may be followed by the smaller operators who are planning development, and to those who already have the 250-volt system and find that the cost of sufficient wire for satisfactory transmission is mounting so high as to become almost prohibitive.

The alternating-current system, with its main generating station and auxiliary substations is, in general, only to be considered for large operations having several openings, with a considerable distance of outside as well as inside transmission.

The 500-volt system is a temptation to most of the smaller operators, owing to the efficiency with which it may be transmitted, but when considering 500 volts one is inclined to go very slow before settling on it as the best, due partly to the fact that there has, in the past, been some agitation in an attempt to outlaw it on account of the danger to life, and partly to the fact that the inside-connected machinery has not proved as durable as has that working on 250 volts, due probably to the moisture generally encountered in mines. So that in most cases the operator has settled on the 250-volt system with its resulting enormous line losses.

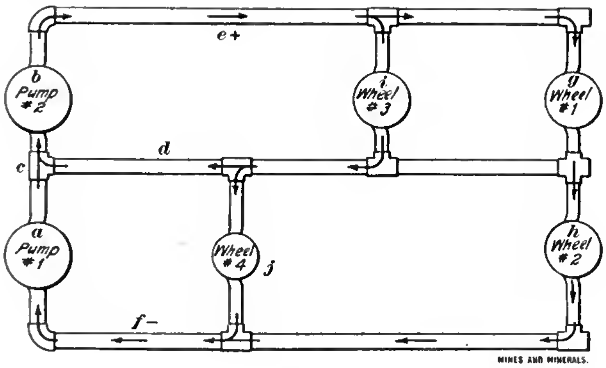


Fig. 1

The Edison three-wire system is in use in West Virginia only to a very limited extent, and that with the special three-wire generator. The writer knows of no such system where the three-wire results are obtained by running two 250-volt generators in series. Is it because it has proved itself to be unsatisfactory? No, for it was long in use before the three-wire generator was developed. In fact the satisfactory operation of two generators in series for lighting plants gave birth to the three-wire generator. Then what is the reason? Is it not that the average operator of small plants has felt that the size of his operation did not justify him in employing an engineer to work out such a small detail as his power equipment? He found that his neighbor's 250-volt plant was giving good results, and rather than get himself mixed up with anything that he had never seen in operation and which sounded so complicated as the Edison three-wire system, he would copy this neighbor.

Now that he has copied him and development has advanced so that he is getting anywhere from 40 to 75 per cent. line loss, and he finds that sufficient copper to cut this down to anything like a legitimate loss will cost him a miniature fortune, he is wondering if he did not make a mistake, and if so, where is the way out.

The writer will endeavor in this article to make the principles of the three-wire system a little plainer than perhaps it has been heretofore, and maybe show the operator that he did not make such a gross mistake as he had before feared.

The action of electricity has often been made more comprehensible to those who are not well posted by comparing it with that of water. It will perhaps serve better here to demonstrate the theory of the three-wire system than any other comparison we could make.

Let two pumps, *a* and *b*, Fig. 1, represent two generators. Assume each pump capable of delivering a 6-inch stream of water, which represents the current, at a pressure of 250 pounds, which would represent the voltage. Then there would be a pressure between the discharge and the suction of one pump amounting to 250 pounds. Should the suction of both pumps be connected to one pipe and the discharge of both to another, there would still be 250 pounds pressure between the discharge and the suction, but a stream equal to two 6-inch pipes. Should, however, the discharge of the pump *a* be connected to the suction of the pump *b* there would be a pressure of 500 pounds between the discharge of *b* and the suction of *a* throwing a 6-inch stream. If a T connection *c* was placed at the point where the two pumps are joined and from this point bring out a third pipe, it would then be connected to the discharge of *a* and the suction of *b*. The pressure between this pipe *d* and the suction of pump *a* would be 250 pounds, and the pressure from the discharge of *b* to this central pipe would be 250 pounds.

Assume that the discharge *e* of the pump *b* is the positive line, and the suction *f* of pump *a* is the negative line of an electric circuit and the central pipe *d* the positive-negative, or neutral.

Suppose that a waterwheel *g* which requires a certain stream of water at a pressure of 250 pounds is connected between the positive pipe and the neutral, the wheel would run, drawing its power from the positive pipe and delivering to the neutral. The circuit being from the discharge of pump *b* through the wheel, back through the neutral pipe to the suction of pump *b*, where the pressure is again raised to 250 pounds, drawing no water from the pump *a*. Should a second waterwheel *h* be installed of equal capacity to *g*, with its intake connected to the neutral pipe *d* directly opposite the discharge of wheel *g*, there would be a circuit from the discharge of pump *b* through wheel *g* to the neutral pipe *d*, thence through wheel *h* to the negative pipe *f* and on to the suction of pump *a*, where the pressure would again be raised to 500 pounds through the two pumps. There would be no current in the neutral, as there could not be a current flowing both ways at the same time in one pipe. Connecting a third wheel *i* of a certain capacity, to the positive and neutral, and a fourth wheel *j* of a smaller capacity and not directly opposite, from the neutral to the negative line, the circuit would then be the same as before for *g* and *h* wheels, but for *i* and *j* wheels would be from the positive through *i* wheel to the neutral, on through the neutral to the opening for the *j* wheel where it would split, only that amount necessary to run *j* would pass through the intake and the wheel to the negative, and so on to pump *a*, the balance passing through the neutral back to the suction of the pump *b*. Wheels could be added at will up to the capacity of the two pumps, and only that amount would pass through the neutral pipe which was in excess of the amount used by one of the outside pipes over the other, the direction of flow depending on which carried the heaviest connected load. The result on the pumps being that the pump which was connected to this pipe would be carrying the heaviest load. To get the full benefit then of the 500 pounds, it is necessary to see that the two outside lines are practically balanced and as little as possible returning through the neutral.

The action of electricity to all practical purposes is the same as that of water, as described above.

To see what will be gained by a change to the three-wire system, start with the following formula for determining the amount of copper necessary for the transmission of electric current:

$$\text{Circular mils} = \frac{\text{Dist. in feet} \times \text{Watts} \times 21.60}{\text{per cent. loss} \times \text{voltage squared}}$$

Then since in any given problem the distance is a constant, the load is a constant, 21.60 is a constant, and the desired loss is a constant. Circular mils $= \frac{\text{a constant}}{\text{voltage squared}}$. In the case of 500 volts and 250 volts the ratio would be

$$\frac{\text{a constant}}{\text{500 squared}} \div \frac{\text{a constant}}{\text{250 squared}} = \frac{1}{4},$$

or 500 volts requires wire only one-quarter the size that is required by 250 volts to transmit a given power a given distance with a given line loss.

Two 250-volt generators running in series develop 500 volts across the outside wires, with 250 volts from either to the neutral. Therefore, the size of the wire required for a 250–500-volt three-wire system, working under perfect conditions, need be only one-quarter the size required for a 250-volt system simple, and equal to that used in a 500-volt simple. But as most 500-volt and 250-volt systems are operated with a rail return, and the three-wire system must have a copper return, the length of the wire will be double that required for the ground return. The weight of the copper will therefore be one-half that required for the 250-volt system and double that required for the 500-volt system simple.

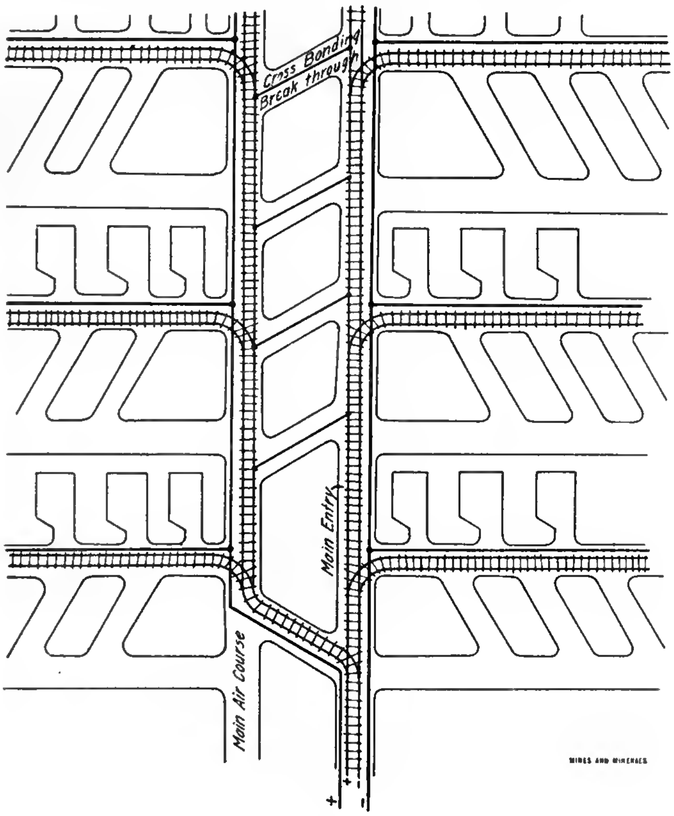


FIG. 2

It should be understood, however, that the necessity for bonding the rail is not done away with, as the rail serves as the neutral wire between the two generators.

In Fig. 2 is shown a sketch of the general plan for wiring a mine with this system. Details, of course, must be worked out on the ground as conditions demand. These would differ considerably in mines operating only a few mining machines and two or three locomotives, from those operating several mining machines and locomotives. In the smaller mines it could not be reasonably expected that the working in alternate cross-entries off the air-course, nor off the main entry, would be done at the same time, thus giving a fairly well-balanced load. The chances would, however, be good that the workings off the main air-course would equal those off the main entry, or very nearly.

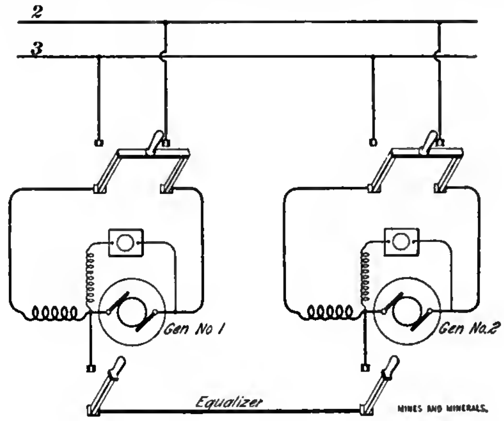


FIG. 3

Therefore, in order to balance the load, which is necessary to get the full benefit of the 500 volts, the one wire should be brought out the main air-course feeding the cross-entries off of it, while the other should be hung in the main entry, taking care of the workings off of it, with the air-course track bonded to the main entry track at each breakthrough. As there is a wire necessary in these entries with the 250-volt system, nothing need be added for the change, except the cross-bonding in the breakthroughs, sectional insulators at each point where the main-entry line is connected to the air-course line, and a different connection in the power plant.

In the larger mines, where the chances are that alternate entries would be worked together, both positive and negative wires should be strung in the air-course and the main entry, and probably in the more important cross-entries, depending of course on the amount of power required, taps being taken from the positive and negative wires, respectively, to feed alternate cross-entries, bonding the track at every available point.

It could not be claimed that under mine conditions the full advantage of the 500-volt transmission could be gained, but the

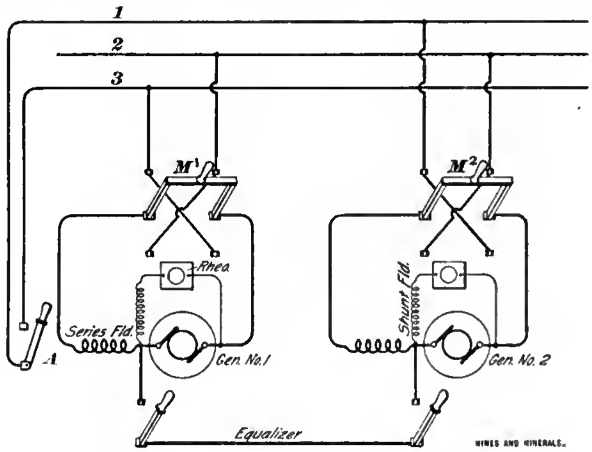


FIG. 4

fact remains that under no conditions could the line loss exceed that of the 250-volts simple, and it could under more or less favorable conditions be made to approach that of the 500-volt system, or one-quarter that of the 250-volt system, with no more danger to life and no change in the connected machinery.

The changes that would be necessary in the power plant would not be extensive, and after once made could be changed

at any time to the old system by simply throwing a switch, in case the system proved unsatisfactory.

Fig. 3 shows a sketch of a standard double-pole switchboard for a 250-volt plant? Comparing this with Fig. 4, which would be the connection for a three-wire board, it will be seen that all that would be necessary to make the change, would be another set of terminals for each switch, thus making double-pole double-throw switches, the cross-connections between the dead terminals, one extra bus-bar and connections to switches, and the short-circuiting switch between the positive and negative bus-bars. With this connection, by throwing in switch A, one generator switch, and leaving the other out, either machine can be made to feed both lines, giving 250 volts simple, and allow for the overhaulage of the other machine. With switch A still in, the equalizer switches in, M^1 up and M^2 up, the two machines may be run in parallel, giving 250 volts simple. With switch A out, the equalizer switches out, M^1 up and M^2 down, or vice versa, the two machines will run in parallel, giving the three-wire 250–500-volt results. With double-throw feeder switches any line may be connected to either bus-bar to equalize the load.

Going farther, should there be three generators in the power plant, two could be run in parallel with the third in series with these two. In which case that line to which both generators were connected should feed all outside machinery and that inside machinery which is close to the generating station, as that part of the load that is out of balance comes back on the neutral wire with a line loss equal to the 250-volt system, for which reason load enough to take care of the third generator should be connected as close as possible, as from the formula it will be seen that the line loss is directly proportional to the distance of transmission.

COAL MINING NOTES

The No. 2 colliery of the Kingston Coal Co., at Kingston, Pa., during the year 1910, mined and shipped 1,020,405.14 tons. This being the first instance in the history of the anthracite region for one breaker to prepare for shipment over 1,000,000 tons. The total shipment of the Kingston Coal Co. for 1910, was 2,131,672.07 tons.

Announcement was made that the holding company of the railroad anthracite producers, known as the Temple Coal and Iron Co., would be dissolved by court order. This seems to be premature as the company will await dissolution until a decision is rendered by the Supreme Court on appeal.

In the January issue of Mines and Minerals it was stated that the United States Department of Justice contended that the "Contract labor law" of Alabama was unconstitutional, even though the Alabama court decided differently. The United States Supreme Court has decided that the law was unconstitutional, and thus wipes out Alabama's, Kentucky's, and Tennessee's legalized grafting, so far as mining is concerned.

What can be said of a coal mine operated under the following conditions; the word system not being permissible? The rooms are driven in pairs with 100-foot pillars between each pair. The pillars between the rooms are drawn "as far as possible" by slabbing as soon as the limit is reached. Sometimes a foot is removed in this way and sometimes nothing. No attempt at all is made to recover the 100-foot pillars and the entry is abandoned when driven "far enough." Perhaps 25 per cent. of the coal in the ground is recovered and perhaps not, but who cares. It is "cheap, government" land of which we have an "inexhaustible supply." Also as there are some 91,000,000 stockholders in the corporation owning the land, the United States Government, who is going to stand up for their rights?

The coroner's verdict in the inquest which sat upon the accident of November 8, at Delagua, Colo., has been rendered and comes as something of a surprise. The jury says that "at the time of the explosion, the mine contained no coal dust of a dangerous character, and such as did exist had been thoroughly sprinkled 2 days before the explosion." How can non-existent dust be sprinkled, or is this to be taken to mean that at one time there was no dust in the mine, or that there was coal dust but it was not dangerous? However, the jury fully exonerated the company from all blame, and in this we believe they acted rightly when all the facts in the case are considered.

In Northern Colorado, where the lignite coal not infrequently carries as high as 20 per cent. of moisture and from 10 to 20 per cent. of ash, "gob fires" are of universal occurrence. In the southeastern part of the state, where the bituminous coal averages some 2 per cent. in moisture and 8 per cent. in ash, gob fires are practically unknown. In both fields the sulphur is well under 1 per cent. No satisfactory theory has been presented to account for the prevalence of fires in the one field and not in the other. A solution of this problem would be of interest.

S. K. Smith writes an interesting article for the *Black Diamond*, in which he concludes that there are from three to four times too many mines in actual operation. Mr. Beebe indorses Mr. Smith's conclusions but cannot agree with his deductions as to the natural remedy. If, he says, big companies were more uniformly profitable or satisfactory, and if consolidations reduced the cost of doing business, or even tended toward the solution of the economic problem involved, he might agree with consolidation of mines as a panacea.

The Interstate Commerce Commission has rendered a decision in the case of the Anaconda Mining Co., of Montana, against the Erie and other eastern roads in which discrimination was charged in maintaining two rates on coke from the ovens in West Virginia and Pennsylvania to Chicago. These rates are $2.65 per net ton "on coke," and $2.35 per net ton "on coke for use in blast furnaces for smelting iron from ores." The commission holds "the rate of $2.65 per net ton applying on coke was, and is, a low rate for the services performed, and that maintenance of a lower rate on coke for use in blast furnaces for smelting iron from the ores, while improper, did not subject the complainants to undue discrimination and is not the basis for awards of damages in these cases." The decision continues that "smelters of copper and iron do not compete in any proper or ordinary sense of the term, and the complainants have suffered no damage."

In consequence, the smelters of the Northern Rocky Mountain region will be compelled to pay 30 cents above the minimum rate for their coke or adopt the local product. The justice of this decision can hardly be questioned if coke destined for points beyond Chicago is, as is ordinarily the case, loaded in box cars, while that intended for use in the blast furnaces in and around Chicago is loaded in open-topped "racks." The proportion of live to dead load is so much greater when racks are used that the railroads can well afford to make a lower rate. Likewise there is usually a return load of iron ore immediately available for the "racks" which can be dropped at Pittsburg, only 75 miles from the Connellsville region, whereas the "boxes" may have to be hauled "light" several hundred miles from the smelter before a return load is obtainable. All these factors of live weight, return load and the like enter into rate making. On the other hand, for coke loaded in the same class or type of car we cannot see but what the rates should be the same. Eastern shippers of coal have been familiar for many years with the three rates on "gas coal for steam purposes," "gas coal for use in gas works," "and gas coal intended for points beyond the Capes," that is, for export.

Shell cameos, onyx, agate, and other semiprecious stones, cut, but not set, and suitable to be made into jewelry, are dutiable under paragraph 435, tariff act of 1897, as precious stones, cut.

EQUALIZATION OF FUELS*

H. K. Meyers

At the present there is much discussion on the question of purchasing coal on analysis and it has met with a large amount of objection on the part of the shipper, principally for the reason that he has been severely penalized.

Methods for Determining the Comparative Values of Fuels of Various Analyses

In general, the only equity in the specifications in the past has been in the number of British thermal units furnished. The other penalties have been arbitary and fail to provide for premiums in the event of supplying coal of a higher grade than agreed.

Most every producer of coal desires an analysis which shows up well on paper, regardless of its truthfulness, and until such time as the operator is willing to accept a general average of analyses as the standard, he is apt to be penalized when he sells on what might be termed the modern basis of purchasing fuel. When the producer learns the heat value of his coal thoroughly, there is no reason why he cannot dispose of his output more advantageously and satisfactorily.

Practically all raw products except coal are sold at a price based on the intrinsic value of their constituent parts.

Gold, silver, copper, lead, zinc, and iron ores bring a price in the market based on their metallic content, and are subject to penalties for impurities and deleterious matter associated with them beyond a fixed percentage.

The basis of settlement is their actual marketable wealth and there is no reason why coal should not be sold on the same basis.

After the various ores are delivered and settlement made, based on analyses, the responsibility of the shipper ceases; but if the purchaser sees fit to operate his smelter so that a large amount of valuable metal goes into the slag, it is of no consequence to the shipper. In order that the smelter may receive the maximum returns from the ores bought, he must have the proper apparatus suitable for the purpose, and in the event of having a fixed inflexible smelting plant, one in which a variety of ores cannot be used satisfactorily, he must confine his purchases to ores suited to his methods. If he purchased ores without a special analyses and looked only to the value of the resulting product, he would place the blame on the grade of ore furnished regardless of bis inefficient help and methods.

This would be an impossible condition in purchasing ores, yet it is practically the condition existing today in the sale of coal.

Probably 95 per cent. of the coal sold at the present time is not analyzed. The variety of boilers and fireboxes under and in which the coal is burned is many times the variety of smelters, and the persons doing the firing are not all gifted with brains.

When the boiler fails to produce steam satisfactorily or the engine fails to do the work required, the trouble is blamed on the fuel. From the complaints registered, one would think that the coal in a mine has a wide range of heating value, although it really is quite constant.

When troubles of the above nature arise, the purchaser immediately demands an adjustment, which is invariably allowed because the producer does not know his product sufficiently well to confirm its value.

Had this fuel been sold on a guaranteed analysis, such objections would not arise, for the analytical determination of the chemist (from whose rulings there is recourse through check analyses) would at once locate the cause. Under the new method it is as important for the purchaser to know the fire and engine-room conditions as for the operator to know the analysis of the coal he is shipping.

There is no doubt but that the purchaser of coal is entitled to the delivery of a certain amount of heat-producing substance per ton, and since the range of calorific powers of the various coals is wide, there should be some general equitable method of equalizing them. The purchaser is interested in obtaining the maximum of energy at the minimum of cost.

Where coal is purchased on analysis, it is customary to base settlements on dry coal; but this is erroneous, because some coals that are high in moisture in their natural state, when dried will compare well with their low moisture competitors. The purchaser of such coals must burn a certain amount of real fuel in evaporating this excess moisture.

The only equitable way of comparing a variety of fuels is by using the analyses obtained at the point of consumption or where they might be termed "commercial coals."

The ideal fuel would be one without moisture, ash or sulphur, and in making the following calculations, all fuels are figured back to the ideal, charging up to the fuel under consideration, the actual cost of the contained impurities.

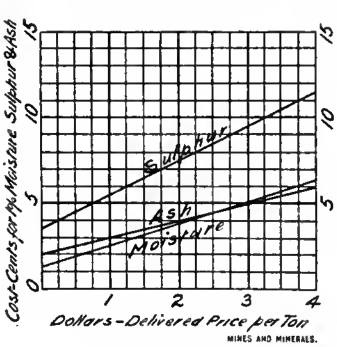


FIG. 1

In making the basic calculations, it was considered that the handling of the ash was an item of expense in addition to its cost, that a certain quantity of water had a first cost and that it required fuel to drive it off, that the sulphur in forming clinker picked up its equivalent in weight of good fuel although it was figured that one-half of its weight was volatile. The item of reduced output due to firing boilers with the impurities was not considered.

With the above conditions, the chart, Fig. 1, was calculated and can be used for comparison of coals, oil, gas, etc. In applying this chart, the delivered price should be used and probably the better method would be to determine the cost per million of British thermal units.

COST IN CENTS, FOR 1 PER CENT. MOISTURE, SULPHUR, AND ASH—VARIOUS PRICED COAL

	Delivered price per ton, in dollars	1.00	2.00	3.00	4.00	Remarks
Moisture	First cost	1.00	2.00	3.00	4.00	1% fuel evaporates 4% water
	Fuel for evaporation	.25	.50	.75	1.00	
	Firing	1.25	1.25	1.25	1.25	
	Total	2.50	3.75	5.00	6.25	
Sulphur	First cost	1.00	2.00	3.00	4.00	1%S. wastes 1% coal ½ S. is volatile
	Fuel wasted	1.00	2.00	3.00	4.00	
	Firing	2.00	2.00	2.00	2.00	
	Removing ash	1.50	1.50	1.50	1.50	
	Total	5.50	7.50	9.50	11.50	
Ash	First cost	1.00	2.00	3.00	4.00	Firing coal, $1 per ton. Removing ash, $1 per ton of ash
	Firing	1.00	1.00	1.00	1.00	
	Removing ash	1.00	1.00	1.00	1.00	
	Total	3.00	4.00	5.00	6.00	

In making comparisons, there must be added to the quoted price delivered the various cash values which the impurities represent, based on the price quoted.

The resulting sums for each coal considered must be divided by the number of British thermal units per ton, guaranteed and expressed in millions and decimals thereof.

The various quotients show the cost per million British thermal units and the lowest. everything considered, would be the one to purchase.

Suppose, for example, a coal whose proximate analysis gave the following: Moisture, 1.5 per cent.; ash, 6 per cent.; sulphur 1 per cent.

* Read before the Winter Meeting of the Coal Mining Institute of America.

Guaranteed British thermal units 14,000 per pound of commercial coal and the delivered price $3 per ton net.

From the chart are obtained the cost of the impurities in cents per ton:

	Cents
1.5 per cent. moisture	7.5
6.0 per cent. ash	30.0
1.0 per cent. sulphur	9.5
Total	47.0

If to this is added the quoted price per ton, the equalized price $3.47 is obtained, and since the coal was guaranteed to contain 14,000 British thermal units per pound, one net ton would contain $2,000 \times 14,000$, or 28,000,000 British thermal units and the cost per million British thermal units would be $\frac{1}{28}$ of $3.47, or 12.4 cents.

In the event the shipper guarantees the above analysis and upon delivery it is found that the coal contains 2 per cent. moisture, 7 per cent. ash, and 1.5 per cent. sulphur, it is only necessary to compute by the same method the equalized price and determine the cost per million British thermal units, the difference being a penalty.

This same chart gives the new equalized price of $3.59 and since the coal contained only 13,500 British thermal units per pound or 27,000,000 per ton, the actual cost per million would be $\frac{1}{27}$ of $3.59, or 13.3 cents.

This is in excess of the agreed price, the difference between 12.4 cents and 13.3 cents, or .9 cent per million British thermal units, the penalty.

Since the coal delivered contained a total of 27,000,000 British thermal units per ton the penalty would be .9 cent $\times 27 = 24.3$ cents per ton and the settlement price would be $2.757. The 24.3 cents penalty would have appeared as a premium if the various percentages of impurities had been less than the guaranteed analysis by the same amounts.

It might seem that this penalty is heavy; however, a glance shows that the total impurities are 2 per cent. in excess and the heat units 3½ per cent. less than the amount guaranteed, making a total of 5.5 per cent. that represents a first cost of 16.5 cents without taking into consideration the extra cost of handling the impurities and the resulting ash. This deduction is not extravagant. It will be observed that the method advocated places every coal on its exact level and shows its comparative intrinsic value.

At equal prices per ton, the operator having the best coal had, and always will have, a decided advantage, but if he is sure that his high-grade fuel is going to be recognized (and it must be in the event of purchasing being done by analysis), he will strive to obtain the maximum price through proper preparation for market. Some operators claim that it will increase the cost of fuel to the consumer, but this is not a logical conclusion although it will be the means of increasing the price per ton of the high-grade coals.

Such a method really makes actual fuel have a constant price and the consumer has ample protection in buying, but he has no recourse for troubles due to inefficiency of plant, as is too often the case under present conditions.

By using this method, after the contract is given, a chart can be constructed based on the actual contract price from which can be readily determined the penalties and premiums.

Every operator should thoroughly sample his mine and his shipments and then make his bid in accordance with the average analyses, and if allotted the contract he will doubtless find that the results are more satisfactory than under present prevailing methods.

Several years ago, the writer sampled a mine and a contract was based thereon. Over 100,000 tons of coal was shipped on this contract covering a period of 12 months and at the end the premiums and penalties practically balanced. At times there may be differences either way, possibly due to climatic changes, but they will offset each other.

In the acceptance of bids, the purchaser should note the various guarantees, and in the event of a bidder showing no general knowledge of the fuel offered he should discard his quotation and save future trouble.

Fuels of widely different natures may show the same calorific intensity and yet they are not equally valuable to the purchaser. Boiler conditions may be such that there is a maximum grate area which requires low-volatile coal whereas with small grate area, high-volatile coal must be used in order that maximum economy may be obtained.

In the New England States, they have been burning Cumberland coal many years, and one will find large grate areas under all the boilers. If they were to purchase Pittsburg coal there would be no trouble with the steaming, but the quantity consumed would be extravagantly large.

If the method described was to be used in some of the western markets where their fuels carry a large percentage of moisture, it would be found that a consumer could frequently afford to pay a higher freight rate on a low moisture and otherwise superior fuel.

The probability is that most of the fault found with coal comes from the fire room. It is much easier for the fireman to blame the fuel than to clean his boiler.

If the coal is sold on analysis, this series of troubles is sidetracked and there is no responsibility placed on the shipper, real or imaginary, after delivery has been made.

OIL OPERATIONS IN TAMPICO

By Consul Clarence A. Miller, Tampico

During the last few years the vicinity of Tampico, Mexico, has been explored with the result that the gulf coast of Mexico has possibilities of becoming an oil field. Several oil companies, including the largest and most powerful, have been engaged in the work of exploring and developing.

Until recently the only success obtained in the immediate vicinity of Tampico was that of the Mexican Petroleum and Development Co. in what is known as the Ebano field, about 35 miles from Tampico. The Pearson interests developed some wells in the Veracruz district, and had some little success in the Tuxpam region. During the last few months the Huasteca Oil Co. met with some success in the Juan Casianos field, and recently the Mexican Fuel Oil Co. brought in a well at Topila. During July, 1910, the East Coast Oil Co., representing the Harriman interests, were successful with a well at Topila and another at Panuco.

The companies operating in this section are: The Mexican Petroleum Co. of California; the Huasteca Petroleum Co., owned by the Doheny interests, and an allied company of the Mexican Petroleum Co. of California; the East Coast Oil Co., which has as yet no corporate existence, but which represents the Southern Pacific or Harriman interests; the Mexican Fuel Co., a Waters-Pierce Oil Co. concern; the Mexican Fuel Oil Co., incorporated under the laws of West Virginia; the Dos Banderos Oil and Gas Co., an Arizona corporation; the American International Fuel and Petroleum Co., which was organized under Delaware statutes; a California corporation, the Tampico Petroleum Co.; the Standard Oil Co. of Mexico, with home offices in London; the Standard Oil Co. of England, with the same official personnel as the preceding company; the Hidalgo Petroleum Co., composed of California parties; the Tampico Oil Co., Ltd., an English organization. The operations of S. Pearsons Sons & Co. have been very extensive.

The Electra Petroleum Co. is being formed, under English law, to develop leased land near Juan Casianos. A company is organizing in California for work on a hacienda 20 miles up the Panuco River from Tampico, and a Salt Lake City company, which has a fruit farm near Caracol, has oil-drilling machinery on the way for sinking a well about 15 miles from Tampico.

Trade Notices

The Western Electric Co. has now disposed of the greater part of its Chicago surplus real estate, which consisted of its Polk Street and Clinton Street properties. Nearly $4,000,000 which was received for the property will be added to its already large working capital and will render financing unnecessary in 1911. Recently the company appropriated $1,000,000 for new buildings at Hawthorne, and doubtless before the present lease expires on the 500,000 square feet of the Clinton Street property which the company still occupies, it will be necessary to erect further new buildings at Hawthorne to carry out the policy of concentration now in effect. Western Electric is employing 24,000 men.—*Wall Street Journal.*

The Jeffrey Mfg. Co., of Columbus, Ohio, whose manufactured articles originally consisted of elevating and conveying machinery, have during their present period of activity added this, that, and the other thing, to their line until a coal mine can be completely equipped from their plant.

Owing to the greatly increased business, the H. W. Johns-Manville Co. announces the removal of its offices, now located at 85 Shelden Street, Houghton, Mich., to more commodious and convenient quarters at 96 Shelden Street, where they will be better prepared to serve their patrons. As in the past, Mr. S. T. Harris, who has been associated with the company for a number of years, will be in charge of the offices at the new address.

B. F. Sturtevant Co., Hyde Park, Mass., has issued bulletin 187 "Economical Fire Room Methods," by F. R. Low, in which the value of the Sturtevant economizer is shown by the tests made at Wood Worsted Mills, Lawrence, Mass. The tests were conducted by A. D. Little and E. G. Bailey, of Boston, Mass.

The Western Electric Co. has just issued bulletin No. 1,116, describing magneto-telephone sets and accessories. This bulletin has been compiled for the purpose of bringing to the attention of telephone companies and telephone users in general the features of Western Electric standard magneto wall and desk sets, together with accessories. It contains 40 pages and is excellently illustrated.

A new booklet "Graphite Products for the Railroads," has just been issued by the Joseph Dixon Crucible Co., of Jersey City, N. J. This, as its name implies, covers the Dixon line of products that are widely used in railroad service.

The Jeffrey Mfg. Co. has opened a new office in the Fourth National Bank Building, Atlanta, Ga., with D. C. Rose, formerly with the Dodge Mfg. Co., as manager. A stock of Jeffrey chains and catalogs will be on hand. This is the tenth Jeffrey branch office in the United States, although there are over 100 Jeffrey agencies in the United States, as well as in the leading commercial centers of the world.

The engineering firm of Dodge, Day & Zimmermann, of Philadelphia, Pa., have added to their organization Walter Loring Webb and James M. Kennedy. Both gentlemen are well known in the engineering world. Mr. Webb is the author of several engineering textbooks, including "The American Civil Engineers' Pocket Book," "Economies of Railroad Construction" and "Problems in the Use and Adjustment of Engineering Instruments."

The Dorr Cyanide Machinery Co. has supplied the Pennsylvania Steel Co. with six classifiers to be used for dewatering magnetic concentrates from the wet-concentration process. This is of interest as showing the additional uses being found for the machine.

The Westinghouse Co. has electrified the Ernestine Mining Co., located about 100 miles from Silver City, N. Mex. Prior to 1905 two companies failed with enormous financial loss in their efforts to operate these mines, although now it is one of the best individual properties in the state.

The Link-Belt Co. is constructing a large washery for the Berwind-White Co., at Berwind, W. Va.

Edgar Allen & Co., Ltd., Imperial Steel Works, Sheffield, England, whose chief American office and warehouse is at 434 West Randolph Street, Chicago, announce that agency arrangements have just been completed with Roehm & Davison, Detroit, Mich.; J. L. Osgood, Erie County Bank Building, Buffalo, N. Y.; John J. Greer & Co., Inc., 207 West Pratt Street, Baltimore, Md., and that stocks of Allen's high-speed and carbon tool steels will be carried at the warehouses of these firms.

COAL MINING INSTITUTE OF AMERICA

The winter meeting of the Coal Mining Institute of America was held in Pittsburg, Pa., December 15 and 16, 1910. According to the secretary, the membership has increased 167 during the half year just closed.

President H. H. Stoek, of the University of Illinois, found it impossible to attend, his address and letter of greeting being read by the secretary. President Stoek also indicated a desire to retire from the office, and at the election which was held during the opening session, Samuel A. Taylor, of Pittsburg, was chosen for the position. The other officers elected were as follows: Vice-Presidents, A. W. Calloway, Punxsutawney, Pa.; J. W. Paul, Pittsburg, Pa., and Elias Phillips, DuBois, Pa.; Secretary-Treasurer, C. L. Fay, Wilkes-Barre, Pa.; Editor of Proceedings, W. E. Fohl, Pittsburg, Pa.; Executive Committee, F. W. Cunningham, Charleroi, Pa.; J. B. Hanford, Morgantown, W. Va.; H. H. Stoek, Urbana, Ill.; Austin King, Scottdale, Pa.

The papers presented were as follows: A Standard Shaft Bottom with Especial Reference to Safety and Economy in a Gaseous Mine, by W. E. Fohl, Pittsburg, Pa.; Mine Fires, by A. G. Morse, Pittsburg, Pa.; Recent Developments in Anthracite Mining Practice, by Charles Enzian, Wilkes-Barre, Pa.; The Best Methods of Removing Coal Pillars, by F. W. Cunningham, Charleroi, Pa.

Mr. Cunningham in his paper gave practical ideas of how pillars should be removed and this led to the discussion of the conservation of coal. Until recently it has been customary to leave the upper bench of the Pittsburg bed in the mine. This "rooster coal," as it is termed locally, protects the mines from roof falls, and is an inferior coal, carrying 12 per cent. ash, and clinkering, making it advisable to separate it from the most excellent lower bench of the Pittsburg bed. The conditions of course were not understood by members of the United States Geological Survey, who have been saying that millions of tons of this coal have been wasted, thus using their mistaken idea to bolster up conservation.

Samuel A. Taylor, said that this coal, which is 60 inches thick, is now being mined in the Panhandle and especially about Bulger.

At the dinner in the Flemish room of the Fort Pitt Hotel, Major Charles Lynch, U. S. A., of the American Red Cross Society, spoke on The Red Cross in Mining Work.

The sentiment of all the talks at the banquet was First Aid to the Injured; and First Aid to Those Not Injured. Among the other speakers of the evening were Jesse K. Johnston, of Charleroi; Dr. W. R. Crane, dean of the Mining Department of Pennsylvania State College; Professor Wadsworth, of the Mining Department of the University of Pittsburg; Lee Ott, of Thomas, W. Va., and A. P. Cameron, of Irwin, Pa.

On the second day the following papers were read: Recent Developments in the Use of Steel Mine Supports, by R. B. Woodworth; Equalization of Fuels, by H. K. Myers, Lecturer on Mining, Carnegie Technical Schools.

In the afternoon the members visited the Westinghouse Electric Co.'s plant.

COAL MINE TRANSPORTATION

Written for Mines and Minerals, by E. B. W.

Next to ventilation, the transportation system is the most important branch of coal mining. When the haulage system is balanced so that there are no delays the entire operation moves as harmoniously as a sewing machine, a condition which appeals to miners and management and even to the mules. The mine may be supplied with the necessary number of men to meet the market demand for coal; it may be equipped with approved modern machinery; the tipple may possess the latest improved tipping and unloading apparatus; nevertheless all these advantages will prove negligible if the transportation system fails to move the coal. From this statement it becomes evident that all branches of coal mining are interdependent, but particularly are they dependent on the physical condition of the tracks and the rolling stock.

Necessity of Proper Road Bed to Obtain Economy in Locomotive Running

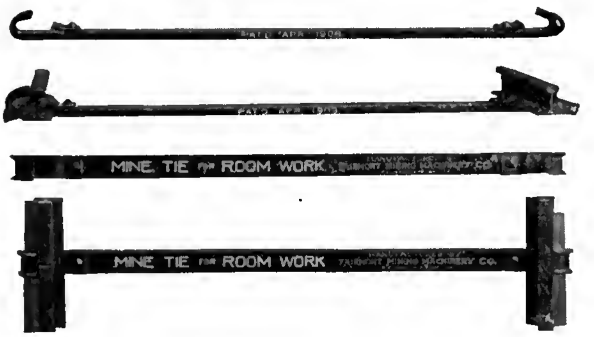


Fig. 1. Steel Tie

To a large extent the cost of car repairs depends on the condition of the mine tracks. Under favorable conditions the car upkeep item will vary from 2 to 4 cents per ton hauled, this, however, is but one item. It is a true statement that the cost of mine haulage is governed by the number of cars of coal moved in a certain time with a given number of men, mules, and locomotives, yet under favorable conditions the minimum cost of haulage varies from 7 to 10 cents per ton. Owing to this expense every mine manager endeavors to economize in this particular department; sometimes with success, but more frequently without, because the subject is not thoroughly analyzed. The first problem involved in economical mine haulage requires for its solution that the tracks be kept in the best possible condition, even to room roads, in fact any neglect of this important matter will be reflected to every department of the operation. Little has been written on the mine roads, and that little is general. Possibly this is due to the assumption that mine roads involve merely manual labor, and if it is so, then it is time for the management to wake up.

The method most frequently adopted to keep the tonnage up to the prescribed mark as the haul increases is to use larger locomotives and haul a greater number of cars than when the haul was shorter. This heavier traffic requires heavier rails, and also that the roadbed be kept in as good condition as a surface railroad. Where 30-pound rails were used a few years ago on the main haulage roads, now 60-pound rails with angle fish-plates are laid on from 6- to 8-inch face hardwood ties spaced 18 inches from center to center. The length of the ties depends on the track gauge, although in every case the ties should extend beyond the track 15 inches, in order to furnish plenty of bite and keep the track in alinement. Under the rails, and particularly at the ends, the ties should be well tamped to prevent spring as the trains pass over them. If this is not done, there will soon be low joints, the spikes will come up, after which will follow track spreading and derailment of cars. Surface-railroad trackmen carefully drain the roadbed in order to preserve the ties and prevent the track springing in wet weather; in fact, where gravel is not available iron-furnace cinders and crushed stone are adopted for ballast. If this practice has been found from experinece to be necessary for surface roads, how much more necessary it is on main haulage roads in mines, where usually mine-track ballast consists of slate and ashes, both of which are sure to go to dust in a short time. Whenever a mine road is thus ballasted and poorly drained, low joints, springy and decayed ties will shortly appear.

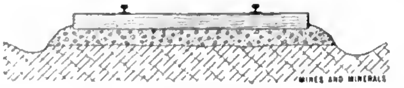


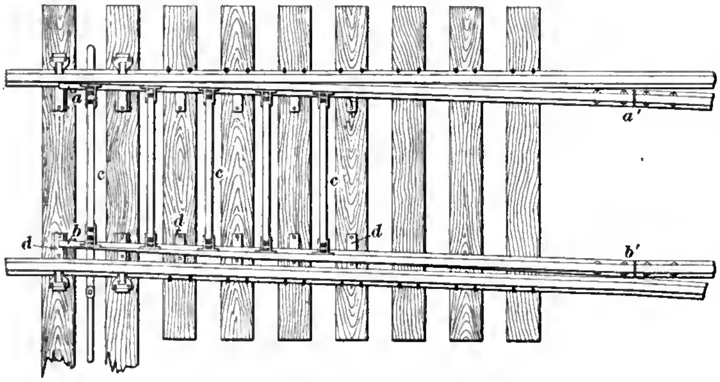
Fig. 2. Road Bed

On cross-entries where 20-pound rails were the heaviest used a few years ago, now 40-pound steel rails are adopted at large operations, and wherever motors travel the ties are given at least 6-inch face and 2-foot centers. Close attention is paid to the curves, partings and room necks on these roads, as it is more thoroughly acknowledged that a stiff well-kept track is easier on live and rolling stock, than a crooked, uneven, dirty, wet road. Usually engineers furnish centers for driving the headings, curves, and room necks, and the management watches to see that the work laid out is done properly by the miners, as a uniform mine on the map looks well, besides the men doing the work are usually contract miners. This remark is not intended for sarcasm but to call attention to the fact that more science and skill is required to lay the curve, turnout and room-neck rails to center than in mechanically furnishing centers with an instrument or making the excavations. If a careful and intelligent oversight of the track laying in these particular places is neglected there will be wear and tear on the rolling stock that might have been avoided.

Only a few years ago, wooden rails, or 16-pound iron rails, furnished the tracks in rooms, now they are almost universally discarded for 20-pound rails laid on small-faced ties and more recently on the steel ties shown in Fig. 1. Too little attention has been given in the past to room tracks, and in some collieries any piece of sapling, green or dry, that could be blazed was thought good enough for room ties and often these were laid with 3- or 4-foot centers. The result of this negligence was that the rails spread, the car wheels fell between the tracks, time was lost and and nine times out of ten the car was wrenched in some part, which hastened it to the repair shop. Good roadbeds are essential to a well-managed mine, for it is difficult, and likewise expensive, to haul a large coal tonnage over poorly-laid tracks. Assuming that the management furnishes good track material, if it is not properly laid on solid roadbeds, the fact will be reflected in the cost of haulage and car repairs. It will be seen therefore that track laying is not entirely mechanical labor, but requires a skill which should be intrusted to experienced men. If good mine roads are advocated on the basis set forth in this article, the writer is apt to be declared impracticable, the suggestions unnecessary and theoretical. Doctor Peters, some years ago, covered this ground thoroughly when he stated that the practical man has infinitely more theories than the theoretical man, but the trouble with them is that they are all wrong. When mines are in their infancy the haulage problem is simple compared to what it will be 10 years later. This very fact makes it essential that the roads over which there is heavy traffic should be kept in alinement, laid on a uniform grade and drained. Sometimes switches are the cause of accident, and it is seldom that they and the frogs are made properly. In order to obtain a good rail frog it is necessary to make it at the plant, and this calls into action the science of the engineer, the boss-track man, and the blacksmith. Split switches should be installed on all main-haulage roads, as well as rail frogs, and it may be further added that there is no economy practiced by cutting down the lead. Whenever

possible, posts should be eliminated from haulways, and if this be not possible then the post should be countersunk, and if the coal spalls, the post should be cemented in place or a concrete post substituted. The main haulage is a permanent road that is to exist probably so long as the mine, therefore all things in connection with it should be permanent. Countersunk posts will not be hit by cars if they jump the track; will not be knocked down in case of an explosion; and lastly, will not injure employes. Haulways should be made at least $2\frac{1}{2}$ feet wider on each side than the top of the car. Then the brakeman will not be knocked from the car when called on to set brakes.

Hauling is done in two different stages in drift mines, and usually four stages in shaft mines, if the pushing or getting cars to the room face be neglected. The gathering is usually done



F gi 3. Split Switch

by mules and drivers, and it is an old saying that the mine boss can control every one but the mule drivers. If all drivers feel inclined, mules will be kept in order and cars delivered on time at the partings; however, if one driver has a grouch the other drivers will be kept waiting on his movements. This may be partly corrected by sending the drivers from the parting in rotation, thus varying the length of haul, and if it does not, the driver should be consulted relative to his ailment. If the driver is sick he should be relieved from duty; if he has a plausible grievance it should be adjusted at once or another run given him, but if he is soldiering a discharge is due him.

The second stage in mine transportation is mechanical, and consists in hauling the cars that have been made into trains on the gathering parting to the shaft bottom or to the tipple. Since the introduction of compressed air and electric motors, rope haulage has been gradually discarded in the United States, and in the consideration of successful motor haulage several propositions appear for discussion.

1. The electric locomotive is capable of hauling so many tons of coal and cars on the mine track, when that is in good solid condition, and when the cars are in good repair and well lubricated. Any one of these factors may entirely upset calculations and instead of a 20-car trip the locomotive may be able to haul only 18 or 19 cars.

2. The second proposition assumes that everything is such that the first proposition may be eliminated from consideration, then time becomes an element and this is dependent on the speed of the locomotive, but more particularly on the drivers having a full trip of cars at the gathering station. While delays at the tipple are not chargeable to the transportation system, nevertheless they are reflected in dollars and cents on the cost of haulage sheet. In order to prevent the motor being delayed at some place along the road, a number of empty mine cars should be held in reserve. The usual delays at the tipple are not so serious as to cripple the system, and then only temporarily, while mine-car loading and waste time cannot be made up. The cost of gathering cars by mules is one of the expensive items in the haulage problem; however, the system is so flexible that if there is additional expense involved for room tracks, mechanical haulage will not appeal greatly to coal operators. Here the steel room ties shown in Fig. 1 feature to advantage both as an improvement on room ties and as a stiffener for tracks. When these are properly laid, gathering locomotives will be found superceding the mule.

The Baldwin Locomotive Works has recently built a special electric locomotive for the Lehigh Valley Coal Co. which is equipped with both traction and gathering reels. The locomotive is shown in Fig. 4.

The frames are of plate steel, placed outside the wheels, and strengthened by angle irons riveted to their outside edges. The front and back bumpers are of angle irons so that the frame combines strength with lightness, and allows more weight to be put into the motors than would be possible if heavier frames were used.

The locomotive is propelled by two Westinghouse motors, wound for 220 volts, that are suspended between the axles. The traction reel which is driven from the rear motor, is placed in a vertical position and carries 400 feet of $\frac{5}{8}$-inch wire rope. By means of clutches, it is possible to drive both the locomotive and reel together, or either may be operated independently.

The gathering reel is mounted in a horizontal position, above the frames of the locomotive. It is operated by an independent motor, and carries 400 feet of double conductor cable, which may be guided over either end.

The application of these two reels enables this locomotive to enter sidings where rails are unbonded and no trolley wires are provided, or to haul cars from the room face while standing with brakes set on the main tracks.

The principal dimensions of this locomotive are as follows: Gauge, 4 feet; wheelbase, 4 feet 8 inches; drivers, diameter, 30 inches; journals, 4 in. × 6 in.; width, 5 feet 9 inches; height, 3 feet $7\frac{1}{2}$ inches; length, 11 feet 8 inches; weight, 15,000 pounds.

TRANSPORTATION IN ALASKA

The continuation of the present prosperity of the mining industry in Alaska is dependent on the cheapening of operating costs by the improvements in means of transportation. The present industrial advancement of inland Alaska is small compared with that which will take place when railway communica-

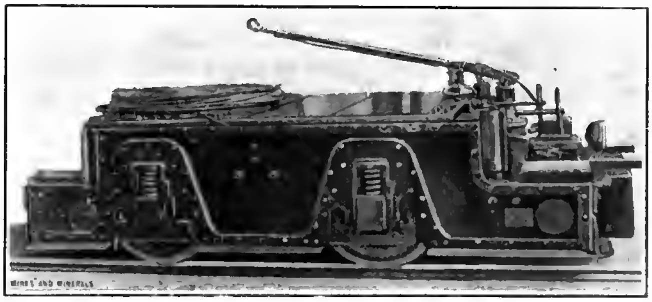


Fig. 4. Baldwin Electric Locomotive

tion with tide water has been established. As it is, the coal, which will furnish the export tonnage necessary to support railways built for opening the coal fields, is of first importance to the entire territory. Alaska has now about 370 miles of railways distributed among nine different systems. Construction work is proceeding on two lines and some progress has been made on a number of other transportation enterprises.

Interest in railway construction now largely centers in the Copper River basin and the Bering River coal field, Kenai Peninsula, and the Susitna and Matanuska basins, and the Yukon-Tanana region.—*United States Geological Survey.*

COMPENSATION FOR INJURY

Written for Mines and Minerals, by Raymond P. Tarr

Classification of Injuries in State of Washington. Plans and Costs of Compensation

There is no question of more real merit before the American people today than that which concerns the maimed and fatally injured. It is an unsettled question because an adequate treatment of such is a tremendous problem. There are various degrees of hazard in the different industries, the business rush is everywhere, and in many cases the complex labor problem increases the hazard. No solution is possible until both capital and labor take hold of the situation whereby each may assume its respective obligations. There certainly should be some satisfactory method of reimbursement for disaster, primarily for the sake of humanity, secondly, in order to bring capital and labor more closely together, and thirdly, in order to obtain conditions which will minimize disaster. Since some industries are more hazardous than others, the compensation could not be alike for all. Since in a discussion of this subject loss of life and injury to limb must be measured in money, one can determine by careful study the actual cost to any particular industry, and thus be prepared with facts upon which to base conclusions that will be fair to that particular industry.

All accidents in any industry can be classified according to responsibility under one of the four following causes:

1. Company negligence.
2. Carelessness of the fellow servant.
3. Carelessness of the individual.
4. Hazard of the industry.

The following resume of the fatal and non-fatal accidents in the coal mines of Washington for the 20-year period ending December 31, 1908, classified as above mentioned, is hereby given for the purpose of showing the causes and effects in this particular industry for this particular state. On account of the length of the period, and also because during the first 10 years the precautions and the mining methods were below par, these results represent a fair average for the country as a whole. Following the resume is a computation which has been made in order to show how much relief would have cost had some sort of mutual insurance plan governed the mining casualties in Washington during the past 20 years. The conclusion is an argument for prevention of accidents and compensation for injury as based upon the results as revealed in this article, and from observations made in various sections of the country.

Mine Explosions.—May 10, 1892, 45 men were killed at Roslyn, in Mine No. 1. This ranks as the worst mine disaster the coal industry has suffered in Washington. Gas was known to exist in the mine and the ventilation was kept in good order. The opinion of Mine Inspector Edmunds was, that "it took place in the cross-cut that was being driven from the airway to the slope, an accumulation of gas having taken place in the manway and cross-cut; that when the hole was drilled through, as it was evident that it had been driven into the roof, cutting the coal at the top as it entered the roof, the gas found its way through this little crack and was ignited by the naked lamps of the men on the slope side, the man that was working in the airway, working toward the slope, had a safety lamp, the others having naked lamps."

Dust greatly augmented the force and intensity of the explosion, which was greatest on the east side of the mine. The miners on the west side were victims of the deadly afterdamp. Since the coroner's jury rendered a verdict "that this explosion was due to improper ventilation," these lives are chargeable to company negligence, although it is known that positive orders had been given not to break through the cross-cut. Inasmuch as the official record stands company negligence, this disaster is so rated here.

The financial loss which results from an explosion is so great that no practical management can afford to poorly ventilate a mine. Miners still employed at Roslyn who were witnesses of that catastrophe emphatically agree that the disaster was due to the carelessness of those at work in the cross-cut. [Mine explosions may be reduced, but they can never be absolutely eradicated. A careful investigation will show that there is as strong desire and effort on the part of mine managers in the United States to reduce these disasters as can be found anywhere else in the world.]

April 9, 1895, an explosion of firedamp at Blue Cañon resulted in the death of 25 men. It seems that this disaster was due to gross carelessness. The report of Inspector Edmunds was that "a hole had been driven into the bottom rock, at the face of the gangway, and it had been charged with giant powder and fired. The hole had been driven in a very improper place. Instead of drilling it in the direction of the gangway, and making it a lifting shot, it was located on the top of the rock, or across the strata, and pointing into the solid, thus making a line of great resistance. The explosive failed to perform the work intended for it, merely making a small cavity on the face of the rock." Such a shot would cause a great concussion, similar to a blown-out shot.

It was stated that the mine was free from gas that morning when the fire boss made his examination, but it is probable that the gas was present when the shot was fired, or a quantity of it was liberated from the strata with the blast, and the heat produced by the stored-up energy of the giant powder, when it failed to perform work in removing rock, must have ignited a pocket or blower of gas. According to the testimony of the men working on opposite shifts, and some of the officials of the company, rumbling noises were heard the day previous under the floor of the gangway.

It was evident that there was not a large quantity of firedamp present, as but few of the bodies were burned. The men in the working places came down and were suffocated by the afterdamp. These 25 deaths are chargeable to the error of the fellow servant.

December 9, 1899, 31 lives were lost in the Carbonado No. 7 mine explosion. This mine gives off considerable firedamp, but is, and always has been, exceptionally well ventilated. It was the unanimous opinion of the State Board of Coal Mine Examiners that "the origin of the explosion occurred by the ignition of a small quantity of gas in some manner unknown, the force due to this raising the dust was undoubtedly the principal factor in the explosion." These 31 lives are chargeable to the hazard of the industry.

April 26, 1907, seven men lost their lives in the Black Diamond Morgan's Slope mine explosion, which, according to the finding of the coroner's jury, "was caused by the concussion and fire produced by an explosion of a pocket of gas, which was brought down by an unavoidable cave, and was ignited by some unknown miner." The disaster was therefore due to unforseen causes, and consequently attributable to the hazard of the industry.

Dust Explosions.—At Coal Creek, October 9, 1894, an explosion of dust ignited by a charge of giant powder caused the deaths of four persons. But for the fact that the floor of the gangway had been well watered the day before, the disaster would undoubtedly have been very much greater. No firedamp has ever been found in this mine. These four lives were lost on account of the carelessness of the individual who fired the hole, and chargeable therefore to fellow servant.

October 1, 1901, 11 lives were lost at the Lawson Mine by an explosion of dust. Investigation proved that a heavy shot had been fired on the bottom, and that about 4 inches of a drill hole was found near the top rock. The finding of the coroner's jury was that "an explosion was caused by two shots being fired one after the other, the second shot igniting the dust created by the first shot." Evidence of poor work on the part of the

individual who tamped these holes would place the responsibility under the head of carelessness of the fellow servant.

December 7, 1904, at Burnett, 17 lives were lost as a result of a blown-out shot which ignited the dust and produced a disastrous explosion in one of the chutes. Twelve men were killed outright, and five were suffocated by afterdamp. The finding of the coroner's jury was that "the men were killed as a result of the explosion of coal dust, caused by heavy shooting." The finding therefore places the responsibility against the fellow servant.

Mine Fires.—In some of the mines of the state the coal is susceptible to spontaneous combustion, and in these mines have occurred disastrous mine fires. The most disastrous mine fire occurred at Franklin, August 24, 1894, when 37 men lost their lives through suffocation by smoke. These men might have escaped, but they had gone to the seat of the fire in the effort to extinguish it, when some incompetent or excited individual stopped the fan and thus cut off the air supply. The men were soon compelled to retreat toward the bottom of the slope. While going through the rock tunnel to the airway, smoke was encountered, which would not have been there had the fan been continued in operation.

Had the exact location of the fire been known the fan might not have been stopped. It is probable that he who was responsible for the execution of this act may have witnessed the stopping of the fan at some previous fire, and have believed that such a precaution was necessary in the case of every mine fire. Since it has not been determined whether the stopping of the fan was due to an incompetent official, or to an excited individual miner, it is but fair to place the responsibility equally under two causes, company negligence and carelessness of the individual.

The second serious mine fire occurred at Franklin, October 14, 1895, when four men lost their lives. A gas blower had been ignited, which in turn ignited the timbers. The men had reached the surface in safety, but the four who had gone below to extinguish the fire, and who had gone in spite of strenuous warnings, perished in the smoke and flame. These four lives are chargeable to the carelessness of the individual.

August 21, 1900, five men perished in a mine fire at Issaquah, which started from a surface brush fire. All but five of the miners escaped after the warning had been given the men to come out. Hazard of the industry is clearly the cause under which these fatalities should be considered.

Individual Gas Explosions.—Seventeen men have been killed in minor gas explosions. Of this number, nine were killed on account of their own carelessness, three by the carelessness of others, and three were due to the hazard of the industry. (Two, causes unknown.)

Powder Explosions.—By this cause four men have been killed through their own carelessness, and two on account of the carelessness of others.

Suffocations.—Seventeen men have been suffocated. In most cases these deaths have been due to the men returning to their working places too soon after firing shots. Of this number, 14 fatalities were due to individual carelessness, two to the hazard of the industry, and one to company negligence.

Fall of Rock.—Out of the 135 fatalities caused by falls of rock or coal, it is positively known that 31 had spurned the advice against continuing at work until the timbers already at hand had been put in place to support the overhanging and dangerous masses, which later fell with fatal results. Thirty-one are thus charged to the carelessness of the individual, 10 to the fellow servant, 91 to hazard of the industry, and three to company negligence.

Runaway Cars or Trips.—Nine trips have wholly or partly broken loose with fatal results, because cables, couplings, or drawbars have broken, been improperly attached, and in some cases defective. Undoubtedly, some of these defects were not reported by the miner, or possibly even not repaired by the company man to whom defects may have been reported. For these reasons it is but fair to distribute the responsibility equally between carelessness of the individual, company negligence, and hazard. Of those killed by cars (besides the nine mentioned), whether by collisions, falling off or under trips, or by being run down, the deaths of 13 are attributed to the carelessness of the individual, 15 to the fellow servant, 15 to the hazard of the industry, and three to company negligence.

Other Accidents.—A large number of causes, such as coal falling off cars, falling timbers, falls, unspragged cars, etc., etc., have killed a large number of men. Under these minor causes 12 men have been killed on account of their own negligence, four through the carelessness of others, and 37 fatalities were due to the hazard of the industry.

Note.—Twenty deaths (two by explosion) from various responsibilities have been charged in proportion to those known, five to the carelessness of the individual, eight to the hazard of the industry, four to fellow servant, and two to company negligence.

Summary of Accidents.—Causes: Explosions, 163; fall of rock, 135; cars and trips, 55; fires, 46; all others, 69.

Responsibilities: Hazard, 195 = 42 per cent.; carelessness, 109 = 23 per cent.; fellow servant, 91 = 19 per cent.; company negligence, 73 = 16 per cent.

During the past 20 years 59 per cent. of all fatalities in Washington were caused by falls of rock and by mine explosions. These same causes produced 59 per cent. of all the deaths in all the coal mines of the United States during the year 1906.

Summary of Outside Fatal Accidents.—Killed by machinery, 9; trips of cars, 4; boiler explosions, 3; falls, etc., 11.

Responsibilities: Hazard, 19; carelessness, 4; fellow servant, 1; company negligence, 3.

Causes of Outside Fatalities.—Six men have been caught in machinery while attending to their duties, and their deaths are here considered chargeable to the hazard of the industry. Two were killed on account of the breaking of machinery and the charge is placed against company negligence in these particular cases. Four men have fallen under trips, the deaths of whom should be placed under the cause of hazard.

The Cost of Fatalities.—Taking the foregoing casualties into account as reliable data, computation has been made on a basis of $2,000 as compensation for each fatality where a widow was left, and in those cases where children have been bereft, $500 for each; and in addition $500 in each case where no widow was left; and since 495 men have lost their lives, leaving 182 widows and 498 fatherless children, a fund of $769,500 would have been necessary to thus compensate those left without means.

Note.—For 4 years the widows and orphans were computed according to the law of averages. This was necessary owing to the condition of the records.

The Cost and Extent of Injuries.—The total number of nonfatal accidents which have been recorded is 1,359. These produced 1,261 inside and 98 outside injuries. Allowing a fair compensation for the various injuries, taken into account according to the following scale they would have cost as follows:

Injury	Number	Compensation	Total
Broken leg	270	$ 300	$81,000
Severe	328	150	49,200
Broken arm	120	150	18,000
Slight	304	20	6,080
Severe burn	173	100	17,300
Slight burn	134	20	2,680
Fractured skull	7	1,000	7,000
Injured spine	4	5,000	20,000
Arm amputated	4	1,500	6,000
Leg amputated	4	2,000	8,000
Hand amputated	8	1,000	8,000
Both eyes out	2	5,000	10,000
One eye out	1	1,000	1,000
Total	1,359		$234,260

How the Burden Might Have Been Carried and What It Would Cost to Carry It at the Same Rate Now.—The grand total

of tons mined was 40,651,289 for the 20-year period, and of men employed 76,628. According to the figures above, the total cost of injury would have amounted to $1,003,760. These figures show an average cost per ton of $.0247, and of $13.10 per man per annum to compensate in this manner. If the state had carried one-half the burden of the hazard, its share would have amounted to 21 per cent., and had the company paid for the remainder of the hazard and the company negligence charge, its share would have been 37 per cent. This would have left to the men 42 per cent., comprising the responsibilities resulting from negligence on the part of individual and fellow servant. For convenience and fairness let us reckon on the following basis, charging to the state, 25 per cent.; to the men, 35 per cent.; to the company, 40 per cent. Accordingly, the results are:

	Cost Per Ton	Cost Per Man Per Annum
To the state per annum	$.006175	$2.75
To the company per annum	.009880	5.50
To the individual per annum	.008645	4.85

Prevention of Accidents.—Prevention of accidents is largely dependent upon the kind of precautionary laws in force, the degree to which they are enforced, and also upon the spirit with which such laws are received, while compensation for accidents must be based upon reliable data, in order to not only afford relief, but also to promote responsibility and thereby react as a great agent for the prevention of accidents. Casualties in this country are greater in proportion to the number of men employed than in like industries in most any other country one might select. Not only this, but in coal mining at least, the ratio of casualties is rapidly on the increase. In spite of these facts we cannot treat the problem as it may be treated in any other country, because the conditions are so different. It is for this reason that we must here deal with carefully prepared facts and figures.

The rigid enforcement of laws framed to insure the safety of the individual and to protect the lives of all others employed, seems the only policy that can help reduce the number and kind of accidents in every sort of hazardous industry. The time is also ripe for men to depart from the false notion that it is unfair (scabby) and useless to complain of fellow workingmen whose actions are wrong. This is a false notion which has assumed such proportions in present-day America, that its results are too frequently disaster and death.

Accordingly, as the laws are made and enforced and the respective responsibilities appreciated, the annual levies will decrease. While the preceeding results show the average of the past, they should be considered as excessive for the future, because there is reason to believe that in the very near future the recommendations resulting from the labors of all investigators will aid in a material reduction in the number and kind of disasters. The recklessness practiced in the use of powder, which causes many mine explosions, and the consequent falls of rock which result as after effects of these explosions, will be the greatest errors to overcome. Since these two causes are practically one, and since they produce the majority of injuries, and because a remedy is easy, the prediction for a reduction is well founded. The belief of those who have studied the question is that the reduction should amount to between 50 per cent. and 75 per cent.

Any contemplated bill aimed to prevent disaster should possess a feature encouraging more care on the part of all concerned, and no feature in any bill could be of much greater value than such a one as would under-rate the amounts to be collected for compensation, because the entire working force would then be made to realize that great precaution is necessary in order to fully recompense for injury. Thus a proper and necessary incentive would be created to help reduce both disaster and levies for compensation. None can afford to be so pessimistic as to not expect such reductions, and those who cannot assume such an attitude have no business seeking employment or engaging in business in any hazardous pursuit. Assuming the reductions to be 30 per cent (and they should go much lower), the amounts necessary to cover requirements according to the foregoing reckoning would become reduced as shown in the following table.

	Cost Per Ton	Cost Per Man Per Annum
To state per annum	$.004322	$1.92
To company per annum	.006916	3.85
To individuals per annum	.006055	3.40

Mr. Richard Newsam, president and mining engineer of the State Mining Board of Illinois, in his article, "Timely Remarks," sincerely but positively states: "We have laws that would help in our state, but we also have some that could be greatly benefited if they were changed. Yet what is the use of making laws if no notice is taken of them?" Naturally, the inference from Mr. Newsam's remarks is fewer but better laws and all enforced. After 55 years of practical mining experience Mr. Newsam states that "he believes 75 per cent. of the lives can be saved that are now being lost in Illinois mining operations, that the only one who can stop the great loss of life in our coal mines today is the miner himself, by living up to the laws, and thinking more of his own life and of the lives of others." There are undoubtedly thousands of others who would testify precisely as Mr. Newsam has done. The sort of logic which these men present must be heard and obeyed, there must be more thought, more duty, more education, more consideration for others, a few good laws, every one alive. These are the requisites to solve not only this, but also every civic problem.

Compensation.—To secure legislation simply to assure compensation for those who are dependent upon those killed, and for those who become injured in hazardous pursuits, is a possibility which may afford relief for those affected, but this is only half the problem. The enactment must be framed to help reduce the number and kind of accidents as well. A cooperative insurance plan might be better than a workingmen's compensation benefit or an employer's liability act. Most men concede that it is well to carry certain kinds of insurance. Thinking men know that it is their duty to assume such responsibility. Since the majority of workingmen possess both intelligence and a high sense of duty, would it not be possible to frame for his benefit some sort of insurance act for his particular industry, which will satisfy both his sense of duty and his intelligence, one equable both to employer and to employe.

In 1907 a carefully prepared bill failed to pass the Illinois legislature. The bill had many meritorious features for the benefit of the workingman. The failure to pass was largely due to the fact that such insurance had to be carried in some casualty company, organized under the laws of the state or admitted to do business in that state. An insurance plan which provides for a minimum of profit ought, when sufficiently improved, to satisfy and become effective. Insurance companies must make a profit, and they can prosper without soliciting business from laboring men. The latter should be able to obtain insurance by some more economical method than that offered by insurance companies.

If a bill were framed making it lawful (after the Illinois bill), viz.: "For any employer to make a contract in writing with any employe, whereby the parties may agree that the employe shall become insured against accident occurring in the course of employment, which results in personal injury or death in accordance with the provisions of this act, and that in consideration of such insurance, the employer shall be relieved from the consequences of acts or omissions, by reason of which he should without such agreement, become liable toward such

employe"; the state might well afford the additional expense for the necessary officials to collect, maintain, and disburse these casualty funds, and also pay one-half the compensation due to hazard responsibilities. For the reasons given it might be a better law if it were provided for the men to enter the agreement as contributors to the insurance plan.

According to our present system, there is financial relief from but one of the four causes mentioned; namely, when an accident is declared by jury as due to company negligence, and under this process but a small portion of this money is actually received by those most dependent upon the awards. Whenever compensation is obtained by law, the disbursement of funds would require a vast amount of consideration. They should be payable in monthly instalments, according to the needs, and extend over as long a time as possible. This would be particularly necessary in cases of large sums due beneficiaries who could easily become victims of fraud or robbery.

The state is a great beneficiary from tax levies in mining operations and it would be a much greater beneficiary with proper compensation laws enacted, whereby legal processes which arise from damage suits were eliminated. It cannot but help working to the benefit of all concerned when the provisions are made for the employer and employe to contribute to a general fund for compensation to the injured. Some great economical change must be brought about in order to make workingmen better satisfied, better protected, more certain to follow their chosen line of work, and what is more important, to develop in them a larger sense of responsibility. If these results are obtained, both the employer and the employe will have the adequate protection to which both are entitled, but greatest of all the benefits will be the results which will come as the individual fully appreciates the feeling of assurance while he proceeds with his daily labors.

MINE CAR CAGING MACHINE

Written for Mines and Minerals

When mine cars are moving smoothly during working periods in mines every one seems satisfied and works with plenty of ginger, but if there is a hitch in any part of the transportation department it is reflected and usually magnified along the entire system. If a car is off the track and trips are delayed, the entire system is immediately disorganized, the miners inside are disappointed at the output, and those men at the surface as well, because hindered in their work.

In order that cars may be kept on the move and delays in hoisting reduced to a minimum, there must be suitable arrangements at the top and bottom landings for running the cars off and on the cage. This "caging" occupies so much time, that in England and on the Continent two- and three-decked cages, requiring expensive landings at the top and bottom of the shafts, are employed. Where there are such landings, of course a greater quantity of mineral can be hoisted at a time, which increases the output. However, in the United States, the output is usually as large from a single shaft with a one-decked cage, owing to the use of larger cars and quicker caging, thus permitting more trips. The caging problem has been studied carefully, and in order to avoid delays at the landing several men are employed. Experienced mining men state that it is economical to expend $1,000 for a positive-acting piece of machinery that will do away with the labor of one man. In the apparatus shown in Fig. 1, which is known as the automatic safety mine-car caging machine, there is a system of levers attached to a shaft situated parallel with and inside the track rails. At the end of this shaft there is a crank lever *A* arranged so as to open or close the horns *B* and *C* in such a manner that they will lock the car wheels and stop the cars.

When the cage reaches the bottom landing its weight coming on the crank lever *A* revolves the shaft sufficiently to work the levers actuating the horns *B* and cause them to open; at the same time the levers attached to horns *C* are moved so as to cause the latter to stand upright, and also the weight *D* attached to the end of a lever arm. This allows the car to run by gravity on the cage, but prevents the second car moving forward past horns *C*. When the cage is raised the weight *D* rotates the central shaft, thereby opening the horns *C*, closing

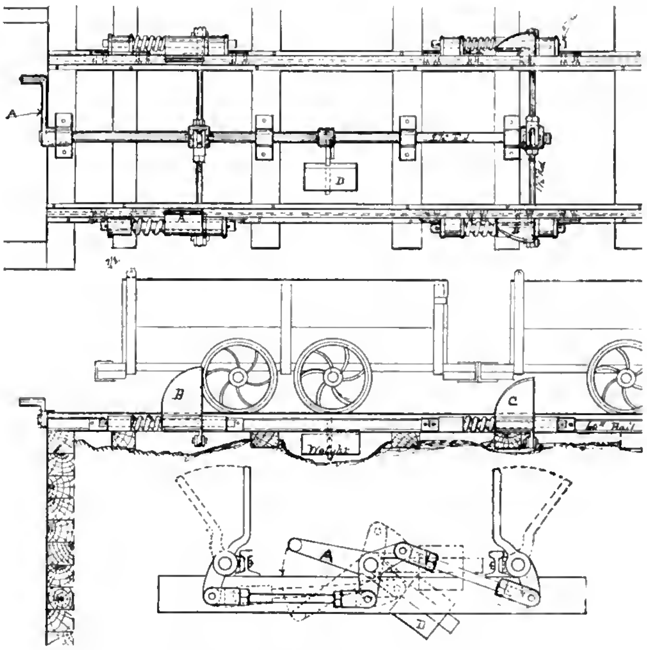


Fig. 1. Automatic Caging Device

the horns *B*, and raising the crank *A* to its original position. The next car to go on the cage then moves forward and is stopped by *B*. A system of hand levers may also be applied to the apparatus if it is desired to use it at intermediate landings. The writer is indebted to the Mining Safety Device Co., of Bowerston, Ohio, for the use of the accompanying cut showing the method of operation of this valuable and interesting adjunct to caging.

EUROPEAN LAWS REGARDING BREATHING APPARATUS

In Austria the law requires a pneumataphore to be kept in order and readiness for every 100 miners employed. The government permits the coal owners the choice of four approved types of breathing apparatus, but the mining office may authorize the use of any new appliance approved in future.

In the different states and districts of Germany the regulations vary according to the natural conditions of the mines, but as evidence of the stringency of the law it may be mentioned that no fewer than 700 sets of breathing appliances are kept ready for use in the Dortmund district alone, and in the Breslau district no mine has less than 2 sets, the government mining engineer being empowered to order any number he deems fit.

In France, all mines employing more than 100 persons underground upon one shift must be furnished with breathing apparatus, the number ranging from 2 to 6 per mine, according to size, gassy nature. number of workers, etc., and double brigades of not less than 8 trained men must be kept for every set of rescue appliances. Some interesting experiments in the utility of safety chambers have been conducted recently in France, and as a result some of the leading colliery companies are putting "blind" galleries, supplied with compressed air, water, food, etc., and signal arrangements, in their pits.

In Holland a number of mine workers must be trained to rescue work, and sets of apparatus must equal 1 for every 50 miners employed.—*United States Consular Report.*

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

PRELIMINARY QUESTIONS

Mine Foremen's Examination Held at Fairmont, W. Va., November 16, 17, 1910

A.—Give place of birth. B.—Are you a resident of West Virginia and a citizen of the United States? C.—Have you testimonials of character? If not give names and addresses of three reputable persons as reference. D.—State the number of years of experience you have had in or about coal mines; and how many of these have been spent in West Virginia. Give the names of the companies for whom you have worked, the mines, and the kind of work performed in each case, in this or other countries.

E.—What quantity of air is required by law in gaseous and non-gaseous mines? Where is such quantity to be circulated? How often, where, and by whom shall this air be measured and recorded?

Ans.—In mines generating firedamp, the minimum circulation required by law is 150 cubic feet of air per minute for each person employed in the mine (Sec. 12); in non-gaseous mines, 100 cubic feet per minute per person (Sec. 11); the same to be circulated around the main and cross-headings and working places; and measured at least twice a month at the inlet, the outlet, and at or near the faces of the advanced headings by the mine foreman, who shall record such measurements in a book kept for such purpose (Sec. 15).

F.—What are the requirements of law in regard to the methods of conducting the ventilating current to the working faces by means of doors, stoppings, overcasts, etc.?

Ans.—See Sec. 12, West Virginia mining law.

G.—To what scale must a mine map be drawn to conform to law? What information should be placed on the map? How are often are mine maps to be extended? Where should they be kept? What useful information should a mine foreman be able to obtain from the map when the mine is approaching old abandoned workings that may contain water or gas, and what precautions should be taken at such times?

Ans.—A scale of 100 or 200 feet to an inch, to be stated on the map. Map must show all shafts, slopes, entries, and airways, with arrows to indicate the direction of air-currents; also, all headings, rooms, pillars, and abandoned places; also, the general inclination of the strata, and all property lines or outcrops of coal within 1,000 feet of any portion of the mine workings. The map must be extended twice a year and not more than 7 months apart. The original map or a true copy of same must be kept at the mine office, and another copy at the office of the district mine inspector. When approaching abandoned workings, a reference to the map should show the exact relation of the workings to each other and their distance apart; but as a precaution, drill holes should be bored 12 feet in advance of the face of the heading (Sec. 15) and flank holes on each side, to insure the supposed thickness of barrier pillar and avoid accident.

H.—What are the legal requirements in regard to traveling-ways and means of entrance and exit from mines? Answer fully.

Ans.—See Secs. 8, 9, 10, and 15 of the West Virginia mining laws.

I.—State what the law requires in regard to safety appliances at shaft mines with reference to cages, ropes, signals, lights, brakes, safety catches, machinery, and methods of inspection.

Ans.—See Secs. 9 and 15 of the mining law.

J.—What dangers are connected with the use of powder for blasting coal in mines? What precautions should be used to overcome these dangers? Where should explosives for use in mines be stored? How much powder is permitted for use of each person per day, and how must it be carried when taken into the mine?

Ans.—Aside from the danger of accidental explosion in the handling of the powder or the charging of holes in blasting, there is the more common danger of a possible windy or blown-out shot occurring, which may stir up a sufficient quantity of fine dust to cause a serious explosion. The danger is more common in the use of black powder in presence of gas or dust or both, owing to the flame produced in its explosion. The precautions that should be observed are: Avoid excessive use of powder; use proper judgment in placing all shots; fire but one shot at a time in any working place, and never fire a shot in a close place where the air is laden with dust and the smoke of other shots; test the place for gas before firing. In many mines the only safe plan is to have all shots examined and fired by competent men, after the workmen have left the mine. Explosives should be stored on the surface in a specially constructed magazine, located at a safe distance from the shaft and other buildings. Only sufficient powder for reasonable use in one shift may be taken into the mine by any person, and this must be carried in a metallic canister that will hold not to exceed 5 pounds (Sec. 11).

K.—State what persons and of what age can be employed in coal mines in Virginia. What is the duty of the mine foreman when an employe disobeys an order resulting in loss of life or injury to persons or property?

Ans.—Males only, who are 14 years of age or over (Sec. 17). In case of accident causing serious personal injury the mine foreman, in the absence of the superintendent, must notify promptly the mine inspector of the district, and in case of loss of life, he must also notify the coroner of the county, or a justice of the peace (Sec. 20).

MINE FOREMEN'S EXAMINATION

Ques. 1.—What qualifications must a mine foreman possess, and what duties must he perform to meet the legal requirements in West Virginia?

Ans.—The necessary qualifications are: Ability and skill in handling men; sound judgment in the planning and direction of work; a firm and resolute character; patience, forbearance, and consideration for others; a thorough knowledge of all work in hand and machinery employed, together with a practical experience in the operation of mines. The legal duties of a mine foreman are given in Sec. 15 of the West Virginia mining law.

Ques. 2.—Name the chief causes of accidents in coal mines, and the methods you as a mine foreman would adopt to avoid the liability to such accidents. Answer fully.

Ans.—The common causes of mine accidents are: Falls of roof and coal; mine cars; explosion of gas or dust; explosion of powder; falling down shafts or slopes; overcome by gas; kicked by mules; and other minor causes. Require a strict compliance with the state mining law and the mine regulations and exact suitable penalties for violation of these rules. Keep on hand an ample stock of all needed timber and other supplies and arrange for their prompt delivery to the men as ordered. Employ good fire bosses to examine the mine thoroughly before each shift and maintain efficient ventilation in all working places. Appoint competent men to examine and, if necessary, to fire all shots. See that miners timber their places promptly as required, and use proper caution in mining their coal.

Ques. 3.—The distance between two pairs of cross-entries is 600 feet; from the main-entry pillar to the boundary is 1,000 feet; the thickness of the seam is 6 feet. How many tons of 2,000 pounds each may be gotten from this block of coal, assuming that a cubic yard of the coal weighs 1 ton, and allowing 25 per cent. to be left for pillars?

Ans.— $\frac{600 \times 1{,}000 \times 6}{27}(1 - .25) = 100{,}000$ tons.

Ques. 4.—To what use is electricity applied in the operation of a coal mine? What are the advantages and disadvantages of using electricity for power or other purposes? State the conditions under which you would recommend the use of elec-

tricity for power in mines. What dangers are attendant on its use, and what precautions should be taken to insure safety to those employed in the mine?

Ans.—Electricity is used to haul coal, operate drills and coal-cutting machines, and in some cases pumps and fans. It is also used for signaling, lighting, telephoning, and for firing blasts. For power and other purposes, electricity furnishes an elastic system that is readily installed and easily changed or extended to suit the variable conditions of mining; its adoption, however, requires the installation and maintenance of expensive machines and the employment of one or more electricians to superintend and operate them; and exposes the mine employes to danger. Electricity may be recommended for power purposes in large mines having long haulage over fairly level roads, and where the coal is drilled and blasted or mining machines are employed. The dangers and precautions to be taken in the use of electricity to insure safety are explained in answer to Ques. 14, page 53, Mines and Minerals, August, 1910.

Ques. 5.—What are the causes of mine fires? Explain how a fire may occur in a mine spontaneously, and what means you would employ to prevent this; or if started how you would extinguish it with safety to the men employed.

Ans.—The common causes of mine fires are given in answer to Ques. 11, page 734, Mines and Minerals, July, 1910. Spontaneous ignition of carbonaceous matter is the result of the absorption of oxygen by the carbon, and the generation of chemical heat in the disintegration of pyrites (sulphide of iron) in presence of moisture. The oxygen absorbed is brought into direct contact with the volatile hydrocarbons of the coal and combustion follows. The best means of preventing this danger is to remove as far as possible its cause, by loading out all fine coal and slack from the mine, and by reducing the heat by the thorough ventilation of the mine. See also Ques. 6, page 52, Mines and Minerals, August, 1910. The method to be employed for the extinction of a mine fire depends on the character and extent of the fire. A gob fire of small proportions may be loaded out before it spreads. Gas feeders in the floor sometimes become ignited and draw the flame back under the gob where it is difficult to reach. These may at times be extinguished by a small stick of dynamite exploded on the floor close to the place. Fires that have gained some headway must be approached from the fresh-air side to avoid the workmen being overcome by the gases produced. The general plan of procedure is to reduce the air supply as far as practicable, and apply water, or carbon dioxide from a chemical engine, or other means to extinguish the fire. When this fails, the fire must be sealed off by building air-tight stoppings in all openings leading thereto, and allowing these to remain till the fire is apparently extinct. Carbon dioxide gas is sometimes injected into the enclosed space behind the stoppings. As a last resort the mine or a portion of it is flooded.

Ques. 6.—Specify the conditions that must be fulfilled in order to secure good ventilation in a mine employing a large number of men. Why is the ventilation of a mine necessary?

Ans.—This question is fully answered in reply to Ques. 4, page 279, December, 1910; and Ques. 3, page 381, January, 1911, Mines and Minerals.

Ques. 7.—How would you determine the quantity of air passing in a mine at any given time? Where the airway is 12 feet 6 inches wide and 5 feet 6 inches high, and the velocity of the air is 375 feet per minute, what is the area of the airway and the quantity of air passing per minute?

Ans.—The quantity of air passing per minute in an airway is determined by measuring the sectional area of the airway, at a chosen point, and multiplying this area (square feet) by the observed reading of the anemometer (feet per minute), as more fully explained in reply to Ques. 8, page 319, Mines and Minerals, December, 1909. The area of the given airway here is $12.5 \times 5.5 = 68.75$ square feet. The quantity of air passing is then $68.75 \times 375 = 25,781.25$, say 25,800 cubic feet per minute.

Ques. 8.—State to what extent the air-current in a mine can be split in the efficient ventilation of the working places. What advantages are obtained by splitting air-currents over a single continuous current in ventilating a mine? Compare the advantages and disadvantages of the different means of producing ventilation in mines, with respect to their effectiveness, safety, and economy.

Ans.—The first two parts of this question are fully answered in the reply to Ques. 6, page 279, Mines and Minerals, December, 1910. The most reliable means of ventilation in mining operations is the centrifugal fan, which, when properly designed to meet actual conditions, will yield the largest volume of air at least expenditure. In fan ventilation the circulation is under better control, and can be readily increased or decreased by regulating the speed of the fan. The mine furnace is an efficient means of ventilation in deep shafts, but is not as easily controlled, requires more constant attendance, and consumes more coal for the same air volume and water gauge produced. In furnace ventilation there is greater danger of fire; it is not efficient in shallow mines, and cannot be used with safety in mines generating firedamp. In mines not generating gas in dangerous quantities, should a dust explosion occur, it might be difficult or impossible to reach the furnace and restore circulation in the mine; while a fan located on the surface is always within reach, and if damaged by an explosion can be repaired. The furnace represents, however, a much smaller investment than a fan. The steam jet is only useful in sinking shafts. The wind cowl or any form of natural ventilation is wholly unreliable.

Ques. 9.—How would you determine the size of pillars and widths of rooms and entries necessary to properly work out the coal, prevent squeezes, and keep the mine in good working condition?

Ans.—This question is fully answered in reply to Ques. 5, page 318, Mines and Minerals, December, 1909. See also Ques. 2, page 574, April, 1910.

Ques. 10.—Make a sketch showing how you would draw back pillars from a section of a mine that has been driven to the

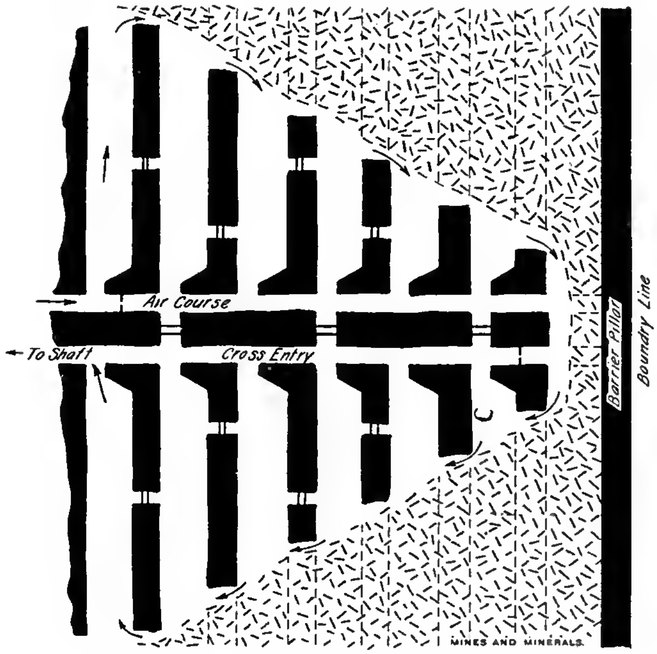


Fig. 1

boundary, in order to obtain the largest percentage of coal with the greatest degree of safety to the miners, in respect to falls of roof or accumulation of gas.

Ans.—When the entries have reached the edge of the barrier pillar to be left at the boundary, and the rooms have all

been driven up to their limit, work is started on the extreme end of the inside room pillars and proceeds in order outby, the work on each pillar following in succession so as to preserve a regular line of pillar work, as shown in Fig. 1. As the room pillars are finished the entry pillars and stumps are drawn. Care must be taken to keep the work in line and leave no timber standing.

Ques. 11.—What precautions are necessary to prevent squeezes in coal mines, and what conditions govern the mode of procedure? What dangers attend a squeeze and how may these dangers be minimized?

Ans.—To avoid squeeze in coal mining it is necessary to lay out and prosecute the work in a systematic manner, according to a definite plan in which the width of entries and rooms, and the widths of the main entry, cross-entry, and room pillars are all made to correspond to the depth of the seam below the surface, the character of the roof, floor, and coal, thickness and inclination of the seam, and the method adopted for the extraction of the coal. All of these conditions govern the manner of conducting the work. It is important also to avoid a large standing area. Pillars should generally be drawn as quickly as the adjoining rooms reach the limit. As pillars are drawn back care must be taken to secure a steady roof settlement by withdrawing all timber. At times it is necessary, in order to relieve the entry pillars of undue roof pressure, to break the roof by a shot placed in the mouth of the room. The effect of fault lines must also be carefully observed; and, on account of such fault lines, it may become necessary to change the direction of driving, so as to cross the fault on an angle. The dangers arising from squeezes are the crushing and loss of pillar coal, and often the closing off of large areas of valuable coal that must then be reached from other directions or abandoned wholly; sudden and extensive falls of roof, and increased danger of mining coal in adjoining workings. To reduce these effects as far as possible, every effort must be made to arrest the squeeze by rapid drawing of pillars adjoining the affected portions; and withdrawing all standing timber and causing roof falls in nearby abandoned places. Where pillars are weak in entries and workings desired to be held, timber cogs should be built and packed solid with waste. Often a squeeze can be diverted and broken entirely by causing heavy roof falls behind or at one side of the trouble.

Ques. 12.—Show by a sketch and explanation how you would stand props or timbers in a mine having a draw slate 16 inches thick overlying a seam 4½ feet high, with a bottom of 12 inches of soft fireclay.

Ans.—If practicable, this coal should be worked by driving up narrow stalls (Fig. 2), say 12 or 14 feet wide and supporting

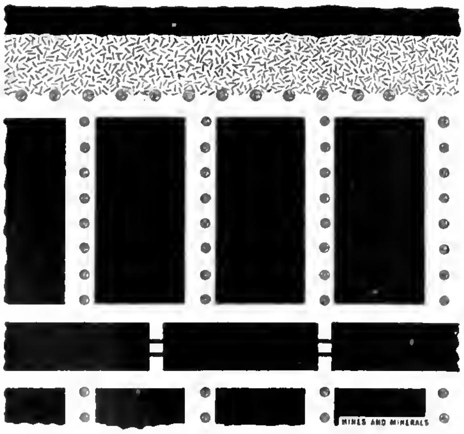


Fig. 2

the draw slate by a line of posts at the side of the road in each stall. The pillars separating the stalls may be any suitable width from 10 to 20 yards, depending on conditions relative to depth of cover, strength of coal, slate, and roof strata above slate. One or more lines of posts should be kept behind the line of pillar work to support the draw slate over the miners. These posts are removed as the pillars are drawn back and the slate allowed to fall. If the fireclay is very soft it may be necessary to set the posts on foot-boards or sills.

Ques. 13.—Explain the principle of action of a pump. What is the height that a pump can lift water and do effective work? Describe the various ways by which coal mines are drained and compare their efficiency and economy.

Ans.—The movement or stroke of the piston or plunger of a pump creates a more or less perfect vacuum in the suction pipe, and the atmospheric pressure acting on the surface of the water in the sump presses it up through the suction pipe into the pump barrel. On the return stroke of the piston, the suction valve being now closed and the discharge valve opened, the water is forced out of the pump barrel into the discharge pipe. This action, in a double-acting pump, is repeated each stroke of the piston, alternately, in each end of the cylinder or barrel. The height a pump will draw water is dependent on the pressure of the atmosphere; it may be taken, in feet, ordinarily, as from .8 to .9 of the height of the barometer reading in inches. For example, if the barometer reads 30 inches, the suction lift of a pump, at that place, may be safely taken as from 24 to 27 feet, depending on the condition of the pump. The means in most common use for unwatering or draining coal mines are pumps, siphons, water buckets, and various forms of displacement pumps actuated by steam or compressed air; as for example the pulsometer, air lift, pneumatic, and steam jet, or injectors. Improved forms of steam, electrical, air, or belt-driven pumps of the piston, plunger, or centrifugal type are in common use and are the most generally efficient and economical for unwatering shafts. Water buckets of special design have been used in many cases owing to their simplicity and effectiveness. Siphons are applicable only in special cases for draining water by a pipe line over a small rise or elevation. They are simple and very economical when properly installed. The pulsometer is a simple and effective means of unwatering a shaft, especially when sinking. The air lift can only be used when it is possible to submerge the pipe to a depth greater than the point of discharge above the surface of the water in the shaft.

Ques. 14.—State how you would lay a good haulage road in a main entry where 12-ton motors are used; the loaded cars weighing 3½ tons, and the track having a 1-per-cent. grade in favor of the loads. Give weight of rails per yard, size of spikes, number and size of ties used, and estimate the cost of 5,000 feet of the track, assuming that the rails cost $26 per ton, ties 10 cents each, spikes $3.75 per keg of 200 pounds, labor for trackmen $2.50 per day, and helpers $1.75 per day.

Ans.—The rails for a 12-ton motor haulage should not be lighter than 40 pounds per yard, which would require $(2\times 5{,}000\times 40)\div(3\times 2{,}240)=59.5$ tons at a cost of $26\times 59.5=\$1{,}547$. For 40-pound rails, use $3\frac{1}{2}'' \times \frac{7}{16}''$ spikes, 12 kegs, at $3.75 per keg = $45; and 4″ × 6″ cross-ties, spaced 2 feet, center to center, 2,500 at 10 cents each = $250. There will be required also, using 24-foot rails, 832 angle or fish-plates, 6,240 pounds, at 1½ cents per pound = $93.60, and 8 kegs bolts, nuts, and washers at $5 per keg = $40; making the total track material $1,975.60. The laying and surfacing of 5,000 feet of track in mine entries, under ordinary conditions, including the handling of the material in the shaft and its distribution in the entry will require, approximately 120 days labor for helpers at $1.75 per day = $210; 50 days, trackmen, at $2.50 per day = $125; and 6 days, drivers, at $2 per day = $12; total for labor $347. The total cost of the track laid is therefore $2,322.60, making no allowance for special grading which might be required at some points in the entry.

The volumetric capacity of a fan is the ratio of the actual volume of air produced to the cubical contents of the fan multiplied by the number of revolutions.

AN ANTHRACITE CRYPTOGRAM

Written for Mines and Minerals

The diamond, which is pure carbon, crystallizes in the isometric system which is similar to the anthracite shown in crystal form in Fig. 1. Graphite, another well-known kind of carbon, crystallizes in the hexagonal system, and consequently, has a different form from that shown in Fig. 1.

Minerals that have definite chemical compositions crystallize from solutions, and since coal is not a definite chemical compound, it has never before to the writer's knowledge been found in crystal form.

All the minerals termed carbonates have a definite chemical composition and contain carbon as a necessary constituent, nevertheless coal varies in chemical composition in different beds and even in the same bed and this adds further interest to the natural cryptogram in Fig. 1; from these facts the following deductions are made:

(*a*) Coal is not a mineral because of its not being a definite chemical compound. (*b*) Coal does not crystallize, because it has not been taken in solution as a chemical compound.

(*c*) Coal before hardening was plastic and was folded and contorted when in that condition; and it is probable that in the anthracite field it was made plastic and afterwards metamorphosed by dynamic earth movements.

(*d*) The anthracite shown in crystal form is an impression made by pyrite in soft plastic material that was afterwards hardened by metamorphism.

Fig. 1. An Anthracite Cryptogram

(*e*) While the anthracite has the isometric form of pyrite, as shown in Fig. 1, it is not a pseudomorph form, for the reason that there has been no interchange of materials; in other words, the pyrite was imbedded in the coal which was in a plastic state.

(*f*) Metamorphosed carbon grades from the free burning variety of Northeastern Pennsylvania anthracite, through the difficultly burning graphitic anthracite of Rhode Island, to graphite.

WEST VIRGINIA MINING ASSOCIATION

The West Virginia Mining Association held its annual meeting at the Willard Hotel, Washington, D. C., December 16, and more than 100 members were in attendance. President W. N. Page presided over the deliberations, and in his annual address, made the statement that, at that time not one life had been lost in West Virginia by a mine explosion for 23 months.

G. H. Caperton, treasurer of the association, read his annual report, which showed that there was on hand a cash balance of $401.44. During the life of the association, 3 years, the receipts have been $13,661, while the expenditures have amounted to $13,259.56.

It was recommended that an additional district be added in the association.

W. D. Ord, M. T. Davis, and Earl Smith, were named as the committee on resolutions.

E. W. Parker, G. O. Smith, and M. R. Campbell, of the Geological Survey, made short addresses, as did also David White, of Charlestown, and Dr. I. C. White, of Morgantown. J. W. Herron, of the C. & O. Railroad, spoke on "Car Allotment," and John Laing, Neill Robinson, and Frank Haas, read interesting papers. Thomas Nelson Page, author of "Santa Claus' Partner," "Two Little Confederates," etc., gave an interesting 10-minute talk. The meeting was both successful and profitable.

COAL-DUST EXPLOSION IN A CEMENT MILL

A most unusual coal-dust explosion by which 8 men were so badly burned that they lived but a short time, while another is not expected to recover, happened at the cement mill of the Portland Cement Co., at Portland, Colo., on January 3. Before the coal is used in burning the cement, it is very finely pulverized, a large quantity of the finely divided dust naturally escaping from the grinders and settling over the walls and floors as in a coal mine. This dust is cleaned up every 2 weeks. While engaged in this work on the day mentioned the dust was either drawn into the boilers and exploded with great violence, or was ignited by coming in contact with a sparking electric motor in the kiln room. The former explanation appears the more reasonable, allying this accident with the explosion in the Washburn Mills, in Minneapolis, Minn., in 1873.

UNITED STATES COAL PRODUCTION FOR 1910

It is estimated by E. W. Parker that between 475 and 485 million tons of coal were mined in 1910.

The following approximate number of short tons were produced by the coal-mining states:

Pennsylvania anthracite	72,468,464*
Pennsylvania bituminous	136,100,000‡
West Virginia	60,000,000*
Ohio	31,500,000*
Alabama	15,000,000*
Tennessee	6,234,922†
Kentucky	9,550,640*
Michigan	2,000,000‡
Colorado	12,089,447†
New Mexico	3,684,627†
Utah	2,526,093†
Montana	3,053,940*
Washington	4,500,000*
Alaska, California, and Oregon	100,000*
Arkansas	2,000,000‡
Wyoming	6,750,000

*E. W. Parker. †State Mine Inspector. ‡Estimated. The coal-mining industry, in Illinois, Kansas, Arkansas, Oklahoma, and Missouri was demoralized by strikes.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

WESTERN EDITORIAL OFFICE: 728 Equitable Building, Denver, Colo.
CHICAGO OFFICE: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to MINES AND MINERALS, Scranton, Pa.

CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER........MANAGER
EUGENE B. WILSON, SCRANTON, PA........EDITOR
GEORGE F. DUCK, E. M., DENVER, COLO........WESTERN EDITOR
P. G. MOORE........CIRCULATION MANAGER
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., LTD., DRYDEN HOUSE, 43 GERRARD ST., SOHO, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 PITT STREET, SIDNEY,
SOUTH AFRICAN AGENT: E. HERBERT BYSSHE, CAPE TOWN, MANSION HOUSE; JOHANNESBURG, TRUST BUILDING; DURBAN, NATAL BANK CHAMBERS; BLOEMFONTEIN, FLORENCE BUILDINGS.

SUBSCRIPTION RATES

Single Copies........25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada........2.50
To all other countries........3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always gives your FORMER as well as your present address.

TABLE OF CONTENTS

CONSOLIDATED FUEL CO., UTAH, by R. J. Turner........385
STEEL SUPPORTS IN COAL MINES, by R. B. Woodworth........387
PERSONALS........390
THE MORAL OF STARKVILLE, by Sim Reynolds........391
OUTBURSTS OF CARBON DIOXIDE IN FRENCH MINES........392
PRINCIPLES OF ELECTRIC BLASTING, by W. G. Hudson........393
COMMERCIAL CLASSIFICATION OF FUELS, by R. E. Rightmire........397
THE BELLEVUE EXPLOSION, ALBERTA, by James Ashworth........399
CATALOGS RECEIVED........401
THREE-WIRE MINE SERVICE, by J. M. Hunt........402
COAL MINING NOTES........404
EQUALIZATION OF FUELS, by H. K. Myers........405
TRADE NOTICES........407
COAL MINING INSTITUTE OF AMERICA........407
COAL MINE TRANSPORTATION, by E. B. W........408
COMPENSATION FOR INJURY, by Raymond P. Tarr........410
MINE CAR CAGING MACHINE........413
EXAMINATION QUESTIONS—
Mine Foremen's Examination, Fairmont, W. Va., November, 1910........414
AN ANTHRACITE CRYPTOGRAM........417
COAL DUST EXPLOSION IN CEMENT MILL........417
UNITED STATES COAL PRODUCTION, 1910........417
EDITORIAL—
Colorado Coal Mine Commission—Mining Engineer as Citizens—Government Oil Land Suit........418
BOOK REVIEW........420
RESTORING DREDGED GROUND, by A. S. Atkinson........422
LOG WASHER IN ZINC MINING, by L. L. Wittich........423
FAKE OIL COMPANIES, by Louis E. Aubury........424
CORRESPONDENCE—
Haulage........425
LEYDEN, COLO., MINE FIRE........426
MEXICAN TERRITORY OF QUINTANA ROO, by Wm. R. Canada........427
EDUCATIONAL........428
DIAMONDS IN GERMAN SOUTH AFRICA........429
THE SIPHON IN MINING, by J. T. Beard........427
MINING IN SPAIN, by Robert J. Frazer, Jr........429
HEAT OF COMBUSTION OF EXPLOSIVES........429
TESTS OF COPPER MINERALS, by Evans W. Buskett........430
ZINC ORE NOTES........431
THE LARGEST ORE CRUSHER........432
THE CLANCY CYANIDE PROCESS........433
NORTHWESTERN ORE NOTES........434
JOPLIN ZINC AND LEAD PRODUCTION, 1910........435
LEADVILLE, COLO., ZINC DEPOSITS........436
DEL CARMEN ZINC MINE........437
BROKEN HILL, N. S. W., MINERALS........439
LAKE SUPERIOR IRON ORES........440
BONANZA COPPER MINE, ALASKA, by U. H. Wilhelm........441
PETROLEUM MINING ENGINEERING, by Raymond S. Blatchley........442
EVOLUTION IN HOISTING, by E. B. W........444
SUBMARINE DIVERS IN MINES, by Geo. F. Duck........446
RUSSIAN FAR EASTERN GOLD FIELD........447
INDUCTION MOTOR RUNS UNDER WATER........448
NEW INVENTIONS........448
SQUIBS—
Department of Agriculture Misquoted, 396—Value of Alaska's Coal Lands, 401—Duty on Precious Stones, 404—Oil Operation in Tampico, 406—Transportation in Alaska, 409—European Laws Regarding Breathing Apparatus, 413—Volumetric Capacity of a Fan, 416—West Virginia Mining Association, 417—Common Casting Copper, 421—Utilizing Mine Refuse, 421—Coal Consumption of the World, 426—Wet Assay of Gold, 425—Wolfram, 432—Fluorspar, 435—Prospecting Advice, 435—Breathing Appliances in Belgium, 440—Oil in Venezuela, 441—Platinum Diarsenide, 443—Mining in the Malay States........448

COLORADO COAL-MINE COMMISSION

GOVERNOR SHAFROTH, of Colorado, has selected as commissioners to inquire into the recent coal-mine disasters of that state, and to suggest remedies to be incorporated in the laws, Dr. Victor C. Alderson, President of the Colorado School of Mines; John B. Ekeley, Professor of Chemistry at the University of Colorado; Mr. R. D. George, State Geologist; Mr. James Dalrymple, Chief Inspector of Coal Mines.

Without reflecting on the personality of the members of the commission, we are inclined to think that Governor Shafroth would have secured very much better results with a somewhat different commission, or even with the same commission augmented by an equal number of practical mining men. By this we mean, men who are, or have been, engaged in the production of coal, either as mining engineers of high repute or as mine managers, and there should be at least one or two intelligent, practical miners on the commission. Such a commission would, naturally, be rather large, but this is about the only thing the Governor can do, after appointing his regular commission. If, in making the original appointments, the Governor had nominated Doctor Alderson, for instance, as chairman, a mining engineer of education and broad practical experience in coal mining, the chief mine inspector of coal mines and a broad-gauged intelligent miner, the commission for a small one would have been ideal, because a mining engineer would not only be competent to pass on questions of mining methods, but he would be fully as familiar in a practical way with the questions bearing on chemistry and geology as either of the two gentlemen the Governor did nominate. It is to be hoped that even if Governor Shafroth does not himself see the advantage of following the suggestion contained in this editorial, that the gentlemen comprising the present commission will recommend that practical mining men, such as we have referred to, be given representation on the commission.

The preliminary report of this commission has been published in the daily papers of Denver, and we were somewhat inclined to criticise it, but as the report was merely a preliminary one and not definitely codified in the shape of a proposed law, we refrain from such criticism, and will reserve our judgment until the proposed law is codified. At that time we expect to take it up, section by section, commend all of it that we conscientiously can, and, of course, criticise all such portions of it as may be deemed impractical or inimical to the best interests of the mining industry. We include in the statement "the best interests of the mining industry," the interests of every one engaged in coal mining from the humblest trapper to the stockholder.

One statement in the preliminary report of the commission to which we do take exception is that, "a coal mine is, with the possible exception of a powder mill, the most dangerous of all places in which to work."

If the gentlemen of the commission will take the trouble to go over the reports of the mine inspectors of the various coal-mining states and get out the number of fatalities and injuries per thousand men actually engaged in mining and compare it with similar reports pertaining to railroads, they will find that the ratio of death per thousand is from two to three times as great and that the ratio of injuries from four to five times as great in railroading as in coal mining. Furthermore, except in abnormal years, the commission will find that less than 10 per cent. of the deaths due to fatal accidents in mines were caused by gas or dust explosions, and that nearly 60 per cent. were due to falls of roof and coal, with about 15 per cent. due to accidents from cars. They will also find, particularly in the matter of accidents from falls of roof and coal, and accidents from cars, that over 75 per cent. of such accidents were due to absolute carelessness on the part of the victims. All men practically engaged in mining know that these statements are approximately true. Men who are engaged in educational work pertaining to mining, no matter how eminent they may be, are not as familiar with the actual details of mining as they should be to enable them, without the advice of a practical miner, to codify a set of laws which would tend to the production of coal with the utmost safety to the employe, and the property of the employer, without making the cost of production prohibitive.

MINING ENGINEERS AS CITIZENS

WHY IS the mining engineer so indifferent to his duties and privileges as a citizen, is a question asked frequently and answered in many ways. Mining engineers as a rule are not indifferent citizens, even though nomadic and living on the fringe of civilization, they mold public opinion to a larger extent in their particular bailiwick than is supposed. They do not touch politics for office, but use their influence for the good of the country generally and their own business particularly, without public notice. Their work demands all their attention, which prevents their attaining prominence in the business world, in politics, or in letters, and being ethical in their ideas of publicity they are seldom heard of outside their own profession. When mining engineers meet in convention they talk mining; in part of their spare moments they talk, read, or think mining; but the remainder of their lives is not shadowy or unsubstantial, for the mining engineer has an economical end to his business second to none, either mercantile or banking. His mental development is not that of a specialist on some particular branch, as some seem to think. If this were so his vision would be narrowed and strabismic, and he would fall into a groove where he might become an expert on a mining specialty, but never a mining engineer. The mental development of the specialist is not that of the mining engineer, for the latter must be informed on civil and constructive engineering, and, in addition, must be a fair chemist, mineralogist, geologist, economist, and also be endowed with executive ability. Almost any intelligent man can run a mine after it has been put in condition and systematized by a mining engineer, provided he has the mining engineer to consult with occasionally; but not every one can put a mine in condition, establish an economical system of mining, and the proper method of keeping cost sheets.

Probably no class of men work harder and to better advantage for their country than mining engineers; therefore, if their names do not appear in print as frequently as lawyers, they are far from being negligible as public citizens.

GOVERNMENT OIL LAND SUIT

ON DECEMBER 10 a suit was filed in the United States District Court, at Los Angeles, Cal., by United States District Attorney A. A. McCormick in the name of Attorney General Geo. W. Wickersham, against the Southern Pacific Co.; Southern Pacific Railroad Co., Homer S. King, trustee; James K. Wilson, trustee; Central Trust Co. of New York; Equitable Trust Co., of New York, and Kern Trading and Oil Co., to recover title to 6,109 acres of land in the Midway district, now claimed to be unlawfully held by the defendants.

This is the long-expected suit about which so much was heard at the Los Angeles convention of the American Mining Congress in the latter part of September last. It is based upon the construction to be placed upon the act of Congress of July 27, 1866, and a subsequent resolution dated June 28, 1870. Under the terms of the above, the railroad company, in order to encourage its construction, was allowed to patent every alternate odd-numbered section on each side of its right of way, but in this original grant mineral lands, either known to be such at the time or subsequently proved to be mineral bearing, were specifically excluded, and it is this clause which is said to render the patents invalid. It is further claimed that maps on file in the Surveyor-General's office, and dated 1902, show that at that time these lands were classified as mineral bearing, while the formal patents were not granted to the Southern Pacific or its trustees until 1904.

Not only are misrepresentation, fraud, and deceit charged, but the bill recites that the Government's own officials were negligent in granting patents to lands to the railroad company without any investigation whatsoever, relying entirely upon the statements of the defendant corporation as to the character of the lands.

The Kern Trading and Oil Co. is a subsidiary of the Southern Pacific Railroad, and in the bill of complaint is stated to have been organized as a dummy for the purpose of furthering the railroad's alleged "fraudulent, dishonest, and unlawful purpose of witholding this land from its rightful owner," the United States.

The outcome of this suit will naturally be awaited with much interest, as upon its successful issue will depend other suits involving title to many hundreds of thousands of acres of land now said to be unlawfully in the hands of corporations and individuals. In the case at issue the lands, being oil bearing, are said to be worth $10,000,000, and other lands now in possession of the railroad and aggregating $50,000,000 are also at stake. An interesting point is the bearing the alleged neglect of the Government's own officials will have upon the case. Is the Government responsible for their negligence or not?

Justly or unjustly, the people of Southern California seem to feel that this suit will never be brought to conclusion, pointing to the affiliations of the Attorney General and the Secretary of the Interior, the strong "pull" of the Southern Pacific, and the well-known flaccidity and lack of back bone on the part of the administration. Perhaps Washington has waked up at last, and after the rebuke of November 8 may now be willing to carry out the plainly expressed wishes of the people. Let us hope so.

Book Review

A Gold Mine in Tennessee Clays. Tennessee Geological Survey, Press Bulletin No. 8, G. H. Ashley, State Geologist, Nashville, Tenn.

American Red-Cross Abridged Textbook on First Aid. This pocket edition is prepared for general readers, students, boy scouts, schools, etc. There has already been published an "industrial edition" adapted to the special needs of miners, railway employes, workers in iron and large manufacturing establishments. This edition will be issued also in Italian, Slovak, Polish, and Lithuanian. This manual of instruction is by Major Charles Lynch, Medical Corps of the United States Army; 183 pages, 55 illustrations. P. Blakiston's Son & Co., Philadelphia, Pa., publishers, price 30 cents.

Gurley's Manual, 45th edition, 516 pages with index. This book is illustrative of American engineers' and surveyors' instruments. It furnishes instruction in the adjustment and use of field instruments in as simple a manner as is consistent with the subject. No attempt is made to be technical or deal with surveying as a study. All instruments used in surveying and mapping are illustrated. Gurley's Manual is in nearly every engineer's office and can be had if not, for 50 cents, from W. and L. E. Gurley, Troy, N. Y., or Seattle, Wash.

Rock Drills, by Eustace M. Weston, Associate School of Mines, Ballarat; Reef Lecturer on Mining, Transvaal University College; 367 8mo pages with index; 193 illustrations; 16 chapters as follows: Historical; Standard Piston Drills; Hammer Drills; Electric Drills; Operating Rock Drills; Piston Drills Designed to Use Air Expansively; Philosophy of the Process of Drilling Rock; Repair and Maintenance of Rock Drills; Drill Steel and Drill Bits; Explosives and Their Use; Theory of Blasting With High Explosives; Examples of Rock Drill Practice, Africa and Australia; of America; Rock Drill Tests and Contests; Notes on the Use of Compressed Air. Mr. Weston has covered the ground thoroughly with the exception of tunnel driving, but for this neglect he makes up in other ways. In the chapter on the Repair and Maintenance of Rock Drills there is an important element lacking; namely, the care of rock drills. A large part of the cost of repairs in rock drills comes from the maltreatment they receive in handling, setting up, pointing, and during running, and not from natural wear alone. The book is published by McGraw-Hill Book Co., New York City, and sold at $4 net.

Practical Stamp Milling and Amalgamation, by H. W. MacFarren, 166 pages and index; no illustrations; postpaid $2. Publishers, Mining and Scientific Press, San Francisco, Cal.; Mining Magazine, 819 Salisbury House, London. There are other books on stamp milling and volumes written about this simplest of crushing machines. "No machine can compare with it in amalgamating or in preparing an ore for concentration, except where the friable nature of the material requires stage crushing." "So thoroughly can it be depended upon that it has passed into an axiom that a standard stamp mill has never caused the closing of a property by failing to do good work where work was possible, and that it has replaced every other kind of crushing machine." "The height of drop of the stamps in a battery, and the other adjustments, are made for each ore according to its hardness and fineness." "The subject of battery foundations is important, and the seal of approval has been placed on concrete mortar blocks; but while realizing that a good concrete block may be perfect, it must be admitted that in a large number of cases they have been unsatisfactory from defective construction, while the wooden block has given satisfaction in practically every instance." "The concrete block gives a cleaner looking mill, and does not decay." "By its solidity and non-cushioning effect, it increases the capacity of a battery sometimes as much as 33⅓ per cent." "Mortars are made of cast iron and approximately 6 or more times heavier than the weight of the stamp to be used in them." "It is generally cheaper to use steel shoes and dies than iron ones despite its increased cost, on account of the smaller consumption of steel per ton of ore crushed, and the less time lost in renewals and setting of the tappets." "The life of steel shoes and dies may roughly be estimated at 4 months or less." "No die should be permitted to stand higher or lower than another, or it will cause its stamp to 'pound' or to 'cushion' through having too thin or too thick a bed of pulp over it." "In putting on a new set of shoes, the stems are first cleaned of the grease below the tappets by passing a long strip of burlap or cloth wet with kerosene or gasoline about the stem and alternately pulling each end of the cloth until the stem is clean." "This is necessary as it would be impossible to make two slippery greasy metal surfaces hold together."

The reader will understand from the above quotations taken at random from the book that the author is practical; that his statements are facts and that these are followed up with the reasons why. The book is the product of experience enriched by observation; and written without adjectives. Did the reader ever try to fill a fountain pen that had accumulated a lot of bubbles; if so, he found it disparaging. Bubbles in a fountain pen barrel are like adjectives in a book. Mr. MacFarren has written a most valuable book, devoid of theoretical discussions and academic distinctions but replete with the details of practical stamp milling. It might be truthfully said that it was the ink and not the bubbles that made the author's fountain pen write.

Books to be Reviewed. Elementary Crystallography, being Part 1 of General Mineralogy, by W. S. Bayley, Ph. D. McGraw-Hill Book Co., New York City.

More Recent Cyanide Practice, edited by H. Foster Bain; Mining and Scientific Press, San Francisco; price $2. This book supplements "Recent Cyanide Practice," published by the same firm.

Manual for Assayers and Chemists, by W. H. Seamon. Published by John Wiley & Sons, New York. Price $2.50.

Books Received

United States Geological Survey Publications, Washington, D. C. Thirty-First Annual Report of the Director of the United States Geological Survey to the Secretary of the Interior for the Fiscal Year Ended June 30, 1910. Bulletin 430,

Contributions to Economic Geology, Part 1, Metals and Nonmetals Except Fuels, by C. W. Hayes and Waldemar Lindgren; Bulletin 431-A, Advance Chapter from Contributions to Economic Geology (Short Papers and Preliminary Reports) for 1909, Part 2, Mineral Fuels, Petroleum and Natural Gas, by A. G. Leonard, H. E. Gregory, C. W. Washburne, and Robert Anderson; Bulletin 433, Geology and Mineral Resources of the Solomon and Casadepaga Quadrangles, Seward Peninsula, Alaska, by Philip S. Smith; Bulletin 434, Results of Spirit Leveling in Delaware, District of Columbia, Maryland, and Virginia, 1896 to 1909, inclusive, by R. B. Marshall, chief geographer; Bulletin 437, Results of Spirit Leveling in Maine, New Hampshire, and Vermont, 1896 to 1909, inclusive, by R. B. Marshall; Bulletin 440, Results of Triangulation and Primary Traverse for the years 1906, 1907, and 1908, by R. B. Marshall; Bulletin 442, Mineral Resources of Alaska, Report on Progress of Investigation in 1909, by Alfred H. Brooks and others; Bulletin 444, Bibliography of North American Geology, by John M. Nickles; the Production of Talc and Soapstone in 1909, by J. S. Diller; the Production of Asbestos in 1909, by J. S. Diller; the Production of Monazite and Zircon in 1909, by Douglas B. Sterrett. Water-Supply Paper 254, The Underground Waters of North-Central Indiana, by Stephen R. Capps, with a Chapter on the Chemical Character of the Waters, by R. B. Dole; Water-Supply Paper 262, Surface-Water Supply of the United States for 1909, by M. R. Hall and R. H. Bolster; Water-Supply Paper 264, Surface-Water Supply of the United States in 1909, Part IV, St. Lawrence River Basin, by C. C. Covert, A. H. Horton, and R. H. Bolster.

Nineteenth Annual Report of the Canada Bureau of Mines, 1910, Vol. XIX, Part 1, by F. Cochrane, Minister of Lands, Forests, and Mines, Toronto, Canada.

Canada Department of Mines, Mines Branch, Ottawa, Canada. The Production of Coal and Coke in Canada in 1909, by John McLeish, B. A., Chief of the Division of Mineral Resources and Statistics; the Production of Iron and Steel in Canada, by John McLeish, B. A.

British Columbia Bureau of Mines, Victoria, B. C., Bulletin No. 2, 1910, on the Portland Canal Mining Division, by Wm. Fleet Robertson, Provincial Mineralogist.

Thirty-Fifth Annual Report of the Chief Inspector of Mines of Ohio, George Harrison, Wellston, Ohio.

Report of the Department of Mines for 1909, Part 1, Anthracite, Part 2, Bituminous, James E. Roderick, Chief of the Department of Mines, Harrisburg, Pa.

Report of the Mining Industry of Natal for the Year 1909, by C. J. Gray, Commissioner of Mines, Pietermaritzburg, South Africa, price 2s. 6d.

Record of First Series (1908–09) of the British Coal-Dust Experiments, issued by the Mining Association of Great Britain. Printed and published by the Colliery Guardian Co., 30 Furnival St., Holborn, London, E. C.; price 10s. net.

Papers and Reports Relating to Minerals and Mining, comprising a statement by the Minister of Mines, Report on the Goldfields, Report on State Coal Mines, by Hon. Roderick McKenzie, Minister of Mines, Wellington, New Zealand.

Biennial Report of the Board of Geological Survey of the State of Washington for the Term 1909–11, J. G. Lewis, Secretary of the Board, Olympia, Wash.

Washington Geological Survey, Bulletin No. 1, Geology and Ore Deposits of Republic Mining District, by Joseph B. Umpleby. Address Henry Landes, University Station, Seattle, Wash.

The Irrigated Lands of the State of Washington, by George M. Allen. Address Bureau of Statistics and Immigration, I. M. Howell, Secretary, Olympia, Wash.

Crop Statistics, Bulletin 21, Bureau of Labor Industrial Statistics, Elizabeth Heeps, Chief Clerk, Omaha, Neb.

University of Illinois Bulletin No. 42, The Effect of Keyways on the Strength of Shafts, by Herbert F. Moore, Urbana, Ill.

Biennial Report of the Colorado State School of Mines, Golden, Colo., James T. Smith, Secretary.

The Tennessee White Paint Tests, Bulletin No. 30, by Henry A. Gardner, Director of the Paint Manufacturers' Association of the United States, Philadelphia, Pa.

Abstracts of the Reports of the French Coal Dust Experiments conducted at the Lievin Experimental Station 1907–10, by J. Taffanel. Published by the Colliery Guardian Co., Ltd., 31 Furnival St., Holborn, E. C., London.

New Zealand Geological Survey, Bulletin No. 11 (New Series), The Geology of the Mount Radiant Subdivision, by Ernest John Herbert Webb. Address J. M. Bell, Director of Department of Mines, Wellington, N. Z.

The Logical Point, an Illustrated Monthly for the advancement of the World's Panama Exposition, New Orleans, La.

Bulletins of the Massachusetts Institute of Technology, Volume 44, No. 1, and Volume 24, No. 3, Boston, Mass.

Mineral Resources of the Philippine Islands, Warren D. Smith, Chief of the Bureau of Science, Manila, P. I.

Report of Tests of a Babcock & Wilcox Boiler for United States Battleships, Wyoming and Arkansas, with notes by Lieutenant-Commander H. C. Druger, United States Navy. This is a reprint from the Journal of the American Society of Naval Engineers, November, 1910. Can be had at the Babcock & Wilcox Co., New York City.

State of Illinois Bureau of Labor Statistics, Report on the Cherry Mine Disaster, David Ross, Secretary, Springfield, Ill.

Fifteenth Biennial Report of the State Mine Inspectors of Iowa, for the two years ending June 30, 1910, Edward Sweeney, State Mine Inspector, Des Moines, Iowa.

Annual Report of the Secretary of Internal Affairs, Part 3, John L. Rockey, Chief of the Bureau, Harrisburg, Pa.

Department of Commerce and Labor, Washington, D. C., Bulletin of the Bureau of Labor, No. 89, Child Labor Legislation in Europe.

Pennsylvania State College Bulletin, Vol. 4, No. 6, State College, Pa.

Geological Notes, by G. Henriksen, Inspector of Mines, Bergen, Norway.

COMMON CASTING COPPER

A very useful alloy for soldering irons, copper hammers, and all copper castings, where high conductivity is not essential, is given by the *Foundry*, as copper 96 pounds, zinc 4 pounds. This formula will always be found satisfactory, as the zinc is sufficiently high to ensure solidity, whether ingot or scrap copper is used. The color of the metal is sufficiently red to pass for copper, and the castings can be relied upon to show up clean, bright, sound, and free from abnormal shrinkage. The copper should be melted in a clean crucible, and, when first charged, two tablespoonfuls of salt are added. The chlorine gas from this protects the copper from excessive oxidation until it settles down in a liquid state, when it should be covered with charcoal, hardwood chips, tan bark, or any organic substance that forms charcoal. The zinc should be added after the copper is melted. The mixture is then thoroughly stirred and the metal is allowed to superheat a few minutes before being cast. If the color of the metal is not sufficiently red, the zinc with careful melting, may be reduced to 2 per cent.

It is believed that the refuse from the mines is capable of yielding profit in the way of gas and briquets. It is argued that if coal men gave the same attention to the problem of by-products as do manufacturers, the rewards would go a long way toward meeting their fixed expenses, and that the labor problem would diminish in importance.—*Fuel.*

RESTORING DREDGED GROUND

Written for Mines and Minerals, by A. S. Atkinson

Australian Gold Dredging Method Which Leaves the Ground in Condition for Agriculture

In view of the constantly increasing use of the modern big gold dredges, and the constantly recurring question of the damage they do to the land which might otherwise be used for agricultural purposes, the efforts being made in Australia to restore dredged ground to a condition which permits of agricultural use should be of more than general interest. The big bucket dredges have come to stay, and their operation in many parts of the world is the most economical of any method yet invented for recovering the precious metal. Even in Alaska and the Klondike, where the cost of getting the dredges there and set up is enormous, the work of the gold dredges has advanced rapidly. It costs something like $120,000 to get one of these dredges in Nome ready for operation, and yet in spite of this first heavy initial cost they are proving extremely profitable.

In California the gold dredges have been in operation so long that their operation has caused more or less friction between the mining companies and the agriculturists. The same is true in parts of Australia. It may be said that in some respects the question is more acute there than in our own country. In the state of Victoria, Australia, the gold dredge has been in operation on such a stupendous scale that it has threatened large areas of agricultural land, and the conflict of interests began early and reached an acute stage some time ago.

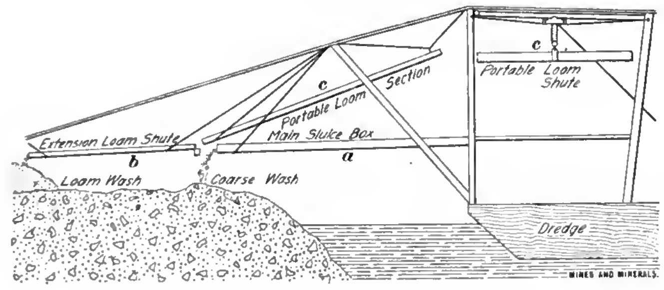


Fig. 1. Handling Loam and Gravel

From official figures, 2 years ago there were 133 dredging plants in operation in the state of Victoria, but only 45 of these were of the bucket type. The rest were pump hydraulic sluices, or the four-jet elevators. The bucket type of dredges are mammoth affairs, three of them capable of treating more than 9,000 cubic yards of soil a week, and four of 8,000 yards capacity. Altogether these 133 dredging plants treated in 1907 over 20,000,000 cubic yards of soil, and obtained nearly 98,000 ounces of gold. About 2,500 men were employed in operating the plants.

Naturally, this wholesale dredging turned up considerable soil,and the fear that a good deal of valuable agricultural and grazing land would thereby be permanently injured led to the adoption of plans for restoring the land thus dredged, so it could be utilized for farming purposes. The development of this work has been a feature of the past year's dredging and a good deal of progress has been made. It is not claimed that entire success has been reached, or that the land is just as good after the dredging as before, but it is quite evident that it is not rendered worthless by the miners.

The method of restoring the soil consists in nearly every instance of some system of classifying and separating the dredged material. By depositing the fine sand on top the dredged area is partially restored to its former productiveness.

In suitable ground where the material is of a loose, friable, sandy nature, the tendency in bucket dredging has been to discard screens and elevators. In their place riffle-bottomed sluice boxes are used. Sand launders are used so that the silt and coarse wash are automatically separated, and with the proper distribution of the material the soil is not entirely destroyed. The sand launders are laid with cocoa matting placed under sheet iron with ¼ to 1-inch openings. This method is an improvement upon the old, so far as extracting and saving the gold is concerned. It is more economical in initial cost and maintenance, and about one twenty-fifth of the gold is saved in these sand launders.

Where the gravel contains much clay this method is not so simple and easy of adoption, for the balls of clay are not always broken up and they carry a good deal of the gold away into the tailing. In order to overcome this difficulty a number of devices have been tried experimentally. One of these consists of a revolving screen that works around a fixed axle, and inside iron chains are hooked up loosely, and attached to the bar are spikes or teeth. The balls of clay are caught between these teeth and the loose chains, so that they are broken up completely. The object of the loose chains is to agitate and stir up the material, and the fixed teeth of iron then act as a sort of rake to break up all coarse particles not separated by the agitation of the chains. As the screen revolves this action of chains and teeth goes on steadily and thoroughly digests all coarse clay substances that may enter it.

In order to solve the question of restoring dredged ground it was found that preliminary work must be undertaken to suit different kinds of soil. The first step was to classify and separate the different materials, and then the problem of distributing them so that the lighter soil could be deposited on the top could be worked out. In Australia, as elsewhere, stripping in advance has been found necessary to keep the surface soil separate and from entering the dredge. In the sand-box arrangements described, the fine silt is apt to be carried down into the underlying coarse gravel in spite of all that can be done, and thus it becomes lost to all future agricultural purposes. Stripping the surface soil in advance, with as little water as possible, the rich top soil can be separated.

At three different places in the state of Victoria excellent results have been obtained through some of the various devices in restoring dredged land. These are at the plants on Crooked River, on Livingston Creek, and at Eurobin, on Owens River, where several dredges are at work. At the latter place the dredges work the gravel and surface soil separately. The first operation is to strip the top soil and deposit it some distance off, and then the gravel is dredged and deposited nearer the dredge. The dredges on Owens River are replacing from 80 to 85 per cent. of the surface loam on top of the dredged gravel, so that the land, after all operations have ceased, is nearly as good for agricultural and grazing purposes as beforehand.

An idea of the working of this type of dredge, which has become general in parts of Australia where resurfacing of dredged land is considered imperative, may be gained from Fig. 1. The main sluice box *a* of this dredge is 4 feet wide and 18 inches deep. Its length from the drop-chute bars to the end of the delivery is 56½ feet, and an opening 5 feet long extends from the lower end of the main sluice box to the upper end of the extension loam launder *b*. When ordinary dredging is in operation the special loam chute is not in use. This extension loam launder is of the same width as the main launder at its upper end, but it gradually tapers down to less than half the width and half the depth. The upper end of the launder is movable, and can be raised or lowered as needed, so that when ordinary dredging is going on it is out of the way. It can be quickly lowered, however, into position when needed, and without much interruption of work.

This movable section *c* of the loam launder is made of iron plate bent into a semicircular form, and along the upper edges there are light iron flanges to fit on top of the upright sides of the main sluice box. This section when dropped into position extends along the top of the main sluice and bridges the 5-foot

space between the end of the sluice and the special extension loam launder. The resoiling material thus passes along this movable part into the extension launder and is deposited some distance from the gravel heap.

The movable sections of the launder are controlled by a continuous wire rope which is operated by a winch on the dredge framework. The device is so simple that it takes only 3 minutes to drop or lower the loam launder. Thus when the dredge is moved forward the top soil can be rapidly stripped and deposited some distance off, and then when the gravel bed is reached an interruption of only 3 minutes is required to change from top soil dredging to gravel dredging. Advance stripping thus proceeds steadily for a few hours each morning, and then the gravel dredging is begun. In fact, it has been found that 2 or 3 hours of advance stripping with this type of dredge prepares enough ground for ordinary gravel dredging that occupies the balance of the 24 hours. On an average it has been found that in advance stripping the top soil is removed at the rate of about 85 cubic yards per hour, against 60 cubic yards for gravel dredging.

In order to make advance stripping successful in this way a certain amount of water is needed to wash the loam along the launder. But the amount of water is far less than that required to keep the sluice box clear of ordinary gravel. Probably 10 per cent. of the amount of water required for ordinary gravel dredging will answer for advance stripping. The saving of this top soil from being washed away by the water and thus being lost in the gravel bed is a matter that requires a little additional forethought and preparation. Usually this is done by building small dams of brush, timber, or stone, so that the liquid part of the loam cannot rush back into the pit. There is bound to be a certain amount of waste in this way, but some dredging companies in Australia have had constructed sheet-iron spiles which are driven a foot into the soil around the loam beds to form protecting dams. These pieces of sheet iron are moved from place to place after the water in the soil has had a chance to percolate through the ground. Some sort of a movable dam of this nature is found to be more economical than stone or brush dams. Material for the latter is not always to be found when needed. A set of sheet irons for dam building will last indefinitely, and they can be set up as easily as a stone dam. The sheet irons are slightly curved so that a semicircular dam can be formed around any loam heap, with corner pieces turned at a sharp angle.

This process of saving the top soil for restoring the land to its previous fertile condition has, in addition to solving the problem of resurfacing, actually improved the economical output of the gold dredges. It has been found that more gold can be saved by separately stripping and conveying the overburden through the loam launder than by sending both gravel and overburden together through the sluice box. The result of this is that surface-restoration methods in gold dredging is becoming popular and general in many parts of Australia. There is no longer any conflict between the farmers and the miners in this respect, especially where the most advanced methods of dredging are pursued.

WET ASSAY OF GOLD

In the October issue of Mines and Minerals in the article on "Wet Assay of Gold" the exact ingredients of the solution were not given, and we are again indebted to Doctor Jerome Davis, of Raymond, Cal., for the correction. The solution is composed of potassium iodide and iodine, not potassium iodide and potassium. The solution is made as follows: Potassium iodide, 4 ounces; resublimed iodine, 2 ounces; distilled water, 94 ounces.

He further states that in this reaction the gold, while it forms a button, when the mercury is boiled out is in its primitive shape, that is crystalline. These crystals are needle shaped and are very distinct. While gold crystals are sometimes found native it is seldom that they are obtained artificially and at the same time give the yellow color of gold. Doctor Davis sends a number of these crystals whose shapes under a microscope are plainly seen. He further states that the crystalline form of gold is common in some of the dark slates of California, and some of the nuggets are beautiful. Sometimes the gold is in the form of a stalactite, which of course leads up to the well-known wire gold form.

THE LOG WASHER IN ZINC MINING

Written for Mines and Minerals, by Lucius L. Wittich

In Southern Missouri, 20 miles from West Plains, the nearest railroad point, is the Alice Mine of the Empire Zinc Co. The ore, which is zinc carbonate, or smithsonite, is shipped to the company's reduction plant at Mineral Point, Wis., where it is converted into oxide for the manufacture of paints. No other zinc producers are near the Alice Mine, which has been in almost constant operation for 12 years and has turned out as high as 58 tons of coarse and 96 tons of fine carbonates per week, 3 days of which time were devoted to conveying the ore to railroad.

To dress the Alice Mine ore, the old style Archimedean log washer, the invention of the Greek mathematician, is employed, and the log washer of today is little changed from the log washer of centuries ago. At the Alice Mine a double log washer is used, which also takes the place of jigs. Throughout the zinc and lead districts of Southern Missouri, Southeast Kansas, Northeast Oklahoma, and Northern Oklahoma, the log washer is not in use. The Alice is one of the few exceptions, and because of its remoteness from other districts it is out of the line of general travel and is seldom visited.

The hard oak log is 26 feet long and 16 inches in diameter. "Flights" of iron are attached spirally at intervals of 20 inches, from center to center, along the flat faces of the log, which is octagonal, and are so arranged that when the log revolves it has the effect of an auger. The logs are inclined at an angle of 4 degrees so that material emptied into the lower end of the trough will be carried toward the upper end. A constant stream of water empties into the trough at the upper end, washing away the fine dirt and mud, while the coarser ores are gradually worked to the upper end of the trough. Here the finer ore passes through a slat screen with ¾-inch openings while the coarser passes over. The different sizes go to different bins.

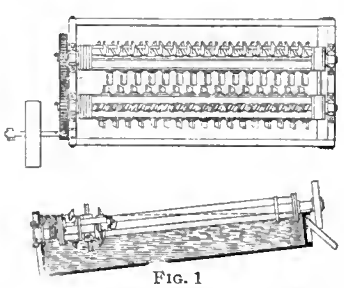


Fig. 1

Before entering the log washer the mine dirt is screened through a 5-inch bar grizzly, the coarse being culled out while all that passes through goes to the washer. The trough is built sufficiently large to permit of the easy accommodation of the logs, with the flights attached, there still being a space of 10 inches between the tips of the flights and the bottom of the trough, which, in the course of time becomes bedded with fine ore. This condition is desirable and the work of the log is more effective.

The flights are 9 inches long, 4 inches wide, and ¾-inch thick, and are attached to the log with lag screws. Fig. 1 shows how the faces of the flights are slanted to create a general forward movement of any material that is emptied into the trough.

The operations at the Alice Mine are conducted largely in open pits, although several shafts have been put down to ore, which is trammed to the odd little plant. The ore, which outcrops in places, is now being worked to a depth of 40 feet, and the

company reports 20,000 tons blocked out. It contains some iron and much flint. Of the mass hoisted for treatment, 60 per cent. is ore.

For hoisting, a 30-horsepower engine is used. For operating the washer, which revolves at the rate of 17 times per minute, there is a 10-horsepower engine stationed at a pond ¼ of a mile away; a 2-inch pump is operated by a 20-horsepower boiler. The water, after being elevated 50 feet to the mines, is used in the washer.

In the early stages of the development of the Alice Mine the ground was scooped out with scrapers, and wagons were employed to haul the dirt direct to the plant.

Laborers are hard to find, and are shiftless as a rule when found. About 12 men are employed. The cost of carting the ore to railroad is $2.75 per ton.

FAKE OIL COMPANIES

By Louis E. Aubury

Owing to the many misrepresentations and fraudulent practices resorted to by fake oil companies operating in California, resulting in continuous complaints to the State Mining Bureau, it is believed that investors at large may be benefited to an extent by the publication of a few suggestions from the Bureau, covering the initial points it is advisable to consider before delivering their money, never to return, into the hands of operators of these fake companies, where the intention is purely and simply to sell stock, and without the first idea or expectation of ever paying dividends.

The following are a few of the principal points suggested:

Has the advertising oil company under consideration a title to the alleged oil land being exploited; in what manner and for what consideration was title acquired, and is it actually within proven or unproven territory?

Many companies operating in unproven territory, advertise that they "own," or have "secured" such and such tracts of land; that they are within proven oil territory, and that the purchase was made for so much of the company's stock in addition to a cash consideration—frequently quite a large one—and added thereto is the payment of a royalty to the owner or owners when production is reached; the latter the only consideration against which the locator of the land will ever have any claim, and that a remote one.

This is of course an extreme, but not an unusual case of imposition, cited for the purpose of illustration, and with a view to opening up to prospective stockholders an avenue of thought which in turn may suggest many other material points not referred to herein.

Lands advertised in this manner are more often merely locations of government land, subject to numerous requirements of law and open to attack and litigation that may eventually render acquisition of the title from the government to the company in question an impossibility—and under any circumstances the payment of such a large and disproportionate consideration to the promoters of the company, stamps it on its face as a proposition purely for the benefit of the operators at the expense of the stockholders.

The promotion stock also is of course unloaded (since it is not intended the project shall prove a success) and the proceeds pocketed, in addition to the cash payments due on account of having discovered the land, and which are derived from the sale of the balance of the stock, which the liberal promoter in his goodness of heart also allows a grateful and eager public to acquire, with the understanding that an opportunity of a lifetime is presented and under no circumstances should be permitted to escape, even though all else is sacrificed. This, it is a matter of record, is done in only too many instances to the everlasting sorrow of the purchaser investing his or her all.

Continuing upon this subject of payment, for example, a company incorporated for $1,000,000, par value per share $1, advises in its prospectus that 500,000 shares have been "paid" for its property; which means literally that one-half of the capitalization of the company benefits only the promoters who discovered this valuable (?) tract in question—or more to the point, that the tract in question would answer their purpose. Possibly it is a location very close to the *edge of productive oil lands* acquired from the locator for a mere song, or solely upon an agreement to pay a royalty from the production derived from operating the land, there being many locators upon the edge of production, and on lands the geological formation of which precludes any possibility of striking oil in paying quantities, or at all, but who have the miners' undying faith in a bright and paying future for their claims. These claimants, usually having little or no money, are readily annexed by the fake promoter under such terms as above indicated; and operations on the land are begun and carried on upon an extensive scale, since these wily promoters in many instances extend their persuasive and convincing abilities to the supply companies, who furnish on credit all the necessary equipment and eventually seize it for failure of payment—but all that is desired has been accomplished by the fake promoter and permits him to begin immediate and elaborate operations which he advertises to the fullest extent, and more if possible; reciting in many cases just such a proposition as is outlined in the foregoing, but in perverted language so cunningly worded that the actual facts are not grasped by the uninitiated, whose attention and interest is maybe more carefully centered upon an illustration on the front page showing the oil fields and indicated in the background by an arrow, "The Lakeview Gusher," and in the foreground the company's derrick, with a note in a conspicuous place on the page something like this: "Just 9,000 feet to the northeast of our property is the great Lakeview Gusher, etc."

Since 9,000 feet is more than a mile and a half distant, it is by no means a surety that production is a possibility at the point the advertising company is operating.

A close study of the average fake prospectus will afford much food for reflection and investigation upon just such points as that above mentioned.

Are the officers of the company you have under consideration in good standing? Obtain a report on them from some commercial company.

The most extensive fake operators have followed their nefarious practice continuously for years, during which time they have repeatedly been roasted and fully exposed by the newspapers in all parts of the country. Notwithstanding this, they regularly incorporate company after company just as fast as the old ones play out, and with the same result, a certain class of the investing public never failing in its desire to contribute to the purses of these worse than highwaymen.

In addition, several other points are referred to as worthy of the most careful investigation—such as whether the company in question is overcapitalized, are commissions deducted from the sale of stock, and what are the salaries paid the officers of the company (in fake companies, of course, the promoters), all of which questions involve the scheme of such operators to make the company a paying proposition—for themselves.

It is not intended or believed that this suggestion will alarm the investing public to an unhealthy degree; nor is it intended to convey the idea that there are not any number of perfectly honest and legitimate enterprises worthy of consideration and investment, or that wildcatting in unproven territory is not the true source of the present wonderful production of oil in California, and deserves the most liberal encouragement from those who fully understand the chances they are taking and who can afford to invest in what may properly be called a gamble; but it is intended to, if possible, stimulate the stock-buying public into making a more thorough and careful investigation before placing its money in the hands of these fake operators solely upon the criminal misrepresentations contained in their exaggerated advertising literature.

CORRESPONDENCE

Haulage

Editor Mines and Minerals:

SIR:—Is it possible to have an endless-rope haulage for a distance of 1½ miles to bring coal out of a mine and pass around a curve at the foot of an incline or slope to a tipple, the length of the incline or slope to tipple top being 1,200 feet, making an angle with the horizon 35 degrees, the cars to carry from 2,000 pounds to 2,500 pounds of coal? If it is possible, what system of endless haulage would you prefer, rope on top or under cars (the cars are well topped)? What size and kind of cable would it take? What horsepower and kind of engine would you prefer?

If the above is not economically possible, what system would you prefer to hoist coal from the foot of an incline to the tipple top, 1,200 feet, making an angle of 35 degrees? Name the system of haulage you would adopt with size of cable and engine. CHESTER MOON

Alberta, Canada

THE LEYDEN, COLO., MINE FIRE

On December 14, 1910, 10 men were suffocated in lignite mines at Leyden, Colo., by a fire, the origin of which is unknown. Leyden is 14 miles west of Denver, near the foot-hills of the Rocky Mountains. The seam worked is from 8 to 12 feet in thickness and does not generate gas. The analysis of the seam, approximately 16.50 per cent. moisture, 3.25 per cent. ash, 31.25 per cent. volatile matter, 48.50 per cent. fixed carbon, and sulphur under .50 per cent., would indicate that the freshly mined coal, at least, was not inflammable. The mine was maintained in most excellent condition, but was regularly watered on Saturdays and Sundays as a matter of precaution.

The surface plant was recognized as one of the best in the state and aside from the usual head-frame and tipple, consisted of a battery of 7 boilers, hoisting engine, air compressor, electric generator with the usual machine and repair shops, all of which were destroyed by the fire, entailing a loss of between $85,000 and $100,000. The capacity of the plant was about 1,500 tons daily, and 275 men were employed.

MEXICAN TERRITORY OF QUINTANA ROO

By William W. Canada, Vera Cruz

The territory of Quintana Roo forms a part of the peninsula of Yucatan, and is to all intents and purposes "terra incognita" to both the inhabitants of Mexico and the United States. Its area approximates 25,000 square miles, between the 18th and 21st parallels of latitude. It has a coast line of about 500 miles along the eastern shore of Yucatan, and is bounded on the north by the state of Yucatan, on the east by the Caribbean Sea, on the south and southwest by British Honduras and Guatemala, and on the west by the state of Campeche. The entire coast is low and sandy, studded with reefs which extend some 500 yards from the mainland.

There are a few natural harbors on this coast where a vessel may find shelter against any winds except those from the north. The average depth of water is about 18 feet. Bahia de la Ascencion and Bahia del Espiritu Santo are the chief harbors. The Mexican Government commenced to construct an artificial harbor at Xcalak, the geographical position of which is 87° 48′ longitude and 18° 15′ 30″ north latitude. It is the only practical natural outlet for the products of the interior.

So far reefs have been blasted and an entrance has been made having an average depth of water of about 20 feet at mean low tide. The port, when completed, will cover over 100 acres. At present all construction work has ceased. There are custom houses at the two ports of entry, Bahia de la Ascencion and Payo Obispo (Puerto de Chetumal).

The territory of Quintana Roo has but one navigable river, the Rio Hondo, which empties into the Bahia de Chetumal and also forms the boundary between British Honduras and Mexico. Once across the bar at the entrance to the river, it is navigable some 80 miles from the coast; but on the bar referred to only about 6 feet of water is found. The Rio Hondo can only be entered from the Bahia de Chetumal on the Mexican side of the river, for there is an old Spanish galleon sunk in the center of its mouth, causing a silting of the British side.

The main obstacle to opening up the country is the yet incomplete subjugation of the Maya Indians, who hold the entire territory with the exception of the strip forming the coast line. Little by little, however, they are being forced farther back into the interior.

Evidence of the existence of coal, petroleum, and asphalt is not wanting, though no prospecting has yet been done with a view of developing this field. There is evidence that copper has been mined in large quantities and that it was shipped out of this country. The entire region hereabouts is cupriferous, and hence even the water is affected, for it is unfit for use and cannot even be utilized in some instances for feedwater in steam boilers.

On the shores of the Bahia de la Ascencion and at Campamento Vega piles of copper ore of a high grade are occasionally found among the wreckage, which would indicate that at some time long ago ore was shipped from here. Another strong indication of the existence of copper is proved by the Rio Azul (Blue River), an insignificant affluent of the Rio Hondo. The water is actually blue with the amount of sulphate of copper it contains. In the dry season it carries as high as 6 per cent. of copper in solution, and crystals of the sulphate are deposited on the margin of the stream. It may therefore be safely assumed that immense copper ore deposits must exist in this vicinity. Springs having their sources in the neighborhood of the Bacalar Laguna give evidence also of the presence of copper.

There are no public records to which one may refer. If they ever existed, they are now lost or destroyed, and nothing relating to former mining operations is known with any degree of certainty. In the church library at Corozal, in British Honduras, there is a small work in which two Spanish padres relate their experiences while ascending the Rio Hondo in the time of the first Spanish viceroys. Among other matters, it is mentioned by these priests that they had found in their travels a large number of copper utensils of all descriptions, which must have been turned out by the Indians of that time or before. The priests, however, did not discover the source of the copper ore.

The ancient ruins of cities throughout the present known territory are positive indications of the existence of a large and prosperous population before or since the time of the Spanish conquest, of which there are no other traces left. Izmul, one of these abandoned cities, has in its immediate vicinity the ruins of prehistoric structures, some of them of enormous size and beautifully sculptured. Judging from present indications, the ancient town must have housed at least 50,000 people. There is not even a trace to be found of a civilized being anywhere within 100 miles from there.

No doubt can exist that this region was very thickly settled at some time, that its inhabitants were highly civilized, as is proved by their architecture, and that the country must have produced something not known at the present time. The destruction and abandonment visible at every hand was doubtless due to the internecine wars among the natives themselves.

A noteworthy instance of the intelligence and civilization of the people at some time remote is clearly demonstrated by an artificial waterway constructed by them. Reference is made to what is now known as the Rio Kik. This is an artificial canal connecting the Bahia de Chetumal with the Bahia de la Ascencion, by way of the Bahia de Espiritu Santo. Unde-

niable traces of the embankments, constructed of coral rocks, and of the watch towers along its length, are plainly seen. The engineering skill, as shown in this work, was of a high order, as the construction of this canal reduced the distance between the extreme points mentioned from 300 miles on sea to 60 miles by canal, and at the same time avoided the dangers attending coastwise navigation in those days.

Present labor conditions are not favorable to the intending settler in this region. The native is not to be counted on, and all labor must be brought from other parts. Dyewood, cedar, and mahogany are gotten out by negro labor from British Honduras and Jamaica. The chicle gatherers have come from Tuxpan, state of Vera Cruz. The Maya Indian will not work. He may possibly plant a small patch of corn, and this, with his fishing, will keep him content. Should he develop a taste for meat, he finds game in the woods.

EDUCATIONAL

Various lines of research work are being carried on in the post-graduate electrical engineering course of the Massachusetts Institute of Technology. Professors Pender and Wickenden have charge of the work. Part of the research work relates to the effects of heat treatment on the magnetic qualities of silicon iron; certain transient phenomena that may occur in long electric circuits; the effect of high frequencies on the permeability of iron; the effective resistance and reactance of steel rails when conveying alternating currents; the selective action of spark-gap lightning arresters with respect to frequency; the reflection of light from walls or ceilings; the disruptive strength of rubber-insulated coatings on wires, etc. Certain of these are continuations of work started last year, and researches in each will be carried on as may be convenient and needful to get knowledge of the phenomena under investigation.

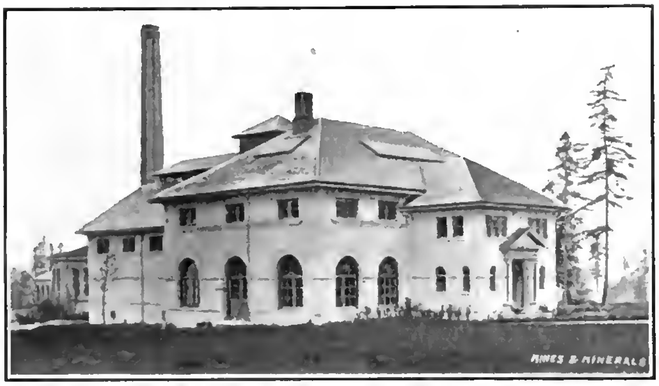


Fig. 1. Mines Building, University of Washington

The Pennsylvania State College School of Mines has a mine-rescue station whose object is to give instruction in mine-rescue apparatus and work. There is a large "smoke room" built gas-tight, in which noxious and unbreathable gases may be liberated, and in which the men may be trained in the use of the helmets in such atmospheres. In the "smoke room" there is a wooden structure representing a mine entry with an overcast, and students are trained to enter this with the helmets, and working in the noxious atmosphere, to build brick stoppings, to climb through the overcast, carrying loads, and to do other work such as would have to be done by a rescue party. On the night of the opening, the "smoke room" was filled with the fumes of burning sulphur, followed by nitrous oxide, the smoke becoming so dense that the electric lights carried by men in the room could not be seen at a distance of 6 inches. The room is provided with glass doors and windows, which permit the interior to be seen from the outside.

At the University of Washington School of Mines, Dean Milnor Roberts has started a short course in mining for practical mining men. Fig. 1 shows the Mines Building, which has been newly equipped with assay furnaces, additional balances, power-driven sampling equipment, and drafting tables. All the usual metallurgical machinery, stamps, rolls, crushers, concentrating tables, classifiers, etc., are installed, so that instruction may be given in the following subjects: The mineral industry, mining, fire assaying, metallurgy, chemistry, mineralogy, geology, mining law, and surveying. Trips to mines, mills, and smelters are also contemplated. No examinations are required for entrance to the short course, and the studies are so arranged that any person interested in mining may follow them, regardless of his previous training. There are no charges, except for books and materials actually used. A full description of the course may be obtained by applying to Dean Milnor Roberts, University of Washington, Seattle.

DIAMONDS IN GERMAN SOUTH AFRICA

Interest has been awakened during the past year by the discovery of an extensive diamondiferous region in the German colony in southwest Africa, near Luderitz Bay. The character of the newly discovered deposits is widely different from that of the peculiar "blue earth" of the great "pockets" in and near Kimberley. The diamonds of the German colony are found scattered in the coarse sand covering an extensive area, in company with agates and others of the more valued forms of quartz. For the most part they occur in the form of tolerably perfect octahedra, and are of the first quality. Their weight is, however, almost uniformly low, ranging from one-fifth to three-fourths of a carat. The average weight is about one-third of a carat.

The novel form of occurrence in these diamond fields has awakened scientific curiosity. Geologists have, in rare instances, encountered limited areas of sand containing well-formed diamonds unaccompanied by the minerals usually associated with this precious stone. But never before has the phenomenon been observed on a scale warranting industrial exploitation.

The diamonds are found in an irregular depression of a desolate region about 1 mile broad by some 30 long, stretching in an arc from Luderitz Bay to Elizabeth Bay.

As in the case of placer deposits of gold, it is evident that the diamonds are remote from their place of formation.

M. E. Frances, of Johannesburg, has carefully studied the problem, and his conclusions may be summarized as follows:

These diamonds are derived from fissures of Kimbertite, now possibly in the bed of the ocean. They are not wind transported, save as they are blown with other small pebbles a few yards from one place to another. At the time when the land was under water these diamonds lay in fissures in the bed of the ocean. The land slowly rose, and for a long geological period, and a comparatively recent one, the long depression was a shallow arm of the sea. From the south, where now is Elizabeth Bay, swept a strong current to the northern outlet, and in the shallow water the action of this current and of the waves deposited the diamonds in the sands. The lands then rose above the surface. The high winds have since blown the finer particles away, and the present gravel deposits where the diamonds are found represent the concentrated result.—*United States Consular Report.*

According to statistics compiled by the United States Bureau of Manufacturers, the consumption of coal in the United States is more than twice as great as that in any other country, and nearly equals the combined consumption of the United Kingdom, Germany, France, and Belgium, and is actually greater per capita than in the United Kingdom.

THE SIPHON IN MINING

Written for Mines and Minerals, by J. T. Beard

Its Use, Adaptation, and Limitations in Practice. Method of Calculating Dimensions

Among the common appliances used in mining, probably none possess a greater interest, considered from the standpoint of economy, than the ordinary siphon. Also, it would appear to be equally true that notwithstanding its simplicity and common use, few of these appliances are as little understood in respect to the theory of their action as the siphon. The writer's attention has been called to this lack of understanding recently by replies of correspondents to a question asking what would be the discharge of water through a 4-inch siphon pipe, rising a vertical height of 17 feet in a distance of 800 feet, and then falling 75 feet vertically, in the remaining 300 feet, the entire pipe line having a length of 1,100 feet, and the difference of elevation between the intake and the discharge ends of the pipe being 75 − 17 = 58 feet. The replies given to this question found, by the use of Eytelwein's formula for the flow of water in pipes, that this siphon would discharge 259+ gallons per minute; and, by the use of Hawksley's formula, 179+ gallons per minute. Here is a wide difference of 80 gallons each minute, and it is not surprising to find on investigation that neither of these answers are even approximately correct. It is never safe to consider the siphon in the same manner as a pipe line in which the flow is caused by gravity. Attention has been called to this on former occasions.

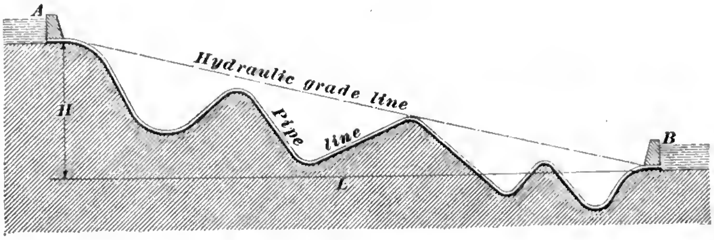


FIG. 1

To properly understand the action of the siphon it must be remembered that it consists of two legs or branches in each of which the flow of water is induced by different means. The two branches, though connected and forming one continuous line of pipe, act, for the most part, independently of each other. In the upper or suction leg of a siphon the water flows from the upper basin to the crown, or summmit, which is the highest point in the pipe line. This flow is not due to gravity, as the water rises instead of falls. It is caused by the atmospheric pressure acting on the surface of the water in the basin, less the gravity head or rise in the pipe. The action here is the same as in the suction pipe of a pump. The water will not rise to a greater height than what the atmospheric pressure will support, which at sea level, under normal conditions, is about 34 feet, and then allowance must be made for the necessary friction and velocity heads required to produce the flow. In the discharge leg of the siphon, on the other hand, the water flows by gravity. Here the head producing the flow is the gravity head or fall in the pipe from the crown to the surface of the water in the lower basin, or the point of discharge, less the atmospheric head, which now acts to oppose the flow. As before, allowance must be made here also for the necessary friction and velocity heads.

The usual method in dealing with siphons is to consider the siphon acting as a whole. In this case the atmospheric pressure at the intake balances the same at the discharge end of the siphon, while the head producing the flow (effective head) is the difference between the discharge and the suction heads, or the vertical fall and rise of the pipe. The friction head is calculated for the entire length of the siphon, and the velocity head is constant for a uniform diameter throughout the pipe. It will be seen shortly that this method of procedure is correct only under certain favorable conditions. The method does not show when the siphon will fail to work, or run perhaps for a short time and then go dry, nor does it in any way suggest the reason for such failure to operate.

The old Bernouilli theory or principle, which applies to the flow of fluids, makes the head (H) producing the flow equal to the sum of the gravity, friction, and velocity heads, as expressed by the formula

$$H = h_g + h_f + h_v \qquad (1)$$

The gravity head (h_g) is the total net vertical rise or fall in the pipe considered. It may be either positive (rise) or negative (fall). In the former case it acts to balance an equal portion of the original head (H) producing the flow (often called the pressure head), the difference between the pressure head and the gravity head being then the effective head. This condition is expressed by the formula

$$H - h_g = h_f + h_v \qquad (2)$$

When the pipe line considered has a net fall, the gravity head (h_g) in formula 1 is negative and acts to assist the pressure head to produce the flow. In this case, the effective head is equal to the sum of the two heads, as expressed by the formula

$$H + h_g = h_f + h_v \qquad (3)$$

It will be readily seen that formula 2 applies to the suction leg, and formula 3 to the discharge leg of the siphon.

Before going further it will be well to write the formulas for calculating the friction and the velocity heads in terms of the flow of the water in the pipe. The friction head is a measure of the pipe resistance, and depends upon the condition of the pipe, its length and diameter, and the quantity of the flow, as given by the formula

$$h_f = \frac{f\,l\,G^2}{8\,d^5} \qquad (4)$$

in which

h_f = friction head (ft.);
f = coefficient of friction;
l = length of pipe line (ft.);
d = diameter of pipe (in.);
G = quantity of flow (gal. per min.).

In mining practice, the pipes become corroded by the action of the mine water, and for this reason it is not generally safe to assume a less value for the coefficient of friction than $f = .01$, which is a fair average value.

The velocity head is found by transposing the well-known formula for velocity due to a given head, $v^2 = 2\,g\,h$, and substituting equivalent values for v in terms of the flow G and the diameter of pipe d; thus:

$$h_v = \frac{v^2}{2\,g} = \frac{1}{64.32}\left(\frac{231\,G}{60 \times .7854\,d^2 \times 12}\right)^2 = .0026\,\frac{G^2}{d^4} \qquad (5)$$

Now, substituting for h_f and h_v in formula 2 their respective values taken from formulas 4 and 5

$$H - h_g = \frac{f\,l\,G^2}{8\,d^5} + .0026\,\frac{G^2}{d^4}$$

or

$$H - h_g = \frac{G^2}{d^4}\left(\frac{f\,l}{8\,d} + .0026\right)$$

and

$$G = d^2\sqrt{\frac{H - h_g}{\frac{f\,l}{8\,d} + .0026}} \qquad (6)$$

Formula 6 gives the flow (G) in gallons per minute, in any pipe having a length l, in feet, and a diameter d, in inches, when the effective head ($H - h_g$), in feet, is known.

In siphon work this formula should be applied to the suction leg first, to ascertain what quantity of water the atmospheric pressure will cause to flow from the upper or supply basin up the pipe to the crown, from whence it will gravitate naturally to the lower basin. The flow of water in the siphon is limited to the flow in the suction leg induced by the pressure of the atmosphere, since the discharge can never exceed the

intake. The discharge, however, may reduce the flow in the siphon below what the atmospheric pressure could supply. This analysis of the proposition it will be granted is correct when it is remembered that in a perfect siphon there exists a more or less perfect vacuum at the crown, or the space here is filled with water, depending on conditions now to be explained.

Considered as a whole, the siphon is a pipe of various length bent upward at some point so as to present in profile one ascending and one descending portion or leg. The point of bend is

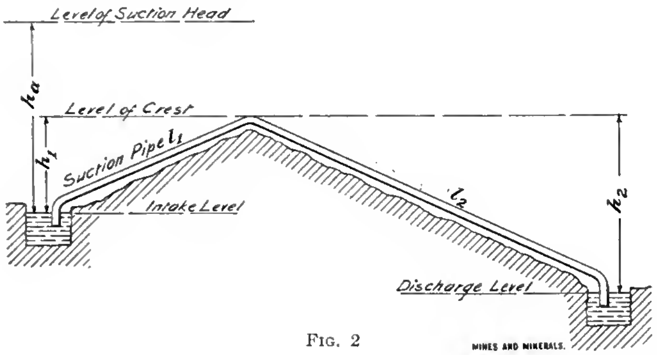


FIG. 2

called the crown or summit of the siphon. The two legs may have any lengths and inclinations, provided only that the mouth of the ascending portion or suction leg is at a sufficient elevation above the mouth of the descending portion or discharge leg to cause the water to flow through the pipe from the upper or supply basin to the lower or discharge basin. The difference between the elevations of the surface of the water in these two basins is called the effective head of the siphon. The intake end of the siphon or mouth of the suction pipe must be sufficiently submerged beneath the surface of the supply basin to prevent the entrance of air into the pipe with the water. Preferably, the discharge end of the siphon should likewise be submerged in the water of the lower basin, but this is not always necessary for the successful operation of the device.

In order that a siphon may work, the following requirements must be fulfiled: (1) The suction head (h_1, Fig. 2), which is the gravity head or vertical rise of the suction pipe, must be less than the atmospheric head (h_a), and the difference between them ($h_a - h_1$) must be sufficient to provide for the required friction head (h_f) and velocity head (h_v) in the suction pipe or leg, as expressed by the formula

$$h_a - h_1 = \frac{G^2}{d_1^4}\left(\frac{f\,l_1}{8\,d_1} + .0026\right) \qquad (7)$$

Formula 7 shows that the required effective head for any flow G is determined by the diameter and length of the pipe, and the flow will cease ($G=0$) when $h_1 = h_a$. Also, it is possible to increase the flow for any given effective head by increasing the diameter or decreasing the length of the pipe. (2) The flow of water in the suction leg of the siphon induced by the pressure of the atmosphere must at least be equal to the flow in the discharge leg induced by gravity. If this requirement is not fulfilled the siphon will have a tendency to empty itself, and will continue in operation but a limited time when it will run dry, unless the discharge is controlled by a cock, reducing nozzle, or other device. Equating the two expressions for the natural flow in the two legs of the siphon by means of formula 6, gives

$$d_1^2\sqrt{\frac{h_a-h_1}{\frac{f\,l_1}{8\,d_1}+.0026}} = d_2^2\sqrt{\frac{h_2-h_a}{\frac{f\,l_2}{8\,d_2}+.0026}}$$

Which, after simplifying, may be written:

$$\left(\frac{d_1}{d_2}\right)^5 \frac{h_a-h_1}{h_2-h_a} = \frac{f\,l_1+.0208\,d_1}{f\,l_2+.0208\,d_2} \qquad (8)$$

Any siphon in which the valuation of the first member of formula 8 is not equal to or greater than that of the second member of this formula, will tend to empty itself, and will run dry after a limited period of time.

To show the application of formula 8 and to verify the statements made at the beginning of this article in criticism of the answers given to a certain siphon question, let us investigate the same question, which was as follows:

Question.—What would be the quantity and velocity of water flowing in a 4-inch siphon line of the following dimensions: Distance to crest, 800 feet; from crest to discharge, 300 feet; difference of elevation from bottom of sump to top of crest, 17 feet; and from top of crest to discharge, 75 feet?

According to the data given, the suction head of this siphon is, as shown in Fig. 3, $h_1 = 17$ feet, and the length of the suction pipe $l_1 = 800$ feet; the discharge head is $h_2 = 75$ feet, and the length of the discharge pipe only $l_2 = 300$ feet. The diameter of the pipe is uniform, 4 inches throughout the entire length of siphon. Assuming the siphon is located at sea level, and the atmospheric head expressed as water column is $h_a = 34$ feet, and using a coefficient of friction $f = .01$, and substituting these values in formula 8, we have, since the diameter ratio $\left(\frac{d_1}{d_2}\right)$ is 1, in this case,

$$\frac{34-17}{75-34}\left\{\begin{array}{c}\text{equal to}\\ \text{or greater than}\end{array}\right\}\frac{.01\ (800)+.0208\times4}{.01\ (300)+.0208\times4}$$

or

$$\frac{17}{41}\left\{\begin{array}{c}\text{equal to}\\ \text{or greater than}\end{array}\right\}\frac{8.0832}{3.0832}$$

but

$$.4 \text{ is less than } 2.6$$

Since the valuation of the first member of formula 8 is shown in this case to be less than that of the second member, the siphon will speedily empty itself if started full.

We can investigate this case in another way that may appeal more strongly to the practical man. For example, apply formula 6, first to the conditions that exist in the suction pipe to ascertain what quantity of water the atmospheric pressure (sea level) will deliver at the crest of the siphon; and second, by means of the same formula, with the change of the effective head only, so as to represent the conditions in the discharge pipe, find the quantity of water that would gravitate through this pipe from the crest of the lower basin; thus,

$$\text{Suction flow } G = 4^2\sqrt{\frac{34-17}{\frac{.01(800)}{8\times4}+.0026}} = 16\sqrt{\frac{17}{.2526}}$$

$$= 131.2 \text{ gal. per min.}$$

$$\text{Discharge (full pipe) } G = 4^2\sqrt{\frac{75-34}{\frac{.01(300)}{8\times4}+.0026}} = 16\sqrt{\frac{41}{.09635}}$$

$$= 329.6 \text{ gal. per min.}$$

This shows clearly that the water will flow from the crest to the lower basin under the influence of gravity very much faster than the atmospheric pressure can force water up this long suction pipe.

One step further will be of interest, in the practical operation of siphons, and that is to determine what size of pipe

FIG. 3

(diameter) could be used for the discharge pipe, in the case just cited, in order that the pipe would continue to run full. Formula 8 may be written

$$d_2 = d_1\sqrt[5]{\frac{h_a-h_1}{h_2-h_a}\left(\frac{f\,l_2+.0208\,d_2}{f\,l_1+.0208\,d_1}\right)} \qquad (9)$$

The value of d_2 in this formula can only be found by trial. Taking $d_1 = 4$ and substituting the other values given previously it is found by trial, $d_2 = 2.76$; or a $2\frac{3}{4}$-inch discharge pipe

could have been used, in this case, with good results. The quantity of water the siphon would discharge in that case would be the same as found above for the suction flow; namely, 131+ gallons per minute. This flow may be calculated for the entire siphon when the diameter is not uniform by the formula

$$G=\sqrt{\frac{h_2-h_1}{\frac{f}{8}\left(\frac{l_1}{d_1^5}+\frac{l_2}{d_2^5}\right)+.0026\left(\frac{1}{d_1^4}+\frac{1}{d_2^4}\right)}} \qquad (10)$$

Thus, substituting the given values

$$G=\sqrt{\frac{75-17}{\frac{.01}{8}\left(\frac{800}{4^5}+\frac{300}{2.76^5}\right)+.0026\left(\frac{1}{4^4}+\frac{1}{2.76^4}\right)}}=131+ \text{ gal. per min.}$$

The same formula may be extended indefinitely, and be applied to any pipe line, not a siphon, where the diameter of the pipe is different for different sections of the pipe line, regardless of the number of such sections. The practical application of such formula is found in determining the flow in a pipe line that crosses several valleys and hills, as shown in Fig. 1, where the diameter of the pipe in each section is proportioned to the fifth root of the ratio of the length of pipe in that section to the head for the same section.

MINING IN SPAIN

By Consul Robert Frazer, Jr., Valencia

Spanish mining industries are passing through a severe crisis. Partial strikes for shorter hours in principal mining centers have culminated in a struggle between capital and labor in the iron-mining district of the Basque Provinces, in which both sides have declined arbitration and direct government intervention, rejecting every proposition to compromise.

In view of the strained situation created affecting one of the chief sources of Spain's natural wealth, the government, aided by the Institute of Social Reforms, whose mission is to solve labor and social problems, is preparing legislation dealing with the question of mining labor in its economic and social aspects. The following official statistics of daily wages and hours of labor in the principal mining regions of the peninsula are illustrative of the conditions obtaining in the country:

District	Mine	Hours of Labor	Under-ground Wage	Outside Wage
Albacete	Sulphur	10	$.73	$.55
Cartagena	Lead	8	.55	.37
Murcia	Lead	12	.54	.36
Teruel	Iron ore	10	.58	.58
Ciudad Real	Lead	8	.68	.36
Jaen	Lead	8	.90	.45
Cordova	Lead and coal	8	.65	.42
Seville	Lead and coal	10	.63	.36
Asturias	Coal	8	.81	.32
Leon	Coal	10	.81	.45
Palencia	Coal	7	.83	.42
Santander	Iron ore	10	.59	.45
Basque Provinces	Iron ore	*	.58	.42

* Eleven hours, summer; nine hours, winter; and ten hours in intermediate seasons.

The first four mines are in the region of Valencia. The last named have produced in the 32 years they have been operated by modern methods and machinery, 150,000,000 tons of iron ore, valued at about $246,000,000, of which about 140,000,000 tons went to Great Britain. The reserve yet to be mined in remunerative conditions is estimated at 61,000,000 tons.

The position of mine owners in the present dispute is that, in the actual precarious situation of Spanish mining industries in their relation to foreign competition in the world's markets, no concession whatever is possible in the reduction of hours of work or increase of wages, while miners hold that the hours of labor in the north of Spain, the low wages and increased cost of food, clothing, house rent, and other elements of living, place them at a disadvantage in comparison with the miners of nearly every other European country.

The difference between the wages of miners and those who handle ore on the surface is very marked, except in the iron mines of Tetuel (Ojos Negros), where little underground mining has been necessary, as ore is dug mostly from the surface.

Finally, the difference in hours of labor and wages in various mining sections of the country does not argue a difference in the supply of labor available, but is due to the kind of ore operated, and the depth, humidity, ventilation, and general sanitary conditions under which the miner works.

HEAT OF COMBUSTION OF EXPLOSIVES*

In order to obtain the true explosion temperature and heat of combustion, it is necessary to know the composition of the explosion gases at the moment of completion of the explosion in the closed vessel. This differs considerably from that found on analysis, on account of the reactions taking place during the cooling. If $A=CO_2$, $B=$the hydrogen, $C=$the CO plus three times the methane, $D=$the water less the methane, all expressed from the analysis as gram molecules per gram of explosive fired, then

$$\frac{(C+x)(D+x)}{(A-x)(B-x)}=K$$

For temperatures about 2,000°, $K=6.6$ (Langen), or 5.5 (Mallard and Le Chatelier). The composition of the gases before cooling is thus obtained by deducting x from the gram molecules of CO_2 and H_2, determined analytically, per gram of explosive, and the addition of x to those of CO and H_2.

The true heat of combustion is calculated from that determined experimentally by deducting 50,600 calories for each gram molecule of methane found, and 10,000 (Ostwald) or 10,100 (Berthelot) calories for each gram molecule of CO_2 formed during cooling.

In order to obtain more accurate results, gravimetric methods are substituted for the usual gas analysis by absorption and measurement of volume. The sample is placed in the bomb usually 1 to 1.5 grams of dry powder in a bomb 10.1 cubic centimeters content, or 10 grams of other explosive in a bomb of 750 cubic centimeters. After sweeping out free air with dry CO_2 and closing the valves, the charge is fired, the bomb allowed to cool, and the gases—consisting of H_2O, CO_2, CO, CH_4, H_2 and N_2—passed slowly through a weighed $CaCl_2$ tube and potash bulbs. The mixture of CO, CH_4, H_2, and N_2 is then led over hot copper oxide in a hard glass tube 25 to 50 centimeters long, connected to a $CaCl_2$ tube and potash bulbs. The CH_4 and (by difference) the CO are determined in a fresh analysis by leading the gases issuing from the first potash bulbs into an absorption apparatus containing cuprous chloride solution, which is renewed several times during an analysis from a vessel at a higher level, through towers containing dilute sulphuric acid (to absorb ammonia), concentrated sulphuric acid (to keep back moisture), and thence to the combustion tube. The CO_2, weighed in the second potash bulbs, will now be due to the CH_4 alone. If the two firings are carried out under exactly similar conditions, the results of the two analyses will be directly comparable. The gases are finally swept out of the bombs, $CaCl_2$ tube, and potash bulbs by a stream of dry CO_2 free air, which, to avoid oxidation of the CO absorbed in the cuprous chloride, has been passed through alkaline pyrogallol, and the rubber connection leading to the cuprous chloride apparatus closed with a pinch cock. By raising the level of the cuprous chloride solution the gases are driven over into the sulphuric acid towers, and thence, with a stream of dry CO_2 free air, into the combustion tube and bulbs. By means of a pump with a drying tube interposed, the $CaCl_2$ tube and bomb, which is warmed to 50° C., are evacuated, and the water remaining in the bomb thus transferred. The nitrogen is determined as described in the preceding abstract.

*O. Poppenberg and E. Stephan (Zeit. f. d. ges. Schiess-und Sprengstoffwesen, 1909, 4,281 and 305). The Analyst, November, 1909, pages 497-498 (E. H. C.).

TESTS FOR COPPER MINERALS

Written for Mines and Minerals, by Evans W. Buskett

Early Discoveries of Copper in America. Properties of Copper and Description of Its Ores

Copper was used by prehistoric man for tools, weapons, ornaments, and domestic utensils. The use of copper and bronze was a step toward civilization, and great progress was made in working these metals. The ancients probably made their bronzes by melting copper and zinc ores with carbon, the product being a crude mixture of the two metals, with copper predominating. The Romans obtained their supply of copper from Cyprus, in the Mediterranean Sea.

Probably the first mention of copper in the United States was by Verrazano, who reported that he saw beads of copper in the ears of Indians along the New England coast. The Indians along the St. Lawrence River told Jaques Cartier that the copper they had came from the Northwest. Champlain, in 1632, mentioned finding copper on an island in a lake northwest of Lake Huron. Alexander Henry visited the Ontonagon River in 1765 and again in 1771, and with him went men who mined copper along the banks of the river. In 1832, Schoolcraft visited the Ontonagon River and found copper, thus confirming the reports of Henry and his men.

Copper was mined in colonial times in Connecticut, New Jersey, Pennsylvania, and Maryland. The Schuyler Mine, at Arlington, N. J., was the first producer in the United States, but copper mining was not firmly established until the discovery of the native copper deposits of Keweenaw Point, Mich. This was followed by the discovery of the Cliff mines in the Lake Superior district, from which shipments were made in 1846. The Calumet was discovered in 1864, and consolidated with the Hecla in 1866.

The Anaconda Mine, at Butte, Mont., was opened in 1880, and large bodies of sulphide ores were discovered carrying as high as 50 per cent. copper. This find resulted in the rapid development of the district, which soon took the first place in the copper production of the United States.

Copper is red in color by reflected light, but in thin sheets transmits a greenish-blue color. The metal in the volatilized state shows these same greenish-blue tints. Copper is excelled in malleability by gold and silver, but is not as ductile as it is malleable. In tenacity it ranks next to iron, its tensile strength often being as high as 60,000 pounds per square inch. It conducts heat better than any metal except gold, and is next to silver in its electrical conductivity. Copper does not oxidize in dry air, but in moist air becomes coated with a film of oxides which protects the metal from further oxidation; moist air containing carbon dioxide forms a green coating of verdigris, which is a copper carbonate.

Copper alloys with gold, silver, lead, and zinc in various proportions. Bronze is an alloy of copper and tin, or aluminum and copper. Brass is copper and zinc, but sometimes contains lead. German silver is composed of copper, nickel, and zinc. The silver coins of the United States contain 10 per cent. copper, and 90 per cent. silver, the copper making the coin harder.

The specific gravity of copper varies according to its physical treatment. The specific gravity of electrolytic copper is 8.914; melted, 8.921; rolled and hammered copper, 8.952 to 8.958 (Marchand & Scheerer). The hardest of copper varies from 2.5 to 3.

Copper melts at 1,080.5° C. It is only slightly volatile. Its linear expansion is .0017; its specific heat is .094.

The valence of copper is one and three, the metal forming two series of salts, cupric and cuprous. Its atomic weight is 63.6; atomic volume, 72. Cuprous oxide, Cu_2O, occurs native, but may be prepared by igniting cupric oxide with metallic copper. Cupric oxide also occurs in nature in the mineral tenorite, but may be prepared by igniting cupric nitrate, sulphate, or carbonate, in air.

Copper dissolves readily in nitric acid, and is soluble in concentrated sulphuric acid. A saturated solution of hydrochloric acid will dissolve copper forming copper chloride.

Fixed alkalies added to solutions of cupric salts precipitate cupric hydroxides, which are soluble in ammonia and give a deep blue solution. Ammonia first precipitates pale blue basic salts, then deep blue hydroxide, which dissolves in an excess of the precipitant. Potassium cyanide precipitates a yellowish-green cupric cyanide which dissolves in excess of cyanide, forming a double cyanide of copper and potassium.

Hydrogen sulphide precipitates from solutions of cupric salts a black sulphide CuS, soluble in nitric acid, difficultly soluble in hydrochloric acid; insoluble in hot dilute sulphuric acid; insoluble in fixed alkalies; and but slightly soluble in ammonium sulphide. The copper sulphide is soluble in potassium cyanide, and by this means may be separated from cadmium.

Copper is easily precipitated from solutions of both cupric and cuprous salts by the following metals: Zinc, cadmium, tin, aluminum, lead, iron, cobalt, nickel, bismuth, and magnesium.

Copper occurs in many countries, but only in a few where the deposits are concentrated sufficiently to be worked at a profit.

The principal copper ores are the native copper deposits, copper sulphides, and carbonates.

The sulphides are:

Bornite: Luster, metallic. Color, copper red to brown. Tarnishes on exposure to the air. Streak, grayish black. Brittle. Composition $FeCu_3S_3$; copper, 55.58 per cent.; iron, 16.36 per cent.; sulphur, 28.06 per cent.

Chalcocite: Copper glance. Luster, metallic. Color, blackish lead gray. Streak, lead gray. Hardness, 2.5–3.0. Specific gravity, 5.5–5.8. Composition Cu_2S; copper, 79.8 per cent.; sulphur, 20.2 per cent.

Chalcopyrite: Luster, metallic. Color, brass yellow. Streak, greenish black. Hardness, 3.5–4.0. Specific gravity, 4.1–4.3. Composition $CuFeS_2$; copper, 34.6 per cent.; iron, 30.5 per cent.; sulphur, 34.9 per cent.

The oxides are:

Cuprite: Luster, metallic. Color, red. Streak, red, shining. Brittle. Hardness, 3.5–4.0. Specific gravity, 5.85–6.15. Composition Cu_2O; copper, 88.8 per cent.; oxygen, 11.2 per cent.

Tenorite: Luster, metallic. Color, iron gray. Hardness, 3.0. Specific gravity, 6.25. Composition CuO; copper, 79.85 per cent.; oxygen, 20.15 per cent.

The carbonates are:

Malachite: Luster, vitreous. Color, green. Streak, green. Hardness, 3.5–4.0. Specific gravity, 3.7–4.01. Composition $Cu_2Co_4 + H_2O$; copper oxide, 71.7 per cent.; carbon dioxide, 19.9 per cent.; water, 8.2 per cent.

Azurite: Luster, vitreous. Color, blue. Streak, blue. Brittle. Hardness, 3.5–4.25. Specific gravity, 3.5–3.831. Composition $Cu_3C_2O_7 + H_2O$; copper oxides, 69.2 per cent.; carbon dioxide, 25.6 per cent.; water, 5.2 per cent.

Detection:

Copper may be detected by fusing the finely powdered ore on charcoal. A metallic button is obtained having malleability and red color. It may also be detected by dissolving the ore in nitric acid and adding ammonia in excess. A blue color indicates copper. If there is much iron present the blue coloration will not at first show unless the solution is filtered.

Qualitative determination:

Copper is precipitated in Group III with mercury, lead, bismuth, and cadmium. To make the separation, dissolve the ore in nitric acid and evaporate to dryness. Take up the water and a little nitric acid and filter. Add hydrochloric acid which will precipitate the first group (lead, silver, and mercury). Filter. Warm the filtrate and pass hydrogen sulphide through

the solution, which will precipitate the members of Groups II and III. Wash the precipitate and transfer it to a beaker containing yellow ammonium sulphide and heat without boiling for 10 minutes. Filter and wash with ammonium sulphide water. The residue on the filter will contain the members of Group III. Place the precipitate in a beaker and dissolve in strong nitric acid. Heat to boiling and filter. The filtrate may contain lead, besides copper, bismuth, and cadmium. Add sulphuric acid, filter, and wash. To the filtrate add ammonia and filter. If the filtrate is blue, copper is present. It may be separated from cadmium by adding to the solution acetic acid, hydrogen sulphide, and then a solution of potassium cyanide. The cadmium will come down as a yellow precipitate which may be separated from the solution containing the copper by filtration. To recover the copper, add hydrochloric acid to the filtrate, when the copper will be precipitated as sulphide free from impurities. This precipitate may be dried, mixed with soda, and fused on charcoal to a metallic copper button.

Quantitative estimation:

The fire assay is not used to any appreciable extent for copper. The method of estimating copper by titrating an ammoniacal solution of the metal with potassium cyanide is probably the simplest, and when properly conducted, is nearly accurate.

The standard solution of potassium cyanide is made by dissolving 60 grams of the salt in 1 liter of water. One cubic centimeter of this solution should correspond to 1 per cent. of copper, but as it changes with age it should be standardized at least once a week. To standardize, weigh out .2 gram of chemically pure copper foil. Place in a beaker and add 5 cubic centimeters of strong nitric acid, and boil until the fumes are expelled. Dilute to 200 cubic centimeters and add 10 cubic centimeters of strong ammonia and titrate until the blue color of the solution disappears. The weight of the copper taken divided by the number of cubic centimeters of solution taken will give the amount of metal which 1 cubic centimeter of the solution will precipitate.

To assay a copper ore, weigh 1 gram of the ore into a beaker or casserole and dissolve in nitric acid. Evaporate to dryness and take up with a little nitric acid and water, and add sulphuric acid. Evaporate to white fumes, cool, and add water. Boil, add about 50 cubic centimeters of cold water and filter into a beaker. Add aluminum foil and boil for about 5 minutes. Remove the aluminum foil, washing it free from particles of copper, stir the solution with a glass rod, giving it a rapid circular motion. Remove the rod and the copper will settle in the center of the beaker. Pour off the solution, fill the beaker with water, and repeat the stirring and decantation several times in order to get rid of all the acid. Add 5 cubic centimeters of strong nitric acid and boil until the brown fumes are driven off. Dilute to 200 cubic centimeters, add 10 cubic centimeters of strong ammonia and titrate until the blue color disappears. If silver is present the solution should be diluted to 100 cubic centimeters and a few drops of hydrochloric acid added and then filtered. Then dilute to 200 cubic centimeters and add 10 cubic centimeters of strong ammonia and titrate. One per cent. of silver, 292 ounces per ton, will increase the assay .29 per cent.

Metallurgy.—The metallurgy of copper has undergone many changes during the past 15 years. The trend of the development has been toward a simplification of the processes used until now, instead of having a dozen or more processes, with a multiplicity of by-products, the metal is won from the ore by three or four comparatively simple processes. Copper may be reduced from its ores to a metallic form containing 98 per cent. metal in three operations; viz., roasting the sulphide ore to drive off the sulphur, smelting the roasted ores in the blast furnace to form a high-grade matte, and bessemerizing the matte, which operation drives off the remaining sulphur and slags the iron of the matte, leaving the copper with only about 2 per cent. impurities. This metal is then refined by electrolysis, which deposits the copper almost chemically pure.

ZINC ORE NOTES

A steam shovel working on the Kansas City Southern Railway 4 miles southeast of Joplin, struck lead ore. The employes commenced to gather the ore and collect it in heaps alongside the track. According to reports one man had a pile of 800 pounds, which should bring him something like $18 from the lead buyer. The land has been sold for $5,000 to mine operators.

The Missouri Lead and Zinc Co., once the leading zinc producer in the Joplin district, is now abandoned, and the shafts are being filled. There were scores of grass-root bonanzas on the so-called "Bankers Land," which extended down 100 feet and more. A custom mill was operated and small lease holders availed themselves of this to clean their ore. Hand-jig outfits were in operation, but a certain proportion of the dirt required milling. After the surface had been worked and reworked and the development extended to deeper levels, the operators were troubled with acid water that corroded their pumps to such an extent that work was abandoned.

The one-time bonanza Alpha Mine, at Spring City, Mo., is producing 10-per-cent. zinc ore after being reopened by Adam Scott. For some time the mill has been working on tailing which became so lean that Mr. Scott determined to abandon that class of work for ore. Two shafts are available from which to get ore to the mill.

Among the mines resuming operation after a long period of idleness is the Symmes, west of Joplin, where a short time ago a large pile of ore was held for higher prices, and which was finally sold at from $44.50 to $45. The ore had accumulated from a long period of active work during the time of low prices. As the price of ore has advanced above $45, the company is again preparing to actively develop its large holdings in that part of the district.

The tailing piles of the old B. & H., A. & R., and the Omega mines on the Granby Mining and Smelting Co.'s land, at Chitwood, are to be reworked by Clarke Marshall, who is an extensive operator on the Granby property. Mills are now in operation at the three mines mentioned, but Mr. Marshall intends to handle the tailings from former operations first, as these promise to be richer. Mr. Marshall has also taken a lease on nine lots of the Granby Mining and Smelting Co.'s land at the head of Poor Man's Gulch, north of Chitwood, and has started a shaft on a drill hole which showed ore from 80 to 130 feet.

The Amalgamated Mining Co., operating in the Granby land at Smelter Hill, is hoisting from two shafts to supply its 300-ton mill, and from one shaft for the old Blackberry mill.

The Church Mining Co., operating in the north part of the Miami, Okla., camp, expects a heavy flow of water. Until the ore is struck the water will be light. The shaft was sunk in a soapstone bar and the ore will not be encountered until the 250-foot level is reached. This is the deepest development in the Miami camp.

The Miami Amalgamated Mining Co., operating on the Sullivan tract of the Miami Royalty Co.'s lease in the Miami, Okla., camp, is producing ore after a shut-down of several weeks due to heavy water which flooded the workings.

Zinc mining in Mexico, which in many camps is crippled, and in some districts suspended, by the prohibitive rate of impost upon shipments to United States smelters, appears recently to be taking on new life and borrowing hope from two sources, the likelihood of the erection of zinc smelters in the republic and the popular belief that there will be amendments to the tariff levied upon imports by the United States.

The Webb City, Mo., Lead Smelting Co. has erected a smelter at a cost of over $60,000 which, with the necessary buildings, covers an area of 10 acres, and the company anticipates having at least 75 men on the pay roll by February 1, 1911.

The Philadelphia Co., near Yellville, Ark., drove a tunnel 100 feet into the mountain several feet above the bottom of the

ore. This ore is being stoped and stacked for the mill, which was expected to be in readiness for work by January 1, 1911.

Pearl Brothers are constructing a 100-ton mill on the Granby Mining and Smelting Co.'s land on the east side of Lone Elm Road, Joplin. They have a lease on the tailing piles on 346 acres which it is expected will keep them operating several years.

Dirt containing less than 4-per-cent. ore is being taken from the Granby Mining and Smelting Co.'s mine on its own land in the Chitwood camp. The company had been led to believe that the dirt might prove richer than this, and it is with the hope of finding better stuff that drifts are being driven. Unless the ore body becomes more promising it is doubtful whether the Granby company will undertake to operate a mill on this property, as had been originally planned. The shaft is now down 170 feet, and two drifts have been started. One is now about 50 feet long running due north from the shaft and the other is of the same length, running due south from the shaft. A good big ore face is being worked and the quality of the zinc blende is said to be excellent. However, as other companies on the Granby land are working in much richer dirt it is possible the Granby company may undertake to open a better proposition before trying to produce concentrates.

THE LARGEST ORE CRUSHER

Written for Mines and Minerals

When in 1890 the gyratory or Gates rock crusher was making its bow to the public, one on the Erie Railroad near Southfields, N. Y., was considered a wonder, because it crushed granite-gneiss to railroad ballast at the rate of 150 tons per day. In 1900 the gyratory crusher was being adopted a little more generally both in the East and in the West. However, the Blake ore crusher was considered to be better adapted to ore crushing, although its capacity was much less. It is difficult to eradicate this impression from the minds of some conservative metal miners, although it is evident that where a rock is at all brittle the gyratory crusher is as suitable as the jaw crusher for this class of work.

In 1910, when the American Institute of Mining Engineers visited the Ancon quarry on the Panama Canal, they saw a No. 12 gyratory crusher which was crushing stone for concrete work at the rate of 400 tons an hour, and this was considered nothing extraordinary. In fact, gyratory crushers have come to be looked upon as machines whose capacity is only regulated by the demand of the purchaser, and the limit of their practicable size has not been determined.

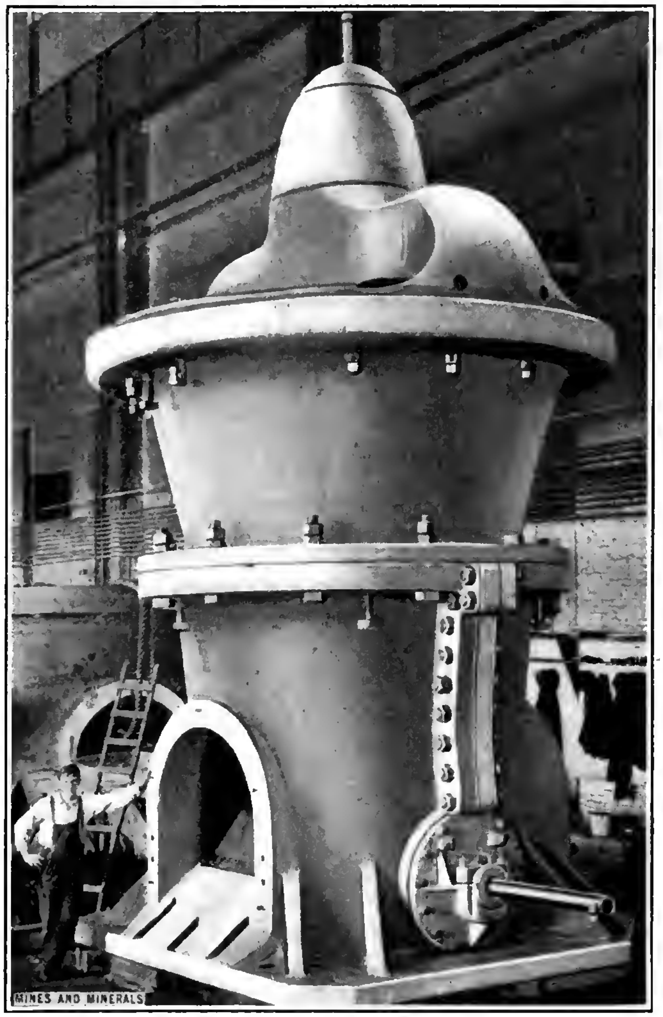


Fig. 1. Gyratory Crusher for Iron Ore

In 1901 the Biwabik Iron Mining Co., working on the Mesabi Range, Minn., installed the No. 21 gyratory crusher shown in Fig. 1. This machine was the largest and most powerful that had ever been built up to that time, and naturally attracted considerable attention. Although it has been in practically continuous service, the developments of the company have increased to such proportions that a new crusher, again larger than ever before constructed, was ordered. The new one that is shown partly loaded on cars in Fig. 2, when erected, stands 28 feet high, weighs 250 tons, and has two openings 4 feet by 9 feet capable of receiving 10-ton pieces of ore. These the crusher reduces to the desired size without exhibiting undue excitement over the strenuous work. The capacity of this huge affair is 40 broad-gauge cars of crushed ore each hour, or practically 800 tons per hour.

The Biwabik iron mines are worked in open-cut benches and after the ore is broken from the bench it rolls down to the steam shovel which loads it into cars. In order to break the ore from the benches it is blasted, and as a result falls in all sizes. Some of the pieces are so large they cannot be handled or used in the iron furnace, so that it is necessary to break them to suitable sizes. It is for this purpose that the crusher is to be used, and although it is to work on iron ore, the ore is hard and could not be broken as economically by any other means.

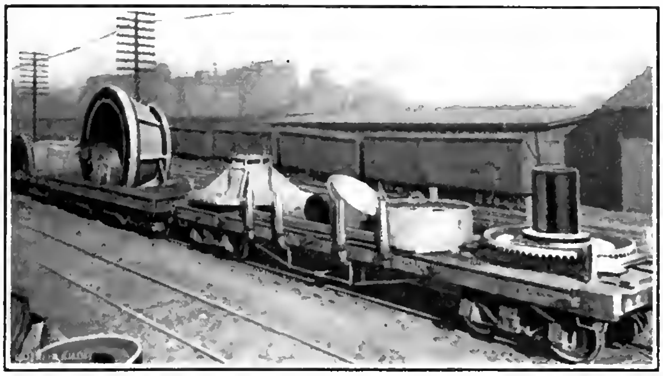


Fig. 2. Crusher Loaded on Cars

According to a commercial letter from Mr. C. H. Sherrill, United States Minister at Buenos Aires, about one-fifth of the world's wolfram is mined in Argentina. Wolfram, or wolframite, is a tungstate of iron and manganese. It is the ore from which the metal tungsten is usually obtained. One mining company, the Hansa Sociedad de Minas, produces six-sevenths of the entire Argentine output. It mines 60 to 100 tons of ore a month, ranging from 65 to 75 per cent. pure wolfram. The price of wolfram is about $1 a kilo (2.2 pounds) of pure metal. The amount of metal in a ton is estimated by the government chemical assayer in Hamburg, Germany, where the ore is sent.

THE CLANCY CYANIDE PROCESS*

The Clancy process is particularly concerned with the treatment of refractory gold and silver-bearing minerals. In this paper Mr. Clancy recounts his experience and his deductions while making his experiments leading up to the final stage employed in his process. From a chemical and metallurgical point the paper *in toto* is so valuable that it would be well for those interested in the theory of the cyanide process to send for it, as only the practical work can be given here on account of the length of the paper.

Chemistry of the Process. Action of the Solutions—Method of Operation and Costs

When a solution of calcium cyanamide is mixed with a solution of alkaline ferrocyanide and allowed to stand a few hours, it becomes an active solvent for gold, and when applied to the treatment of ores amenable to cyanide solutions, gives equivalent results, even in very dilute solutions. The process, Mr. Clancy says "can be used on cyanide tailing that contains a large proportion of Prussian blue $KFe(CN)_6$, which, when treated with alkali becomes potassium ferrocyanide $K_4Fe(CN)_6+3H_2O$ and potassium ferricyanide, $K_3Fe(CN)_6$ or soluble prussiates."

This furnishes an inexpensive method of treating material containing gold or silver, that could not be treated economically by straight cyaniding. For example, subject one part of the gold-bearing material for 8 hours or more to two parts of solution made as follows: Water, 2,000 pounds; calcium cyanamide, 1 pound; alkaline ferrocyanide, 1 pound; lime, 1 pound. When the material is tailing from a former cyanide process and contains Prussian blue, etc., one part of tailing used with two to three parts of the following solution will furnish results equivalent to straight cyaniding: Water, 2,000 pounds; cyanamide, 1 pound; lime, from 1 to 5 pounds, depending on the acidity of the tailing. If the mixtures of the gold-bearing material and solution are subjected to electrolysis the solution of the gold is rapid.

Calcium cyanamide mixed with alkaline sulphocyanide, electrolyzed, immediately becomes cyanide and dissolves gold rapidly.

To show the action of another amide compound, for example—guanidine carbonate $C_3H_{12}N_6O_3$ mixed with potassium cyanate, in conjunction with electrolysis of the solution, dissolves gold leaf in less than 10 minutes.

In the midst of this work on cyanogen-bearing materials I made a solution of calcium cyanamide, adding a solution of iodine to see if this combination would dissolve tellurium, and was rewarded by finding it to be an excellent solvent. I next tried gold leaf on the same combination and had the satisfaction of seeing the gold go into solution in a short time—a scum of calcium carbonate forms on the surface preventing of more rapid dissolution. From this I deduced that the addition of iodine formed cyanamidogen iodide. I am unable to find any reference to this compound in chemical literature, and take the liberty of calling it by the above name, and further presume the formula to be CN_2I_2. I next applied cyanogen iodide ICN, in a solution of calcium cyanamide to telluride of gold, which at once dissolved, showing that the reaction was probably due to cyanamidogen iodide and I again presume the reaction to be in accordance with the formula

$$2ICN+CaCN_2=Ca(CN)+I_2CN_2$$

I will now describe the general method of using the Clancy process on the working scale. The ore is crushed by stamps, rolls, ball mills, or any other efficient grinding mills. The degree of fineness necessary for the process being about 100 mesh, it is probably most economical to use tube mills. The ore is crushed in cyanide solution containing calcium cyanamide, sulphocyanide, and the halogen salts; it is, therefore, under the influence of straight cyanide treatment practically after leaving the rock breakers until it reaches the agitation tank. In the agitation tank it meets with electrolysis, which acts upon the cyanide containing the additional chemicals in solution, as already described. Calcium cyanamide of commerce comes as a black powder and is from 58 to 65 per cent. soluble in water. It is necessary, therefore, to dissolve the cyanamide in a separate tank and filter from its insoluble residue. A small air agitating tank is admirably suited for this purpose, and at the same time will act as a cyanamide storage tank. The cyanamide solution may be made as saturated as desired and the calculated quantity drawn off when required, and added to the cyanogen-bearing solution. The treatment solution is made up to 2,000 pounds of water containing 1 pound of cyanide, 2 pounds of alkaline sulphocyanide, 2 pounds of calcium cyanamide, and $\frac{1}{2}$ pound alkaline iodine. If using after the rock breakers, say, for example, crushing rolls, the product leaving the rolls about 12 mesh is fed into the tube mill and converted into a pulp, by feeding the mill with the treatment solution and ore in the proportion of one part of ore to one part of solution. The discharged pulp after separation of the oversize is transferred to the agitation tank to undergo electrical treatment. If it is thought necessary to remove the sulphides before, after, or during the treatment, the following method presents an ideal scheme. When solution containing finely divided ore in suspension is contained in the well-known cone-shaped tank and agitated in the proportion of two of solution to one of ore, or in the proportion of three of solution to one of ore—if the agitation is stopped for a few minutes, the finely divided sulphide particles settle to the bottom of the cone, and by simply opening the discharge valve at the cone apex, the sulphide may be completely drawn off together with a small proportion of the non-sulphide pulp. This sulphide product may be run over blankets or similar contrivances and the finely divided concentrate collected, the excess of pulp solution being returned to the agitation tank for treatment. Here is a means to eliminate the use of concentrating tables and obtain a product of very high value in a very small bulk. Again, the pulp being in a very finely divided state the pyrite or sulphide portion is not accompanied with quartz or gangue—therefore, a clean high-grade concentrate—a result not capable of accomplishment by the use of concentrating tables, without the employment of a large quantity of solution with its attendant expenses.

The pulp now in the agitating tank carries the correct alkalinity, this being previously established in the tube mill by adding lime so as to contain from $\frac{1}{10}$ to $\frac{2}{10}$ of a pound "protective" alkalinity per ton of solution.

The conductivity of the pulp is adjusted by adding common salt until the required voltage is obtained, 20 pounds of salt per ton of solution, as already stated, will invariably decrease the resistance of the pulp so that the voltmeter will register from 5 to 6 volts. In the majority of cases a current of about 50 amperes per ton of ore is adequate. It will be easily seen from this that the cost for electrical energy is not prohibitive. With iron oxide electrodes it is possible to obtain a current density considerably over 50 amperes per square foot of anode surface, so that one electrode 3 feet in length by 3 inches in diameter will be sufficient for the treatment of from 3 to 4 tons of ore; in other words, approximately 30 of these iron oxide electrodes would be required for the treatment of 100 tons of ore per day. If the treatment tank is constructed of iron it may be used as the cathode. This arrangement would of course decrease considerably the cost of installation. The electric generator is the chief item of cost, a low-voltage generator, say a 10-volt machine capable of giving the necessary amperage, can be obtained at any of the electrical warehouses. The process may be applied to any existing fine-grinding plant provided with agitating tanks; all that is necessary is simply to introduce the electrodes into the circulating ore pulp containing the necessary chemicals and switch on the current. It is essential in every

*Abstract from a paper read by John C. Clancy before the American Electro-Chemical Society in New York City, entitled "The Clancy Electro-Chemical Cyanide Process."

case to maintain the protective alkalinity at about $\frac{1}{10}$ of a pound alkali per ton of solution so as to allow of the formation of cyanogen iodide and cyanamidogen iodide. About 8 hours treatment under electrolysis usually is sufficient to obtain the necessary extraction.

After treatment with the current, the pulp solution is brought up to about 1 pound per ton of protective alkalinity by adding caustic soda, and the cyanide contents regenerated up to about $\frac{5}{10}$ to $\frac{6}{10}$ of a pound cyanide per ton of solution. The regeneration of the cyanide is then accomplished by giving the pulp about 2 hours more current. It will be understood that the reason for adding the extra alkali is that cyanide regeneration cannot take place in the presence of a halogen compound unless the solution containing sulphocyanides and cyanamide is made alkaline.

It will be seen that the value of the process depends upon the recovery of the halogen compound. While in the examples a proportion of two parts of solution to one part of ore was used, a proportion of three parts of solution to one part of ore may be used with advantage, that is to say, by using three parts of solution to one part ore, a much smaller amount of alkaline haloid may be used per ton of solution; thus giving the same ratio of haloid salt per ton of ore as in the two of solution to one of ore pulp, and consequently a less proportion of soluble haloid to be displaced by the water wash in the final slime cake.

In this incomplete description of the process the use of chemicals and current have been described, but no mention of costs has been referred to. I will, therefore, take the following to represent the typical working solution: 2,000 pounds of water containing 1 pound of cyanide, 2 pounds of sulphocyanide, 2 pounds of calcium cyanamide, and $\frac{1}{4}$ pound alkaline iodide. This appears a formidable mixture when looked at cursorily, but on analysis it does not work out beyond the limits of economic treatment. For example:

	Cents
1 pound of cyanide	18
2 pounds of cyanamide	6
2 pounds of alkaline sulphocyanide	12
$\frac{1}{4}$ pound of alkaline iodide	35
Total	71

This does not represent the total cost of 1 ton of solution, for, notwithstanding the effect of electrolysis, practically all the haloid salts previously added, together with the sulphocyanide will be found unimpaired at the end of the operation, the cyanide and cyanamide alone suffering decomposition. It is clear, therefore, that no matter what the proportion of solution to ore, only the consumption of cyanide and cyanamide per ton of ore is to be taken into account. The amount of cyanide consumption on the ore in presence of cyanamide works out at about 1 pound of cyanide per ton of ore treated. This cyanide is regenerated at the expense of 3 cents for cyanamide and at the very outside 3 cents for current (figuring current at 1 cent per kilowatt hour), making a total cost of 6 cents per ton of ore.

Added to the above cost is the cost of the electrical energy necessary for the electrolysis of the ore pulp. The cost of electrical energy for this purpose works out at about 10 cents per ton of ore treated, this added to the cost for cyanogen-bearing material and regeneration would make a total of 16 cents per ton of ore. These figures would represent the total cost provided that all the solutions were recovered without mechanical loss. From this it is evident that the recovery of the solutions for reuse is a matter of vital importance to this process. The mechanical recovery of the solutions is, therefore, entirely dependent upon the efficiency of the filter employed.

This filter process is too well known to describe minutely, suffice it to say that when the cake has been washed with barren solutions, the soluble salts in the solution saturating the cake at this juncture, may be completely water washed by giving it the requisite amount of water for displacement. The necessary amount of water for displacement is readily observed by the diminution of the water level in the wash tank, further, this can be adjusted to a nicety at the will of the operator. In a properly constructed filter the loss of solution under proper manipulation should not exceed 10 per cent. of the total solution.

It should be understood that the above figures are based on the treatment of refractory ores and that they would in all probability be much reduced when dealing with non-refractory ores.

NORTHWESTERN ORES

By August Wolf

Veteran prospectors predict that an important miners' rush will follow the passage of a bill at this session of Congress, opening to entry the mineral lands of the Spokane Indian reservation. United States Senator Wesley L. Jones, of Washington, who introduced the bill, advises that its adoption seems assured. More than 25 syndicates have been formed in Spokane in anticipation of the passage of the bill, and as soon as it becomes a law several hundred prospectors will be rushed into the district to stake claims. Large deposits of silver, copper, and tungsten have been traced across the boundaries of the reserve, part of which was opened to homesteaders last spring, and at one time development work was under way on 70 claims, when the prospectors were ejected by agents in the employ of the United States Government. Senator Jones' bill provided that "from and after the passage of this act any lands of the Spokane Indian reservation, classified and reserved as timber lands under any act of Congress heretofore passed, shall be opened to exploration, location, occupation, and purchase under the mining laws of the United States." L. K. Armstrong, secretary of the Western Conservation Association, who interested Senator Jones in the new measure, declares that the timber and mineral provisions were purposely left out of the original act opening the reservation, adding that, while the responsibility has not been fixed, he believes it was brought about by some one friendly to the Indian Bureau.

Sixty-five per cent. interest in the Standard Mine, a silver-lead property in the Dominion of Canada, has been acquired by Patrick Clark, of Spokane, on the basis of $2,500,000 cash. The property, located near Silverton, B. C., was originally bonded for $30,000 and financed for a half interest in the mine. Thus one man became a millionaire without the investment of a penny. The bonanza ore shoot was encountered about 6 months ago. The No. 5 tunnel level, in which the development of the ore body is in progress, is about 400 feet from the surface, and has been driven 175 feet on the ore shoot. When first found the ore shoot was 12 feet thick, where cross-cut; at the present face, it is 40 feet wide with only one wall in sight. From this ore body 36 samples averaged 38 per cent. lead and 51 ounces of silver to the ton. The cleanest of the ore taken out while this level was being driven was sent to the smelter at Trail, B. C., and averaged $62 a ton net.

The Bunker Hill & Sullivan Mine at Wardner, Idaho, ranks fourth among the dividend-paying gold, silver, and lead mines of America for the first 11 months of 1910, according to government statistics. The 60 largest dividend-paying mines in the United States for the current year disbursed a total of $17,783,631. The four largest profit-sharing mines are given as the Goldfield Consolidated, $7,118,196; Tonopah, $1,500,000; Homestake, $981,000; and the Bunker Hill, $925,350. According to Stanley A. Eaton, manager of the Bunker Hill & Sullivan mines, much of the lead now being produced in the United States is mined at a very small profit, and this is especially true at the mines in Missouri. It is out of question for American lead mines to compete with those of Spain, a country which produces a tonnage equal to that of the United States and consumes hardly any lead because of the poverty of the people. Competition with Canadian mines is not to be feared, as con-

ditions under which they are operated are similar to our own, and there might be no objection to reciprocity with Canada in lead and lead products. But for the uncertainty of the tariff, the lead market would be in a much stronger position than that which obtains for copper, chiefly because no great lead producers have been developed in the United States in years, while enormous deposits of copper ore, the product of which can be profitably marketed in New York as low as 8 cents a pound, have been opened recently. At the Bunker Hill the old mill is running at its proper capacity, which is about 800 tons a day, instead of overcrowding it as sometimes in the past. The first unit of the new mill is in operation and is treating 550 tons a day. The second unit is entirely enclosed and ready for the installation of machinery. It will be put into operation early in the spring, when the present production of 40,000 tons a month will be increased to 55,000 tons or more. An electric hoist is to be installed in the Kellogg tunnel, the present lowest working being 600 feet below that level. The plant will be of a type similar to that installed by the Hecla company at Burke, which has proved satisfactory.

JOPLIN ZINC AND LEAD PRODUCTION

Written for Mines and Minerals, by L. L. Wittich

The total valuation of zinc and lead ores shipped from the Missouri-Kansas-Oklahoma district in 1910 was $14,225,015, compared to $14,615,048 in 1909, $11,069,969 in 1908, and $15,419,727 in 1907, the record year of the history of the district.

The zinc ore production, including both blende and calamine, was 296,670 tons, compared to 301,206 tons in 1909, 259,598 tons in 1908, and 286,587 tons in 1907.

The lead ore production was 44,415 tons, compared to 44,186 tons in 1909, 39,119 tons in 1908, and 42,034 tons in 1907.

The accompanying table shows shipments, with their values, of zinc and lead ores from the various cities, towns, and camps of the Missouri-Kansas-Oklahoma district for the year 1910:

JOPLIN ZINC AND LEAD PRODUCTION, 1910

Camps	Blende		Calamine		Lead		Totals Values
	Pounds	Value	Pounds	Value	Pounds	Value	
Webb City–Carterville, Mo.	215,713,555	$ 4,515,466			47,716,005	$1,258,799	$ 5,824,265
Joplin, Mo.	98,758,190	2,075,844	3,573,690	$ 42,744	14,426,465	379,608	2,498,196
Duenweg, Mo.	37,999,160	780,960	6,413,180	85,543	5,761,160	149,293	1,015,796
Galena, Kans.	38,589,230	801,557	287,220	3,364	4,136,710	108,729	913,650
Alba–Neck City, Mo.	37,168,990	805,190	87,040	1,076	403,150	10,439	618,705
Miami, Okla.	20,311,390	332,250			6,775,025	176,489	508,739
Granby, Mo.	10,692,790	194,424	19,217,190	250,474	826,000	19,739	464,637
Oronogo, Mo.	16,828,545	347,059			2,576,050	70,828	417,887
Spring City, Mo.	4,900,480	99,436	8,375,000	106,068	3,818,980	98,617	304,121
Badger, Kans.	12,982,930	273,589			925,630	25,673	299,362
Aurora, Mo.	8,368,820	171,379	6,615,475	79,642	454,475	11,530	262,551
Cartbage, Mo.	11,111,170	238,611	349,450	4,288	36,620	866	243,765
Quapaw, Okla.	7,937,720	167,068			443,420	11,109	178,177
Carl Junction, Mo.	6,680,480	143,210			48,370	1,151	144,361
Zincite, Mo.	6,314,860	133,829			242,880	6,212	140,041
Sarcoxie, Mo.	4,721,820	97,256	2,500,065	32,370	29,770	817	130,443
Cave Springs, Mo.	3,511,500	73,124			18,370	408	73,532
Stotts City, Mo.	1,678,855	35,444			30,320	780	36,224
Reeds, Mo.	681,490	13,781	173,670	2,146			15,927
Wentworth, Mo.	166,540	3,577	226,080	2,720			6,297
Seneca, Mo.			108,250	1,331	124,650	3,383	4,714
Greenfield, Mo.			273,710	3,325	37,000	962	4,287
Peoria, Okla.			111,320	1,228			1,228
Springfield, Mo.	10,510	210					210
Totals	545,129,025	$11,303,264	48,211,340	$616,319	88,831,050	$2,335,432	$14,255,015
Average values per ton, 1910		$41.47		$25.56		$52.56	
Average values per ton, 1909		$42.29		$24.38		$54.81	

Fluorspar, as mined near Jamestown, Colo., contains 70 to 85 per cent. calcium fluoride. Veins less than 1 foot thick can hardly be worked profitably. Few veins exceed 6 feet in thickness. The principal handicap to the industry is the absence of railways. Wagon hauls of 13 miles are not uncommon.

PROSPECTING ADVICE

Mr. Stanley A. Eaton, manager of the Bunker Hill and Sullivan Mine, recently said that if he were to go prospecting he would devote his attention to the older camps, adding that the operator who rushes into a new district takes double chances, while in approved districts he has to take chances only on the property. Continuing, he said:

"What has made me a convert to this idea is the way the old mining districts of the United States are coming to the front with new discoveries of the first magnitude. Recall what has been done of late years in Nevada, the oldest mining state in the Union, which was almost abandoned for a generation. During the last year Leadville has surprised the world once more with ore discoveries of almost equal importance to anything in its wonderful history. This time large bodies of zinc ore have been found and are being worked.

"Coming nearer home, we find that the old camp of Butte, is opening up great and rich ore bodies in many unexpected places, while such districts as the Republic camp in Washington and Sheep Creek and Slocan, in British Columbia, are proving that the neglect they experienced at the hands of mining men the last decade was due to lack of knowledge and appreciation of their mineral resources.

"The Wardner district, though the oldest of the Coeur d'Alenes, seems to be holding out better than any other part of that field. The Wardner Hill has produced to date more than $80,000,000 that can be accounted for, and probably a good deal more than that, because many of the records of the early days, which were poorly kept at best, have been lost.

"Such mines as the Tyler, Stemwinder, and Sierra Nevada were big producers in their day, but that is about all known about them now.

"The Bunker Hill & Sullivan Mine alone is known to have produced $60,000,000, and in view of the fact that the older records are missing, it may have produced considerably more than that amount.

"The early ideas regarding ore occurrences in the Wardner camp have had to be revised as the result of the last 2 years' developments. The statement made some time ago by Robert N. Bell, former state mining inspector of Idaho, regarding the Frances property, was just about right. In that case deep development proved a great ore shoot where, according to the old ideas, it could not possibly occur. There are also other instances."

LEADVILLE, COLO., ZINC DEPOSITS

*Written for Mines and Minerals, by Howard E. Burton**

Recent Discoveries of Zinc Carbonate That Had Been Passed by When the Mines Were First Worked

Zinc carbonate was unknown in the Leadville district of Colorado until a few months ago. Zinc sulphide has been shipped from Leadville in large quantities for many years, but zinc in its oxidized forms was not thought of until recently.

Calamine was first found in the Monarch district, above Salida, in the Madonna Mine, which is still shipping this kind of ore.

It was next heard from two years ago in the Horseshoe district on the divide between Lake and Park counties. James B. McDonald, who was working the Hilltop and Last Chance mines under lease, when examining some old stopes and drifts, noticed a brown, heavy material that aroused his interest. He took liberal samples, not knowing what it was, and after having it assayed found it contained 50 per cent. zinc. Shipments were at once made to the different smelters, and the returns held up to the assays. The properties are still shipping this class of ore.

The next discovery was made in Leadville, in the famous Robert E. Lee Mine, on Fryer Hill, by W. E. Jones, a leaser. He opened a large body of carbonate of zinc, but the material was very low grade and valueless. That was a little over a year ago. This class of ore did not appeal to the miners of the district, consequently no attempt was made to discover it. The general opinion held was that the zinc had leached out of the original ore bodies and been precipitated below only in the form of sulphide, and it was thought no carbonate of zinc worth mining would be found in the district. Very few men in Leadville mines were familiar with carbonate of zinc ore, and as a result it was not recognized even when seen.

The next property to open up zinc carbonate was the May Queen in Leadville. The writer became identified with Messrs. H. K. White and Alfred Thielen in a lease on the Hayden shaft of the May Queen. This was the property first to find and ship high-grade calamine, and shipments have been continuous since its discovery. The shipments attracted little attention at first, but soon the possibilities of finding zinc in the other old mines of the camp became apparent.

S. D. Nicholson, manager of the Western Mining Co., which has always been a heavy shipper of zinc sulphides, started an investigation of the old workings of the Maid of Erin, Henriette, Waterloo, Adams, Wolftone, Mahala, Big Chief, and other mines, and large bodies of smithsonite were found in all of them, with an average value of 37½ per cent. One drift in the Maid of Erin alone was driven through this class of ore for over 750 feet. Following the discovery in the large producers of the camp the miners got busy, with the result that calamine has now been found from Fryer Hill south to Weston Pass. The ore is found from the parting quartzite to the gray porphyry, and also below the parting quartzite wherever the formation is oxidized.

Mr. E. W. Keith, of the Empire Zinc Co., who has examined the principal zinc deposits of this country and Mexico, says that the Leadville deposits are the largest he has ever seen.

Prof. S. F. Emmons, chief of the United States Geological Survey, 1880, writing to the editor of the Leadville *Herald-Democrat*, says: "It is in the normal order of development of a mining district that when exploitation has gone so deep that increasing expenses and decreasing yield have reduced the margin of profit, exploitation turns again to upper levels in search of ore that may not have paid to work when first opened, but which under present changed conditions may be profitably extracted. At the time of my first study of the Leadville district in 1880, I was much puzzled to know what had become of the zinc, since by analogy with similar deposits in the Ten Mile district, I reasoned that the original sulphides of Leadville must have contained much zinc as well as lead. I only succeeded in finding a few needle-like crystals in limestone joints that resembled gypsum, but proved on chemical examination to be 'silicate of zinc.' I assumed then that owing to the inferior solubility of the zinc sulphate, the oxidation products of that metal had been carried much further than those of lead before being transformed into the now stable carbonate and had possibly been entirely removed in the run off. Blow's observation that on Iron Hill secondary zinc blende had accumulated in the upper part of the sulphate zone seemed to account for some of the missing zinc, and from the accounts published by you it is evident that much of it has accumulated as calamine in the zone of change from sulphide to oxide. Though I have particularly desired to study the zinc of Leadville I have never been able to, because in 1880 mine workings had not reached it and when I next visited the district, 1890, they had gone beyond it and, owing to the soft nature of the ground in that zone, the drifts leading to it were for the most part caved and inaccessible.

"It certainly seems rather strange that those in charge of mines, when the zinc was exploited, did not notice such bodies of calamine as you describe, but it must be borne in mind that calamine is generally a white-brown, earthy looking material which would not attract attention unless especially sought for, and that it was pay ore rather than material of only mineralogical interest that they were seeking, and at that time zinciferous ores were a particularly undesirable product."

The original ore bodies of Leadville were undoubtedly sulphides. These bodies became oxidized through the action of percolating waters. The zinc was taken into solution in the form of sulphate. This solution was carried down until it came to the impervious quartzite formation and was precipitated as the insoluble carbonate by the limestone formation. In other words it is a replacement of the limestone by the zinc. The same process can be seen in operation on a small scale today in the old workings of some of the mines. Where a drift has cut a body of sulphide ore the water percolating through the zinc takes it in solution as sulphate. This solution when it drops into the drift evaporates and leaves big stalactites of zinc sulphate. If this solution had gone a little further and remained in the limestone it would have been precipitated as insoluble zinc carbonate.

There are several reasons why the mine manager did not discover calamine years ago. When the large bodies of this material were opened he was not looking for zinc in any form, as what had been found had so penalized the lead and iron with which it was associated that these large bodies could not be shipped at a profit. The miner was hunting for other ore bodies and when the carbonate of zinc was encountered he took it for "waste" and allowed it to remain standing. When the time came so that zinc could be mined at a profit he was hunting for zinc sulphides which everybody recognized. Calamine has been recognized now, new mines have been discovered within old ones and the discovery has again brought to the front the town of many surprises and again astonished the world by the immensity of the ore bodies of carbonate of zinc now in sight.

At present the only grade of this ore being shipped is what is known as high-grade material, that will run 30 to 40 per cent. In opening the ore bodies of this grade, mountains of low-grade matter that carries 15 per cent. or more have been encountered, and when a process has been discovered for treating this low-grade material at a profit, Leadville will be in a position to supply the world for years to come. The School of Mines at Golden, has taken up this problem in the new experimental ore-testing plant, and it will probably be solved by the use of fire concentration or some chemical process, and not by mechanical separation, because the smallest particle of ore that one can take is still a mixture of zinc, iron, etc. People in Leadville are very much interested in this problem and any one that can solve it will find a rich field to operate here.

* Assayer and chemist, Leadville, Colo.

DEL CARMEN ZINC MINE

Written for Mines and Minerals

The Del Carmen Mining Co., at Boquillas, Coahuila, Mex., uses three different systems of surface haulage in moving its ore to the Southern Pacific Railroad station, at Marathon, Tex.

A Transportation System Composed of Mules and Wagons, An Aerial Tramway, and Traction Engines

The company owns a large zinc deposit which is high grade, but about 100 miles from the nearest railroad station. Between the mine and the shipping point the country varies in topography; stretches being rough and precipitous, others arid, while the Rio Grande has to be crossed.

As no gold or silver exists in the ore, the cost of shipment by the system shown in Fig. 1 was too expensive, and after deliberation it was decided to divide the transportation into three stages.

Fig. 1. Burros Packing Water

The first stage of 5 miles commences at the mine and ends at the storage bins of an aerial tramway. The ore is hauled in wagons drawn by mules over a fairly good road.

The second division of the transportation system is an aerial tramway 6 miles long. It begins at a point in Mexico 2¼ miles from the International boundary line, crosses the Rio Grande, and extends 3¾ miles in Texas. Owing to the tramway being an international means of transportation, it is necessary for the Del Carmen Mining Co. to establish ports of entry and custom houses at the loading terminal in Mexico and discharge terminal in Texas, as there is a duty on zinc ore. The ore is weighed by Mexican customs officials at the loading station, shown in Fig. 2, and is then dumped into storage bins. In the discharge terminal, shown in Fig. 3, which is also the United States customs house, the inspectors weigh the ore in the tramway buckets by means of a tramway dial scale. This is a clock dial on which are marked the graduations in pounds from zero up to whatever weight may be required for the load. By means of a pointer the weight of each bucket as it passes over the scale is indicated on the dial. By setting the pointer to the left of zero to take care of the weight of the bucket the pointer will indicate the exact net weight of ore in each bucket. There is a short section of the terminal rail connected with the fulcrums of the scale, and when a bucket passes over this section the weight is indicated by a series of levers on the dial. The weight is not automatically recorded; the man in charge of the terminal records the weight as it is indicated on the dial. Surrounding the discharge terminal is a wire fence with a gate to establish a boundary and safeguard the custom house officials in their duties.

The wire-rope or aerial tramways being independent of differences in elevation can be made to carry material either up or down hill, across deep valleys or wide streams, and deliver it at destinations inaccessible to other transportation systems. Aerial tramways are constructed in favorable situations to be worked as gravity planes; in other situations where the topography varies they are designed to work partly by gravity and partly by power; and in flat countries power is required to move the loads over the system. The Del Carmen Co.'s tramway requires power to assist gravity in moving the loaded buckets, and this is obtained from a 45-horsepower gas engine.

In many instances where water is scarce and fuel for boilers is expensive suction gas producers are installed to supply gas engines with gas. Charcoal for the gas producer at the Del Carmen loading terminal is burned in pits situated near the plant, any available wood being used for the purpose.

The tramway was constructed by A. Leschen & Sons Rope Co., of St. Louis, Mo., who also kindly permit the use of the illustrations for this article. It consists of two track ropes and two traction ropes, supported on towers spaced at distances to conform to the contour of the ground. The track rope for the loaded buckets is 1-inch diameter crucible steel, flattened-strand rope, well adapted to this purpose owing to its strength and to the ease with which the carriage wheels run over it without excessive wear. Empty buckets do not require the same strength of rope as the loaded buckets, owing to the decrease in tension stress produced by the load, therefore, the

Fig. 2. Bucket Leaving Mexican Loading Station

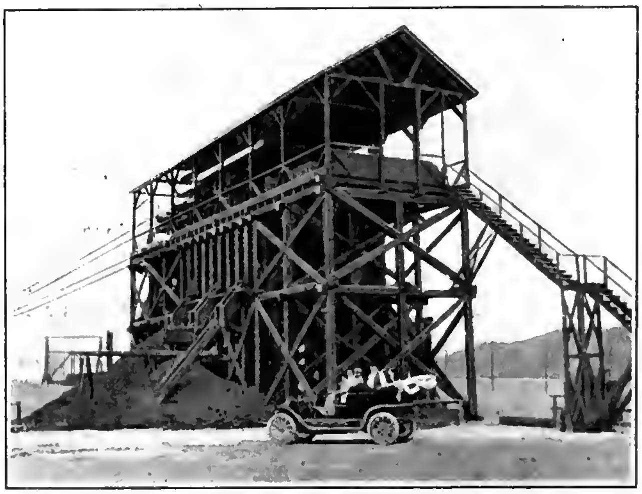


Fig. 3. Discharge at American Terminal

return rope on this system is ¾-inch diameter. Movable or grip ropes are the well-known ⅝-inch in diameter flexible Hercules hoisting rope, having six strands with each strand made of 19 wires.

In Fig. 2 a loaded bucket is shown at the commencement of its 6-mile journey.

When a wire rope is suspended between towers of an aerial tramway it will sag from its own weight and form a catenary

Fig. 4. Loaded Bucket Passing Over Tower Saddle

curve. At the center of the span there will be a horizontal distance where there is tension due to the weight of the rope on each side of the center. If the cable is loaded, the tension will be increased by the tension due to the load and the deflection it sets up. The tension at the point of greatest deflection where the track ropes become horizontal equals the load between the points at which a horizontal line from the top of the lower tower intersects the cable, multiplied by the distance between these points of intersection, divided by eight times the deflection below this line. The tension at either tower equals the square root of the sum of the squares of the horizontal tension so obtained and of the load between the tower and the point of greatest deflection. When the towers are on the same level this becomes:

$$\text{Tension of span} = \frac{\text{total load} \times \text{span}}{\text{deflection of cable} \times 8}$$

$$\text{Tension at towers} = \sqrt{(\text{tension at center})^2 + \left(\frac{\text{total load}}{2}\right)^2}$$

$$\text{Length catenary span} = \frac{\text{span} + (\text{deflection of cable})^2 \times 8}{\text{span} \times 3}$$

To obtain the total deflection, that of the rope alone must be added to that of the load alone. When towers are at different elevations the formula becomes complicated and lengthy, for which reason a graphical method is adopted which gives results of sufficient accuracy for the requirements.*

Long wire-rope tramways are divided into divisions in order to relieve tension on the track ropes and to compensate for variations in length due to stretching or temperature changes. For this purpose an anchor station is arranged at one end of a division where the track ropes are securely fastened in the ground; and at the other end a tension station is arranged where the ends of the track ropes have weighted boxes attached to them. Each track rope of the Del Carmen tramroad is divided into five sections with a tension station at one end, while the other end is fastened at an anchor station. Wire ropes may be obtained in any length within reasonable limits; for instance, it is possible to produce a ⅞-inch cable as long as 2,000 feet, but owing to the difficulties in transporting such heavy ropes it is customary to use shorter lengths and join them with suitable couplings.

The details of tower construction are shown in Fig. 4. The upper crosspiece has a grooved iron saddle at each end, in which the track ropes rest, and which are of such diameter that the carriages have no trouble in passing over them. The next lower crosspiece has a sheave wheel at each end, in the grooves of which the traction ropes run. This arrangement supports the ropes, takes up tension, thus relieving the track ropes, and further allows the traction rope to travel without excessive friction. In the figure a loaded bucket is shown passing over a tower saddle, with the clip on the bucket arms passing over the traction rope pulley. Universal clips on the bucket arms furnish positive grips on the traction rope, obviating all danger of the buckets becoming detached and running away on an incline.

Tower spacing, as has been stated, depends on the contour of the ground, and while horizontal tension is the same on each tower, yet the vertical tension will be more at the higher of two towers placed at different elevations. The vertical components of the catenary curve tend to promote stability by pressing down on the towers while the horizontal components tend to overturn them. The longest span on this tramway is 1,500 feet; the next longest is the span over the Rio Grande, which is 1,300 feet. Just beyond the first tower shown on the Mexican side of the river in Fig. 5, there is a tension station fitted with a water tank. At this station cylindrical tanks holding 40 gallons of water each, suspended from carriages the same as the ore buckets, are filled. The country through which the tramway extends is an arid desert, and as water is needed at both terminals, 15 water carriers pass over the line from a point near the Rio Grande, where there is a pump station erected for the purpose of filling the water tank.

Fig. 5. Cable Across the Rio Grande

When a water carrier reaches the water station an attendant switches the carriage from the track rope where it can be filled without interfering with the working of the tramway. All water carriers are supplied with friction grips, and can be attached and detached between buckets readily.

On the Del Carmen tramway, 90 steel buckets having a capacity of 600 pounds each, travel at a speed of 300 feet per

*See Mines and Minerals, Vol. 24, p. 423.

minute, thus requiring 3.5 hours to make a round trip. About 25 buckets are loaded and discharged every hour, and if necessary, their number may be increased. The buckets are automatically loaded without stopping for the purpose, by means of the hopper loader shown in Fig. 6. The operator standing on the platform to the rear of the loader fills it from the ore bin. When the bucket approaches, the loader commences to move, and while both are traveling together side by side at the same speed, the ore slides from the loader to the bucket. The loader travels on a four-wheeled carriage, and after discharging its load into the bucket returns by gravity to the ore bin for another load. There is a buffer composed of an air cylinder with a piston which checks the speed of the loader on its return. At the discharge end of the tramway the bucket carriages run from the loaded track rope on to a rail with about a 10-foot curve, then on this curved rail to the empty track rope. As the bucket goes on the rail it is weighed by the dial scale mentioned, the latches are unhooked automatically, and because the bail is attached to trunnions below the center of gravity the bucket turns upside down and discharges its contents into a bin. All this is done automatically while the traction rope continues to move the bucket around the curve toward the return-rope track, where the bucket is automatically righted and latched ready for its load when it reaches the distant loading terminal. The entire system is connected by telephones, and is almost automatic throughout. The third and final transportation stage is from the Texas terminal of the aerial tramway to Marathon station of the Southern Pacific Railway, 90 miles distant. This is accomplished by means of traction engines which haul ore-loaded wagons over the desert roads.

The diversified haulage systems used in connection with the Del Carmen Mine show conclusively that no one system of haulage will meet all requirements; further that it may require the adoption of several systems before the ore reaches its destination. The writer is indebted to A. Leschen & Sons Co. for the data from which this article is prepared.

BROKEN HILL, N. S. W., MINERALS

Written for Mines and Minerals

Mr. Aldridge, of Broken Hill, New South Wales, had an opportunity to collect from the Broken Hill mines, silver, lead, copper, and zinc mineral specimens, which are marvels of beauty. A large part of this collection is now at the Sydney

Fig. 7. Traction Engine Hauling Ore in Wagons

University, at least it was intended to send it there. The Aldridge collection of minerals is a unique collection of specimens obtained during the palmy days of Broken Hill.

The history of mining at Broken Hill forms one of the most remarkable and sensational chapters in the story of economic progress Discovered quite accidentally, the huge barren outcrop along the summit of the ridge was opened up by a small syndicate of proprietors, and, after some vicissitudes, the little party of poor men found themselves almost fabulously wealthy. Few mines have ever yielded such an output of precious metal, and certainly none has ever surpassed Broken Hill in the scientific interest of the problems in ore deposition which it presents.

Without going into technical details, it may be stated that

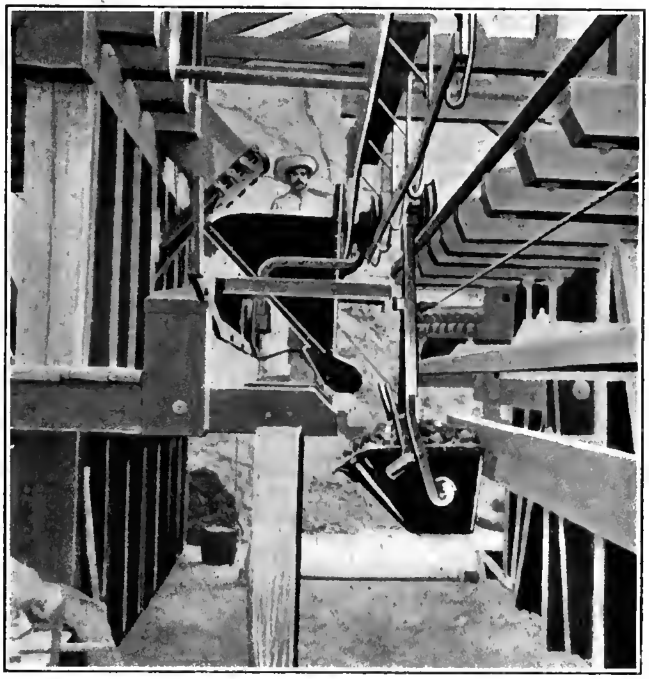


Fig. 6. Loading Bucket While in Motion

the ore below water level consists of an intimate mixture of galena (sulphide of lead), blende (sulphite of zinc), rhodonite (silicate of manganese), and garnet (a complicated silicate).

The galena and blende both carry relatively large percentages of silver. The minerals mentioned by no means exhaust the list of those found in the sulphide zone, though they bulk most prominently there. On this complex mixture the even more complex processes of Nature's chemistry have been at work for untold ages. Solutions ascending, solutions descending, have mingled, acted, and reacted until certain parts of the lode have been converted into an enchanted palace of wonders. Rare elements like bromine and iodine, tungsten, and molybdenum, present in unprecedented proportions, have combined into unique mineral species. Some types have never been met with elsewhere in the world, as, for instance, raspite (a tungstate of lead), marshite (an iodide of copper). Others, while known previously, were regarded as scientific curiosities, but at Broken Hill they were found in masses of almost unlimited extent. It was due to this fact that in the early days the silver output of the mine was phenomenal. Embolite (a chloride and bromide of silver) became a commercial ore of that metal.

Marvelous as is the variety of the minerals, even more wonderful is the beauty of the forms in which they occur. In the "oxidized zone" above the water level, the chemistry of Nature excavated cavities, large and small, and lined them with brilliant crystals of varied shapes and colors. So wonderful and unique are the forms presented by native silver and copper, cerussite (carbonate of lead), smithsonite (carbonate of zinc), azurite (carbonate of copper), embolite, and a host of others that specimens have been eagerly sought by collectors, and hold places of honor in museums the world over.

On account of the intrinsic value of the minerals, the companies operating the mines at length found it imperative to issue very stringent commands to prevent specimens leaving

the mines. Thousands of pounds worth of crystallized minerals were sent to the smelters, and it became increasingly difficult to obtain specimens. At the present time all the mines have worked out their oxidized zones, and, excepting a little patch left here and there, no more crystals are obtainable.

LAKE SUPERIOR IRON ORES

Hoisting has been suspended at the Cleveland-Cliffs Iron Co.'s Maas Mine at Neguanee, Marquette Range, because the upper portion of the shaft is being given a thick lining of concrete. It will take at least 6 months to finish the work.

The Lake Superior Iron and Chemical Co., which has absorbed a number of plants, is reputed to be controlled by Standard Oil capital. In its attempt to dominate the charcoal-iron industry of the West, it has for one competitor the Cleveland-Cliffs Co. This latter concern's Pioneer furnace at Marquette, Mich., is the largest charcoal-iron furnace in the world, and, in addition, it owns the Carp furnace at Marquette, and the Kipling furnace at Gladstone, Mich. Other producers of charcoal iron in Michigan are the Cadillac Furnace Co., Antrim Iron Co., Spring Lake Furnace Co., and East Jordan Furnace Co. These various "independents" have an annual capacity of close to a quarter of a million tons of iron, in addition to large quantities of chemical by-products.

Prospecting in the Dead River Valley, northwest from Negaunee, has been suspended for the winter, although it continues active in the Iron River district, where the most recent important discoveries in the Menominee range have been made. Two drills are in action in section 26, 43-34, owned by Wisconsin Land and Lumber Co. One drill is at work in section 20, 43-34 on the Waite farms. One drill in section 27, 43-35 for United States Steel Co.

After being idle since 1902, the Hilltop Mine, in the Crystal Falls district of the Menominee, is to be worked. The mine was not regarded with especial favor at the time of its suspension, but now, as a result of diamond-drill discoveries, it is expected to become an important producer. The matter of reopening the Cundy Mine at Quinnesec, Menominee range, is under consideration by the steel-corporation officials. The property has been idle almost a decade.

Rogers-Brown Co.'s Susquehanna Mine, at Hibbing, which for years operated as an underground proposition, is being converted into an open pit producer. The work of removing the overburden, estimated at 7,000,000 cubic yards, was awarded to Winston Brothers & Dear early last spring, and already in the neighborhood of 1,500,000 cubic yards have been stripped. Operations will be continued throughout the winter. Half a dozen steam shovels are in commission, many locomotives, and a large number of cars. The overburden is the thickest ever stripped on the Mesabi, its depth in places approximating 150 feet.

At its Kennedy Mine, on the new Cuyuna range, the Rogers-Brown Co. has completed the installation of a powerful new pump. The machine almost doubles the 2,200-gallon capacity of the pumps previously in position and assures that no trouble will be experienced with water in the underground workings.

The Cleveland-Cliffs Co. is increasing its holdings in Bates Township. It has recently started diamond drill work on the Kranz and Gustafson properties, in section 23, 43-34. Another crew of this company is located on the Sjoquist farm, in section 23, 43-34

Pickands, Mather & Co., which has been exploring the Spies lands, in section 24, 43–35, directly east of the James Mine, has suspended operations and it is likely the option will be surrendered.

Pickands, Mather & Co.'s Menominee mines, under C. E. Lawrence, of Iron Mountain, shipped 520,000 tons of ore last season. The shipments from the Florence Iron Co.'s Florence Mine were 208,000 tons, and those from Oglebay, Norton & Co.'s new Buckeye Mine, 89,000 tons. These two producers are the only Menominee range mines located in Wisconsin. The Menominee range shipments of Corrigan, McMinney & Co., which operates principally in the Crystal Falls district, were 630,000 tons.

The old Bird Mine at Crystal Falls has been taken under option by Joseph Croze, of Houghton, and will be given another test. It is said that $150,000 has been spent on the tract without finding ore in quantity.

It is reported that the Interstate Co., the Minnesota mining division of the Jones & Laughlin Steel Co., Pittsburg, is about to award a contract for stripping the Longyear Mine, 1 mile east of Hibbing, Mesabi range. It is estimated that the removal of 4,000,000 cubic yards of earth is necessary.

Preparations have been made to prospect a large tract of land in Keweenaw County by the Senter-Dupree Development Co., which has taken options on 1,520 acres near the middle of the district. Captain Thomas Hoatson, of Calumet, will be general manager, and Howard Wright, an engineer of experience, superintendent. The region to be explored is generally conceded to be worthy of a thorough test, as it is believed to carry several copper-bearing lodes. The property is from 2 to 3 miles due south of the Delaware station on the Keweenaw Central Railroad, and a short distance northeast of Gratiot Lake. There is over 2 miles of the strike of the formation on the Keweenaw Peninsula, and the property is from 1 to 1½ miles wide. The land lies close to the contact between the copper-bearing lodes and the eastern sandstone, and in a region which has received practically little attention. Diamond-drill contracts have been let to A. P. Silliman & Co., of Duluth, and Houghton; a road has been completed into the property; drills delivered, and several carloads of building material and fuel shipped.

On account of the low prices the Calumet & Hecla Mining Co. will report a materially lower production of refined copper for 1910 than for 1909. During the year the corporation has taken advantage of the situation to effect many economies, both in its own mines and at the properties of its subsidiaries, so that when production is increased copper will be produced cheaper than before. Although Osceola will show a reduction in output for 1910, the money made will be satisfactory to shareholders, and they will also have the satisfaction of knowing that their property is in better shape physically than ever. The South Kearsarge shafts command a highly mineralized formation; the North Kearsarge shafts have immense blocks of stoping ground, and in the Osceola proper there are thousands of feet of ground blocked out.

During the past year Ahmeek has given evidence of its coming greatness as a producer. This property is among the lowest cost producers of copper. Ahmeek, with the four-head stamp mill in operation, can produce 20,000,000 pounds of copper annually, which even at present prices, would give earnings of $1,000,000, or $20 per share. The daily capacity of the mill is approximately 3,000 tons. Ahmeek now has a market value of about $10,000,000, and the original investment was only $850,000, there being but $17 a share paid in on its 50,000 shares. The company has enjoyed the advantage of an excellent management from the start, and it is believed its copper will not cost more than 7½ cents per pound to produce.

In Belgium there must be a breathing appliance for every 200 underground workers in fiery mines of the second and third degree, with trained rescue brigades of not less than 4 men per set of apparatus. A leading company is organizing a well-equipped rescue station, staffed with 25 picked and trained men, who know every inch of the company's mines.—*United States Consular Report.*

BONANZA COPPER MINE, ALASKA

*Written for Mines and Minerals, by U. H. Wilhelm**

The Bonanza Copper Mine is the sole objective point of the Copper River & Northwestern Railroad, 200 miles in length, and costing approximately $22,000,000.

The railroad runs through an absolutely undeveloped district of no agricultural possibilities. The railroad is financed by the Morgan-Guggenheim syndicate, and will be soon advertised as a scenic marvel.

The first 40 miles of the railroad follows tide flats. At mile 50 is the Miles glacier bridge. Leaving this magnificent piece of engineering, the railroad is built in solid rockwork up the Copper River cañon. From mile 132 the railroad proceeds eastward paralleling the Chitina River at a distance of a few miles to the mouth of the Nizina. The Nizina is followed to the Kennecott, 4 miles from the mouth of which is the lower camp of the Bonanza mines.

The camp consists of the bunk house, mess house, ware-house, barn, assay office, and the saw mill, operated by steam. All the machinery and supplies have been freighted into the camp from Valdez, 180 miles distant, by means of horse sleds, at a minimum cost of 14 cents per pound. Mr. R. F. McClellan, who grub staked one of the original locators, is the superintendent, and Mr. Steven Birch is the general manager.

The lower camp is connected with the mines by a Bleichert tramway 15,000 feet in length, with a difference in elevation of 4,000 feet. The tramway has a maximum capacity of 300 tons in 24 hours, and cost $400,000, most of the cost being due to freight. The tram discharges at the lower end in ore bins, directly beneath which, on the hillside, a concentrator will be built, foundations of which are now being laid.

There are in all about 2,000 feet of mine workings, the highest tunnel being of the altitude of 6,200 feet; the lowest about 100 feet or less. The ores are chalcocite and azurite; the azurite appearing on the surface of the chalcocite.

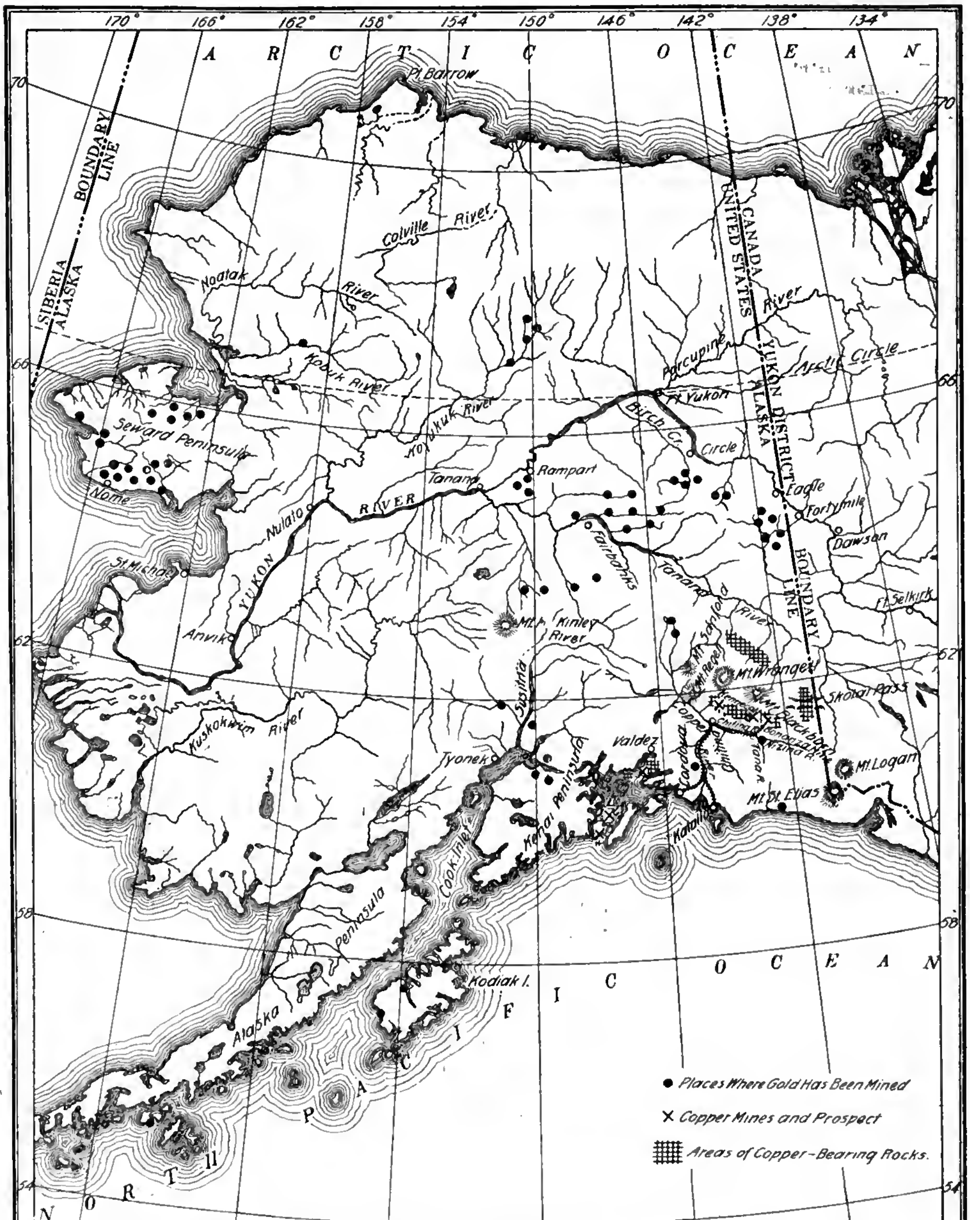
From the lower tunnel a raise passes through 40 feet of solid chalcocite, not a trace of limestone being visible. There is also a 235-foot tunnel running through the same ore, all walls showing solid chalcocite, with cross-cuts of 20 feet or less showing the same. At present but six men are working at the mine, though 30 men are working at the lower camp.

The track for the ore cars runs from the main tunnel to the upper ore bins over a landslide which averages 20 per cent. copper. This slide, or blow-out, contains individual boulders of chalcocite weighing as much at 200 pounds, and has been estimated to contain $8,000,000 in copper. The tramway and concentrator have been built to handle the ore from this slide, a switchback and main tunnel to tap the ore at depth being planned for the main ore body.

The main ore body is about 40 feet thick and of undetermined extent in other directions; it dips back and down so that the two original tunnels driven in at the outcrop run through it, and the tunnel 100 feet lower struck it at a distance of a little over 100 feet. The ridge of the mountain shows immense outcroppings along its entire length.

The same company owns the Jumbo Mine, said to be even richer property.

* Portland, Oregon.



Map of Alaska

OIL IN VENEZUELA

The eastern part of Venezuela and the delta of the Orinoco have been investigated and several tracts containing petroleum have been located. Geologists and mineralogists in the field under the direction of an expert mining engineer are at present exploring the regions around Lake Maracaibo. As this territory is especially rich in oil and asphalt lands, it is expected that the center of the company's operations will be in this locality and that exploitation will be commenced soon.

PETROLEUM MINING ENGINEERING

By Raymond S. Blatchley

Specialization has become a potent factor in modern progress, particularly in constructional work and development of great enterprises. Upon civil, mechanical, electrical, mining, and chemical engineers has fallen the evolution of great industries and the overcoming and harnessing of great forces of nature. These specialists are being turned out by hundreds annually to all quarters of America, where great railroad problems are being solved upon a methodical basis, where new machinery is being developed to minimize human toil and produce added power, where new resources of mineral wealth are being unearthed and converted into value, and where new industries with cheaper methods are being unfolded by the delving into latent mysteries of chemistry. Great results are apparent through the efforts of engineering, from which America has secured a world-wide reputation for energy and progressiveness. The future holds even greater promise to the advance of the nation along these same lines, in fields unexplored, and in many enterprises of the present day that have scarcely any specialization. In the latter class falls the petroleum industry.

Importance of the Oil Industry. Need of Special Knowledge for Successful Work

The petroleum industry, though but 50 years old, has presented a notable chapter in the world's commercial life, and particularly in America, where it has taken on its greatest activity, it has assumed such gigantic proportions in recent years that it wields a strong influence in modern life. The use of various oils, gasoline, and by-products of crude oil is universal and essential to modern progress, creating continuous demand for petroleum that will always be in excess of supply. Of the many branches of the industry, perhaps the most advanced is that of the manufacturing side. Chemists and physicists have made wonderful advance in creating new by-products, and especially in the utilization of waste materials. The future of this department of the industry is one of great opportunity to trained men, especially to chemists, for as yet the chemistry of petroleum is little known and varies widely for oils of different fields. A second and vital branch of the industry is that of the producing side. This offers greater opportunity to the trained man, in that it has little or no specialization, and certainly, through this lack, is in need of engineers to more thoroughly promote its many branches of technical work.

The present day does not warrant any worry over the lack of production, yet the inevitable decline of all of our present fields suggests a care for the future. In some near day production will need a new stimulus, and it seems that the proper way to create this is to develop a type of men peculiarly fitted to work out the problem of oil accumulation and to solve the many varied questions pertaining to its production. The object of this paper is the discussion of this new need.

The men who have developed the vast oil-producing areas of today have done the frontier work of a great industry. They are the pioneers who, through pure grit, costly practical experience, and an intense lure of the game, have made possible the present status of America's growth. Their patient toil and indifference to loss is significant of American success, yet, however profitable their operations may have been, it is easy to see that had they had a better knowledge of geological conditions their losses would have been greatly minimized. Their development of new fields, in practically all cases, dates back to wildcat discovery and to the gradual "feeling out" of pool limits. Often drilling was stimulated by oil seepage, either in outcropping or faulting of formations, in springs, or from mining of various types. Outside of these influences but little work scientifically has been done until within very recent years. Of late, government and state geologists have contributed to the development, and a gradual belief is arising that they are of service to the oil trade. These men are few in number in proportion to the enormous field. They are, for the most part, men trained in geological problems and are able to handle only certain phases of the development. They are not, as a rule, strictly trained petroleum engineers, because these are scarcely known in America, but they are trained geologists. Whatever benefit they have been able to render to the oil-producing business is of a geological nature. Their work has been practically limited to the vicinity of new proven areas, and the results accruing from the investigation of various fields is of immense value toward a broader interpretation of the work of the future petroleum engineer, especially in the search for new areas.

The questions now arise, is it possible to develop engineers of petroleum who can carry on work that will build up the producing side of the oil industry and put it on a scientific basis, as in other types of mining? Is it possible to go into new territory and be able to outline oil areas? The answer to the first question is "yes," and the method toward this end is to educate men along the lines of the petroleum business and at the same time bring them in touch with the chemical, physical, geological, technical, mechanical, and economic possibilities of the industry. The answer to the second question is, strictly speaking, "no," yet in a general way this is possible. The possibility lies in a close study of outcropping, dips, water saturation, and stratigraphy, the latter meaning the sequence of formations, and constructing therefrom a structural diagram showing the position of anticlinals, synclinals, terraces, and domes. The engineers, through the determination of rock structure and water conditions are reasonably assured of a working basis and an elimination of a goodly portion of the gambling nature of the business that oftentimes is astounding for its expense and impracticability. They are enabled to lead drilling to the most favorable conditions for the accumulation of oil and gas. Accompanying this phase of the work, as the advance is made into new territory, there arise problems of transportation, engineering, fuel supply, and other intricate portions of the business that can only be handled by the trained engineer.

Are there new fields to be opened up? As for this question it is sufficient to say that there undoubtedly exist many untapped fields in the United States, Canada, and Mexico. The fact that most of the fields of the world have been limited to the sedimentary rocks of varying age and that these are generally prevalent over the earth's crust is an answer in itself.

So far the discussion has been limited to the field for specially trained engineers. More systematic development of oil fields may be accomplished by including the operators and drillers themselves in a scheme of special training. The majority of the several hundred thousand persons connected with the oil and gas business in America are young men. Few of these have secured education that was of special benefit in their occupations. Why not educate the future generations with some of the technical training of petroleum engineers and thus provide them with better understanding of the scientific character of the oil business? At the same time that they are securing this they might receive the usual academic education and thus be equipped for the business of their choice. It is my opinion that the industry is ready for such action.

These few ideas are given merely to illustrate what the advantages would be to the industry as well as the individual. Whether these ideas have the favor of the oil trade is uncertain, since but little consideration has been given the subject. It is a large plan and an expression of approval or disapproval would mean much to those universities that have the establishment of such an engineering course in mind.

Some definitions and possibilities of petroleum mining are presented herewith. Petroleum mining is a branch of engineering. The term is as yet unfamiliar to the American oil industry, since it is more particularly a foreign expression, meaning in its strictest sense the development of oil properties and the winning of petroleum. The exploration for petroleum

is essentially mining, and it seems to fit no other classification. It is different from ordinary mining, however, in that it involves the laws applying to liquids rather than to those of solids. Moreover, the distribution and accumulation of petroleum is dependent on stratigraphic and structural features that do not apply directly to other minerals. Petroleum, accompanied usually by gas and salt water, is, to a certain extent, migratory and offers unsolved problems not found in any other type of mining. The underground manipulation in securing petroleum calls for peculiar machinery unlike that of ordinary mining, and thus introduces special mechanical problems.

Petroleum mining is closely related to the other branches of engineering and scientific work, such as civil and mechanical engineering, physics, chemistry, and geology, and necessarily borrows from them for the development of the engineer. In fact, there is occasion for specialization within the subject of petroleum mining and the development of petroleum mechanical engineers and petroleum chemical engineers becomes a part of the scheme. The possibilities to be gained under each subject are treated individually below.

Civil engineering offers training in constructional work of various kinds. The surveying and laying of pipe lines within the oil fields and from them to distant refineries requires accurate leveling, especially if extensive gravity systems are to be employed, similar to those of the Illinois fields. Some oil territories may require work in the construction and maintenance of wagon roads and light railways, such as are in use in several foreign fields. Overland pipe lines and light railways require possible rights-of-way. Future oil areas may lie close to large bodies of water and in event of this, docks or piers may be needed. The design and erection of power plants, field buildings, storage tanks, loading racks, and pumping stations require some training in civil engineering. This training is especially needed in order to determine well elevations for purposes of geological and structural study. Only in this way can one determine the dip and rise of various formations and the approximate limits of new and old areas.

The oil business has developed certain types of machines that are peculiar to its needs, and the possible development of additional useful tools and various mechanical devices opens up fields of wide opportunity to young men of an inventive turn of mind. Mechanical engineering offers a wide technical understanding of arrangement and use of machinery, and the incorporation of phases of this work in petroleum mining is desirable. The design and manufacture of special drilling tools for penetrating great depths of peculiar formations and for surmounting the many difficulties that overtake the driller is an important subject both for present knowledge and future investigation. The design of various pumping machinery for transporting viscous liquids, without blockade of movement, has a value of its own. The arrangement of power plants and pumping stations; the making, setting, repairing, and recovering of casing; the manufacture, assembling, and principles of steam boilers, steam economizing devices, steam, oil, and gas engines; the knowledge of steam, gas, and electric power; methods of excluding water by means of casing, packers, and use of cement; methods of sand pumping, bailing, and devices for inducing flow of oil, gas, or water, and many other detailed forms of mechanical work are useful, if not necessary, to the study of petroleum mining. Special opportunity is open to the petroleum mechanical engineer today in the use of various oil burners and oil-fuel apparatus in marine, railroad, and domestic application, especially since recent experiment has shown the practicability of oil as a source of cheap and efficient power. The condensing of portions of natural gas into gasoline and other similar volatile liquids at the gas well, without the loss of the gas, is commanding present attention. This field demands success because of the universal use of gasoline in motor power. Many other possibilities might be mentioned of the mechanical side of the industry, but the above are sufficient to show the need of specialists and also the general knowledge of the petroleum engineer.

The study of physics provides training to the petroleum engineer in various simple and everyday problems. The inequalities of temperature and their effect upon pipe lines, machinery, and even the product itself, is a subject of interest. The mechanics and laws of liquids, evaporation, congealing, pressure, equilibrium, weight, motion, etc., form the basis of mechanical engineering and its relation to petroleum mining.

The chemistry of petroleum is slightly known today except from its industrial side. It is a subject that has the earnest consideration of government officials and their efforts will pave the way to a wider investigation. The possibilities for successful research by chemists of petroleum are exceedingly promising. Aside from the scientific gain through research, new methods of analysis, fuel possibilities, new methods of distillation and refining, new by-products, more complete utilization of waste, products, etc., are open to investigation. An intimate study of the petroleum itself is needed. The problem of its origin is as yet not definitely solved, and new facts concerning its composition and organic relations may be of considerable aid in establishing clearer theories of its origin.

The knowledge of geology is perhaps one of the greatest aids toward the development of the petroleum engineer. In event that he will be called upon to do the bulk of outlining new fields, he must need a knowledge of the rocks of the area which he intends to investigate. An understanding of stratigraphy and geologic structure is indispensable. Some knowledge of paleontology will assist him in determining the character and classification of the oil formations, through various fossils of plants and animals. Mechanical means of portraying the structure of rocks is necessary, such as contouring, cross-section work, etc. The study of the dip relations of outcropping will materially assist him in his structure determinations. The principles of geology will aid the operators, drillers, and other men connected with the business in a systematic development of oil fields after the engineer has paved the way toward drilling.

The economics of petroleum would offer a valuable addition to the education of the petroleum engineer. The subject involves the discussion of necessary steps to successful handling of oil properties. Phases of leasing, choosing well sites, contracts for drilling, shooting of wells, taking the oil, power and property equipment, costs of the innumerable details of the business, costs of operating leases, oil investments, buying properties, gauging properties, geographical distribution of oil, and its transportation, storing the oil, keeping the records of samples of wells, statistics, legal phases of the business, etc., are various topics worthy of note.

The foregoing discussion of various subjects suggests the need of special training for the petroleum engineer. The list is long, yet by selection a good sound training can be gotten from them that will greatly aid the industry, as such training is not given at the present writing in any university in the world, though several foreign schools are considering the establishment of such a course and one American university has the plan in mind. The founding of petroleum mining as a university study will depend largely upon the attitude of those in the industry.

Some 15 years ago a mineral hitherto unknown to the scientific world was discovered in the Vermilion River district and was taken to Mr. Sperry of the Canadian Copper Co. to be assayed. He discovered it to be a diarsanide of platinum and the mineral was called after his name. Platinum is more valuable at the present time than gold and there is an active market for it, so that if the reports as to the amount and quantity of it are to be taken as reliable the discovery is of the greatest value.

EVOLUTION OF HOISTING

Written for Mines and Minerals, by E. B. W.

(Continued from January)

Owing to the first electrical engineers not understanding the conditions governing hoisting at mines, and in addition accepting entirely too many possibilities for motors, the first electric hoists were failures.

Methods of Calculating the Capacity of Electric Hoists and Output of Shafts

Most of the earlier motors used for hoisting were of the direct-current street-car type, geared to friction hoists. While direct-current motors are suitable for small hoisters situated near the generating power, they are not satisfactory when at a distance from the power, for at high voltage there is generally short circuiting at the commutator. The single-phase alternating-current dynamo can be made to generate a current of exceedingly high voltage; however, it is not suitable for hoisting machinery, because no single-phase alternating motor has been designed that is self-starting under load and capable of speed regulation.

The successful development of the multi-phase system has secured the advantages possessed by the direct and alternating currents, and aided in a large measure in meeting the requirements demanded of a mine hoister.

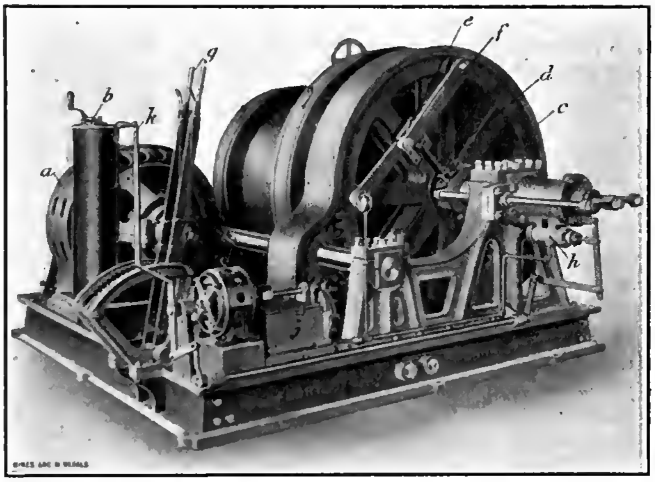


Fig. 52. Electric Hoist at Bliss Colliery

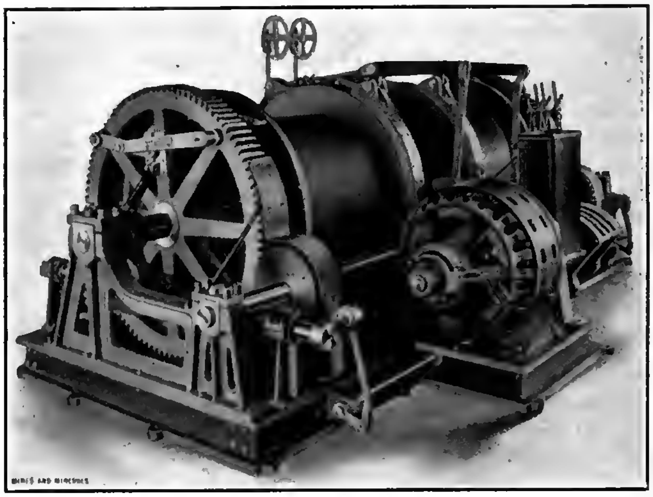


Fig. 53. Double Drum, Electric Hoist, Messina, Ala.

Frequently in the last few years electric hoisters have been described in technical journals; however, it is not to be taken for granted that all hoisters have been successful. At first glance it would seem that electric power would act the same as steam power in a hoister, but it must be understood that an electric motor will develop, for short periods, a torque equal to twice that which it can develop continuously. The steam engine is therefore better suited for an acceleration torque not much greater than the average, while the electric motor permits the acceleration torque to be about twice the equivalent of the horsepower it can continuously develop, that is, its rated horsepower.

Where an electric motor is driving a hoisting drum with variable load, the rated horsepower may be taken as being proportional to the square root of the mean square of the equivalent horsepower developed by the motor, taken over the time it is in motion, as the rating of a motor depends to a large extent on its capacity for dissipating heat, and the heat produced in the motor is proportional to the square of the equivalent horsepower exerted by the motor. If the depth of the shaft be S in feet, and s_1, s_2, s_3 feet be the distance described by either cage during acceleration, full speed, and deceleration, respectively, and the times $t_1 + t_2 + t_3$ be T seconds, the velocity in the shaft at full speed being V yards per minute, then the mean velocity over s_1 and s_3 will be $\frac{V}{2}$ feet per minute, and

$$V = \frac{120S}{t_1 + 2t_2 + t_3} \text{ ft. per min.}$$

The value of t_2 which gives the smallest motor under fixed winding conditions is $t_2 = T\left(1 - \frac{I}{I_c}\right)$ in which I is the total inertia of the winding engine, cages, etc., and I_c is the critical inertia of the motor obtained from the equation $I_c = \frac{1{,}220\ T^2 M}{120\ W}$ in which M is the load in pounds-feet, and W the number of revolutions made by the drum in completing one wind.*

A definition of this value would be: The critical inertia is that inertia which if existing in the hoisting motor under consideration would require the total time T seconds in order to run out from the speed attained, making the wind in T seconds with $t_2 = 0$.

Several calculations† having $\frac{I}{I_c}$ varying from .2 to .75, under various values of t_2 were made by Arthur Whitten Brown, in order to calculate the effect on the root-mean-square horsepower (R. M. S.). The curves from these calculations show that in every case the lowest R. M. S. horsepower is obtained when $t_2 = T\left(1 - \frac{I}{I_c}\right)$ and that at this point the equivalent horsepower during acceleration is about twice the R. M. S. horsepower.

The applications of this expression are many when considering the cylindrical-drum hoister, and it is seen that the more nearly $\frac{I}{I_c}$ approaches 1 the less practical will the winding become. Where possible the ratio should be between .4 and .65, the lower values giving the smaller motors and the lower power consumptions. The limit of output is probably reached with a ratio value of .7.

Similar expressions might be developed for other motor hoisters with cylindrical drums without balance ropes, or for conical drums, etc.

Starting with the assumption that the mineral load is G pounds, that the cars weigh $.5\,G$ pounds, and the cage and rope $1.5\,G$ pounds, a commercial value for the expression t_2 is found in an empirical formula. The total weight on the end of the rope is $3\,G$ pounds, and if the shaft be S yards deep the

* See November, 1910, Mines and Minerals, page 255.
† Arthur Whitten Brown, Trans. Manchester Geological and Mining Society.

weight of the rope per yard is found from the well-known formula $\frac{3G}{2{,}000-S}$, and the diameter of the rope $.45\sqrt{\frac{3G}{2{,}000-S}}$ inches. Taking x to represent the number of times that the rope diameter is contained in the drum diameter, then the drum diameter $=.45\,x\sqrt{\frac{3G}{2{,}000-S}}$ inches, or $.0375\,x\sqrt{\frac{3G}{2{,}000-S}}$ feet.

The torque due to the weight of the mineral would be $.01875\,x\,G\sqrt{\frac{3G}{2{,}000-S}}$ pounds-feet, but assuming the efficiency of the equipment is 85 per cent., then the total torque on the drum shaft becomes $.022\,x\,G\sqrt{\frac{3G}{2{,}000-S}}$ pounds-feet, and is the value of M. As a reduction gear is used between the motive power and the drum shaft, the efficiency must be reduced in order to cover losses in the gearing. The number of revolutions made by the drum to complete the hoisting is given by

$$W=\frac{S}{.04\,x\sqrt{\frac{3G}{2{,}000-S}}};$$

and assuming that the width of the drum is two and one-half times the width of W turns of closely packed rope, the width of the barrel is $\frac{28\,S}{x}$ inches.

The inertia of the drum has now to be considered, corresponding to weight in pounds × (diameter of gyration in feet)². The inertia of a straight drum may be taken as being equal to the inertia of an open cylinder equal in width to the drum, and having both its diameter and the diameter of gyration equal to the diameter of the drum. The thickness of the wall of the cylinder may be taken as 2½ inches. The inertia of such a cylinder, and therefore of the drum, would be as follows:

$$.45\sqrt{\frac{3G}{2{,}000-S}}\times\pi\times\frac{28\,S}{x}\times2\tfrac{1}{2}\times.28\times\left(.0375x\sqrt{\frac{3G}{2{,}000-S}}\right)^2$$

or,

$$.0388\,x^2\,S\left(\frac{3G}{2{,}000-S}\right)\sqrt{\frac{3G}{2{,}000-S}}\text{ pounds-feet}^2$$

This applies only to a single drum; a double drum would have greater inertia.

In a number of cases it was found that the inertia of all the moving parts in a hoisting equipment varied from three to four times the inertia of the drum. Taking three as a suitable factor, the total inertia I, becomes:

$$I=.117\,x^2\,S\left(\frac{3G}{2{,}000-S}\right)\sqrt{\frac{3G}{2{,}000-S}}\text{ pounds-feet}^2$$

From the definition of the critical inertia, I_c is equal to:

$$\frac{.00895\,x^2\,T^2\,G\left(\frac{3G}{2{,}000-S}\right)}{S}\text{ pounds-feet}^2$$

The ratio $\frac{I}{I_c}$ is then

$$R=\frac{13\,S^2\sqrt{\frac{3G}{2{,}000-S}}}{T^2\,G}$$

and from this is obtained the empirical formula

$$T=\sqrt{\frac{13\,S^2\sqrt{\frac{3G}{2{,}000-S}}}{R\,G}},\text{ or}\sqrt{\frac{22.5\,S^2}{R\sqrt{G}\sqrt{2{,}000-S}}}$$

One method of using this equation is in determining the cage loading for a given output of mineral from given depth. By reference to Fig. 54, which shows the winding time T seconds for any given output with any cage loading, based upon the assumption that the decking period is approximately 5 seconds + 5 seconds per ton. A table similar to the following is then prepared, calculated for an output of 100 tons per hour from a depth of 600 feet.

TABLE, SHOWING THE CAGE LOADING FOR A GIVEN OUTPUT OF MINERAL FROM A CERTAIN DEPTH

G	$\frac{3G}{2{,}000-S}$	$\sqrt{\frac{3G}{2{,}000-S}}$	T	T^2	T^2G	$13S^2$	$R=\frac{I}{I_c}$
2,240	3.74	1.930	26.0	676	1,520,000	520,000	.660
2,800	4.67	2.160	33.5	1,120	3,140,000	520,000	.355
3,360	5.60	2.365	41.5	1,720	5,800,000	520,000	.211
4,480	7.46	2.730	57.5	3,300	15,200,000	520,000	.093

From this it will be evident that a 1-ton load is practicable, but that the most suitable load would be 2,800 pounds.

If the decking period is inadequate any other value may be substituted from personal experience. The formula and data in the table will enable one to calculate approximately the maximum output of any shaft for any cage loading or to find the minimum practical cage loading for a given output. It is also of interest to note that R. M. S. horsepower is approximately $\frac{.0125\,Z\,G\,S}{(2-R)\,T}$, where Z is obtained from the chosen value of $\frac{I}{I_c}$. The equivalent brake horsepower during acceleration may be taken as twice R. M. S. horsepower, and the brake horsepower developed during the time t_2 is $\frac{.0125\,G\,S}{(2-R)\,T}$. These values do not make allowance for reduction gears.

In Fig. 52 is shown a mine hoist manufactured by S. Flory Co., which has several features that distinguish it from other

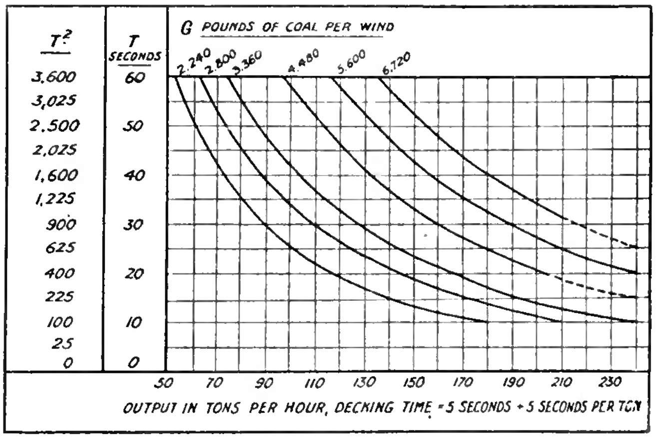


FIG. 54

electric hoists. This hoist was installed at the Bliss colliery of Delaware, Lackawanna & Western Co. It is a geared hoist supplied with a friction band, not shown, that operates on the inside cast-iron rim of the drum. The drum is 5 feet in diameter, 4 feet long between flanges, and capable of withstanding an 8,000-pound rope strain at a speed of 500 feet per minute. The hoist is operated by 150-horsepower Westinghouse alternating-current induction motor a suited for application where a strong starting effort is needed. The motor is governed by the single-handle, reversing drum controller b, which inserts resistance at the start and gradually cuts it out as the acceleration increases. The band friction which permits the rope to uncoil when it is free from the drum, or wind up when the drum is in motion, is known as the Werner band friction. It consists of a sleeve c, moved backwards and forwards on the drum shaft by a system of levers. The movement of this sleeve causes the spider arms d to move in slides e and force up or pull down the levers f which are keyed to rocker-shafts at their ends. The operation of the rocker-shaft tightens or loosens the friction band on the drum flanges. After the friction has been thrown in contact, it will remain in position until released. When the

friction is thrown out of contact, the drum is loose on the shaft and will permit the rope to pay out at high speed. The friction is thrown in or out of contact by the hand levers *g* or by a Westinghouse electric-railway air-cylinder *h* similar to that used on street-railway cars for setting brakes. The air compressor *i* and the air reservoir *j* are mounted on the base of the hoist. The reservoir is connected by pipes to the air cylinder at the end of the drum shaft, the latter being actuated by the lever *k*. The double-drum hoist shown in Fig. 53 is of the same type and was installed for the Bessemer Coal, Iron, and Land Co., Messina, Ala.

SUBMARINE DIVERS IN MINES

Written for Mines and Minerals, by George F. Duck

The use of divers is a comparative novelty in the West, therefore the following account from letters of Herbert A. Wilcox, superintendent of the Smuggler Mining Co., at Aspen, Colo., and W. J. Stevenson, manager of the Helena shaft, at Leadville, Colo., dealing with the work of two submarine divers in repairing submerged pumps at their respective operations, is of interest.

In describing the work at Aspen, Mr. Wilcox says that the collar of the Free Silver shaft, which is 1,196 feet deep, is 8,036 feet above sea level. The first connection underground is 848 feet below the collar and is with the ninth level of the Smuggler Mine. Below the ninth level, at intervals of 116 feet each, are the tenth, eleventh, and twelfth levels, the latter being at the bottom of the shaft. The mine below the ninth level has been under water for 12 years except for a few months in 1902 when it was unwatered to the eleventh level for exploration work.

At the lowest level, the shaft bottom, is a Jeanesville $32'' \times 14'' \times 48''$ duplex plunger pump, which, when the mine was allowed to fill 12 years ago, was left with connections arranged that it might be started under water with compressed air. This pump has not been in operation since that date except during the brief period of exploratory work in 1902.

When the present unwatering was contemplated it was found impossible to operate this pump, and its action led the management to believe that its packing had failed. As the inflow of water was great and the shaft small, it was found impracticable to unwater with sinking pumps. Although the flow of water at the eleventh level was at the rate of 1,500 gallons per minute it was lowered to this point by means of Pohle air lifts and Starrett air pumps delivering to the regular station pumps at the ninth level. As the mine equipment was insufficient to lower the water beyond the eleventh level, Messrs. Fred Johnson, diver, and George Peterson, tender, were called from New York, arriving in Aspen on October 17. On October 23, when the water was 103 feet deep, its surface being 6,943 feet above tide, the diver, after several attempts, reported his inability to work in this depth at the altitude, although at sea level he could work in 120 to 130 feet of water.

A station pump was then installed at the eleventh level and, with this and the Pohle and Starrett pumps in the shaft, the water was lowered to within 71 feet of the bottom by November 16, when the diver returned to work. During the next 10 days, while the diver was making the repairs, the water was held at a depth of from 71 to 65 feet. After the long submergence the packing in the water plungers was found to be in fairly good condition, but that in the steam end was entirely gone, and the piston rods were found to be very rough. By November 26 additional packing had been placed in the water end and all the glands tightened, the steam end had been repacked throughout, the rods smoothed, all nuts carefully gone over and tightened and jam nuts added where it was necessary.

The pump was started the next day on a mixture of steam and compressed air and has been running ever since with a piston speed of 120 to 136 feet per minute, with the exception of one shut-down of 4 days and several others of a few hours each, none of which, however, was due to any failure of the pump. The diver examined the pump on December 2, and again on the 14th, each time finding it advisable to add more packing to the steam end and to tighten the glands. At the date of Mr. Wilcox's letter, December 18, the pump was working nicely, and as there was but 10 feet of water in the station, he expected to recover the main pump in a few days.

Mr. Johnson had only been in Aspen 6 days when the preliminary trials were made. In diving in 103 feet of water he complained of shortness of breath, panting, and inability to exert himself without immediate and complete exhaustion, and described his sensations as being the same as when working at sea level at greater depths. Mr. Johnson estimated the effect of the altitude to be equivalent to that of about 27 feet of water at sea level. While the repairs were being made Mr. Johnson had no occasion to work under a greater pressure than 71 feet of water, and while not certain, does not believe he could work in mountain regions at as great depth as at sea level, even after becoming accustomed to the altitude.

Immediately after leaving Aspen, Messrs. Johnson and Peterson went to the Helena shaft, in Iowa Gulch, Leadville, the collar of which has an elevation of very nearly 11,000 feet, considerably above that of the Free Silver shaft at the former town. In describing the work at the Helena, Mr. W. J. Stevenson, the manager, says that when the mine was closed down some months ago the valves of the station pump at the 500-foot level were left closed and consequently had to be opened before the pump could be started and the lower portion of the shaft unwatered.

The Helena shaft consists of two $4' \times 8'$ compartments, in one of which are the pipe lines connected with the pumps, and which are supported by the stulls. Mr. Johnson at first attempted to lower himself in the pipe compartment, but was stopped by a pair of misplaced stulls just above the pump, and was unable to get between them and the side of the shaft. A second attempt was made to get down through the hoisting compartment, in which a Starrett air pump was working. This pump works entirely under water, and at the Helena shaft had been so placed that the exhaust took place about 2 feet above the top of the pump. This attempt also resulted in failure, as each time the diver got within about 4 feet of the exhaust he was blown back by the force of the air.

Unfortunately, a Cameron sinking pump had been wedged across the hoisting compartment, just below the station, so that ordinary sinking pumps could not be lowered. After several attempts, the pipes to give the necessary submergence for an ordinary air lift, were pushed past the obstructions, and when the water had been lowered to a depth of 50 feet Mr. Johnson was able to descend and open the valves in the station pump.

It will be noted that the surface of the water under which the diver worked was at Leadville 10,450 feet, and at Aspen 6,943 feet. This difference of 3,507 feet must have had some effect upon the ability of the diver to work, but Mr. Stevenson is of the opinion that the men labored under no greater disadvantage than would any one unaccustomed to the altitude.

While the work required of the diver at Aspen was much greater than at Leadville, in each instance most important service was rendered, and portions of the mine per force abandoned or only recoverable after long delay and great expense for new machinery, were cheaply made available for development. As Mr. Stevenson justly says, the remarkable feature is that the divers work entirely in the dark, guided only by their sense of touch and a thorough knowledge of their work.

RUSSIAN FAR EASTERN GOLD FIELD

Written for Mines and Minerals

The opinion of most geologists who have examined the Priamur Province is that the gold area is comparatively untouched and that up to the present only sands of the richest placers have been worked. Rock gold or quartz ore is found in every mining district, but scarcely any work has been done upon it excepting at four or five points. In the opinion of these geologists the alluvial gold in the Priamur district is most frequently found in the following primary combinations.

The Priamur Province With Its Mining and Labor Problems

In the Selemdzhinsk district it lies in a zone of granite and porphyry; in the Bureinsk system the deposit of gold is combined with gneiss and crystalline schist, but among the granites and porphyries there it never has been, and probably never will be, found. In the Amgun system the metal is found in connection with the clayey schist and schistose metamorphosed beds, and, finally, on the Ochot-Kamtchatka shore the alluvial sands are found associated with granite-syenite and gneiss, in hornblende schists.

In view of what is stated, if geological knowledge were wider and more detailed in the Priamur district, the gold industry might be on a firm footing and it might be developed for the future in a way more calculated to inspire confidence. Unfortunately, little geologically is known of the Far East, and the plans for working the area and of investigating it, as adopted by the geological committee, certainly suffer from a number of radical defects. After making an allowance of a fair sum per annum to the chief of the geological surveyors and their assistants, the committee is satisfied with their working from 2 to 3½ months, allowing them to pass the rest of the year in getting there and back, and living about 6 months in St. Petersburg. Notwithstanding this, the gold industry in the Priamur Province is gradually growing, as can be seen from the following statistical figures of the Irkutsk Mining Government:

	1905	1906	1907	1908
Gold produced, poods*	458	481	547½	478
Number of mines worked	251	237	278	297
Mines unworked	617	643	617	790
New claims	300	360	556	457

*A pood equals 36.07 pounds avoirdupois.

It is to be observed that the figures given for the production of gold are less than the actual figures, since with the free circulation of gold in Russia, part of it escapes registration, particularly since the greatly increased gold workings of recent times. To what extent the concealment of gold takes place can be seen from the fact that in 1905 M. Oparin and the Russo-Chinese bank bought at Blagoveschtchensk, 90 poods of gold. Of late years it has particularly been observed that the number of buyers has increased at all the mines. These deliver gold privately to the laboratories under the name of free gold producers, and partly get rid of their gold secretly and abroad. In this business the Chinese workman and the Japanese merchants have particularly distinguished themselves.

It is not uninteresting to observe that when measures began to be adopted against flooding the gold mines with Korean labor the number of Chinese workmen correspondingly increased on the gold workings, and along with this the movement of gold into China likewise increased. On the gold workings the average earnings of a Chinese come to about 50 kopecks (25 cents) per day but with the locally high cost of living he can hardly exist on this, whilst in the autumn every Chinaman takes hundreds of roubles to his home.

One may judge the dimensions and the growth of poaching by the quantity of free gold delivered at the Irkutsk gold smelting laboratory. In 1905 such gold was smelted to the extent of 86 poods and in 1908 to over 138 poods. The workmen at the mines constitute a particular type and are known generally under the name of miners. They are an uncommonly noisy and shifty people that can easily be recognized by their manners and customs. These characteristics—which have been produced by the severe conditions of the virgin forest and the gold fever, are obvious in each workman.

With such a contingent it is always difficult to get along, but recently there has been the addition of general strikes with which to contend, and concessions must be made to the workmen in view of a direct loss in business. The number of miners in the Priamur gold fields was, in 1905: Amur district, 955 Russians, 1,125 Chinese, 550 Koreans; Zea district, 2,187 Russians, 2,775 Chinese, and 1,156 Koreans; Primorsk, 1,700 Russians, 600 Chinese, and 700 Koreans; total, 4,842, 4,500, and 2,406, respectively. In 1908 the respective totals were 5,932, 7,483, and 2,492. It will be observed that while there has been a marked increase in the number of Russian gold miners, that of the Chinese has been much greater, while Korean figures remain about stationary.

There has been considerable agitation against Korean and Chinese labor, the Koreans particularly, because their government is not strong enough to make terms for them. The Russian insists on having Russian work done by Russian labor, and the particularly severe laws enacted to exclude foreign labor have forced the Koreans to resort to subterfuge in order to get back to their favorite work. One of the latest, which is attributed so frequently to another race, has been adopted by them. They are adopting the Russian orthodox faith, whereupon they are made Russian subjects and they become so, says a note from Kharbin, under the pretext of having subscribed to all that is required of a Russian, which entitles them to all a Russian's rights. It is not unlikely that when they have all they want they will revert to their old faith and return to their own country. Koreans are recognized as excellent miners, and were it not for that, the cheapness of their labor would not count much. In connection with the foregoing statement of laborers in the Priamur Province, particular attention is called to the fact that they stood firm in their numbers in spite of the exceptional measures taken against them.

During later years living conditions have considerably changed in the district, and the number of Russians arriving at the mines has increased. These are not entirely given over to gold mining, whereas the Chinese and the Koreans are engaged exclusively in the mines. The rate of pay for taking out a cubic sazhen (12⅔ cubic yards) varies between 2 roubles, 50 kopecks and 3 roubles, and the pay for the gold working is on the average 1½ to 2 roubles per zolotnik of gold produced. (A zolotnik equals 65.8 grains troy). At the same time the gold worker is obliged to give to the proprietor of the mine from 15 to 30 dolis per day (a doli equals .68 grain Troy or 2½ cents) according to the conditions, and dependent upon the richness of the alluvial. The gold produced by the gold workers over and above the quantity stipulated is considered as their own property and this attracts both Chinese and Koreans, who frequently sell some to the Bureau of Mines and take the remainder home to their own country.

The technique of mining is on a low level in the Far East, and seldom is there shown any desire to take advantage of modern improvements. Therefore, the industry is passing through a rather quiet time, and to a certain degree this is straining the owners. The use of the horse as power has outlived its period; and the application of steam and electricity is not everywhere introduced. Not long ago the introduction of excavators proved to be unsuccessful and unprofitable. Hope mostly rests on dredge working, although in the use of dredges one often sees a complete ignorance of the business. On account of the defective equipment of the owners and managers of the mines, many of them have gradually fallen into the hands

of more educated foreigners In order to show to the Russians the possibliity of taking advantage of the gold reserves, there must be better informed people, and for this it is not enough to have a mining school with 2 teachers and a budget of 30,000 roubles ($15,935) for the whole of the wide area of Eastern Siberia. The Irkutsk Mining Government might do more for the district and might win more in the opinion of society if it dropped the care of salt factories and mineral water and devoted attention to wider geological surveys and to the technical training of young people.

INDUCTION MOTOR RUNS UNDER WATER

The performance of apparatus under exceptional circumstances reveals its suitability for the service for which it has been installed. Due to the fact that reliable manufacturers design their apparatus with a large factor of safety, remarkable records are sometimes made.

A 20-horsepower, three-phase, 220-volt, standard induction

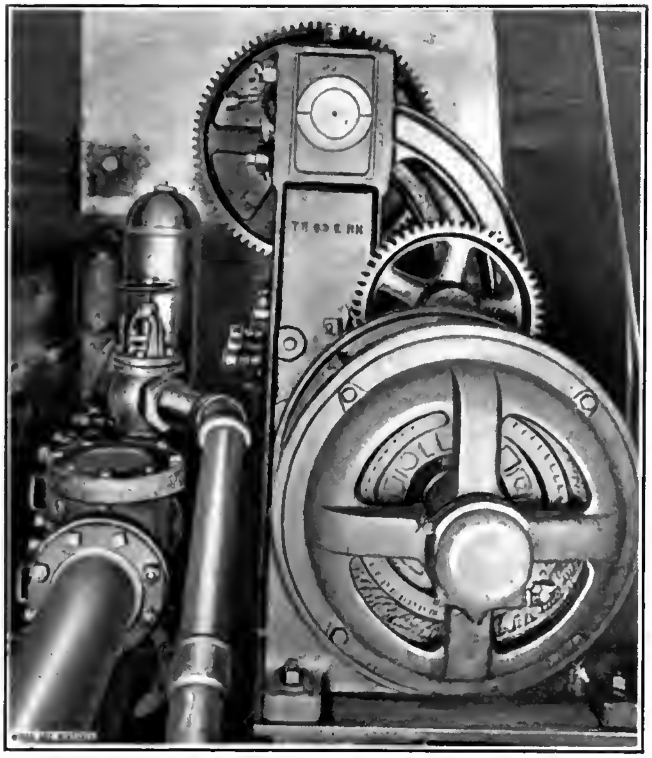


Fig. 1. Motor That Ran Under Water

motor, made by the General Electric Co., recently demonstrated the ability of their motors to operate successfully under the trying conditions met with in mines where the air is damp, without their being totally enclosed. This is of importance, as thorough ventilation of the motor is absolutely necessary due to the high temperatures which prevail if enclosed. This motor is geared to a mine pump at the bottom of the shaft at the mines of the Richmond Iron Works, Richmond Furnace, Mass., and is installed where the air is always damp, and moisture is continually dripping from the roof timbers. It is protected from this water by a piece of tarred paper tacked above it, so as to conduct the water away.

During a winter thaw the surface water broke in on one of the upper levels, and flooded the mine. The water rose so rapidly that the electric pump was soon under water, the top of the motor being 2 feet below the surface. Under such circumstances it was imperative that the pump should continue in operation so long as possible, and it was not shut down. It continued to operate perfectly, and at the end of 2 hours, during the whole of which time the motor was submerged under water, it had pumped itself clear of water. It was then stopped only long enough to clean the dirt and chips from around the rotor and put oil in the bearings, and then was started up again. This pump has been running about 20 hours a day ever since, and the motor has apparently suffered no injury from its unusual experience.

MINING IN THE MALAY STATES

The following from an engineer in charge of a mine in the Federated Malay States, gives color to the labor troubles in that Mohammedan country:

"Soon the Malays celebrate their great festival, lasting for three days, during which time eating, drinking, driving in carriages and rickshaws, constitute the program. This festival is a fitting end to the great fasting month during which no true Mohammedan is allowed to eat but twice in 24 hours, and that when dark, at 6:30 p. m. and 2 a. m. Between these hours they are allowed neither to eat nor drink, not even water, nor are they allowed to swallow their own saliva, but must expectorate. They fast 15 hours every day. The festival is a nuisance as the laborers are always getting sick and some become too weak to work as the days advance. It is fortunate that it only lasts one-twelfth of a year."

NEW INVENTIONS

Complete specifications and drawings for any of the following patents can be obtained from the Commissioner of Patents, Washington, D. C., at the following rates.

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
At entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED DECEMBER 6 TO DECEMBER 27, 1910, INCLUSIVE

No. 977,522. Coke-drawing machine, Howard Greer, Jr., Chicago, Ill.

No. 977,444. Apparatus for mining sulphur, Herman Frasch, New York, N. Y.

No. 977,956. Separator for lump material, Arthur Langerfeld, Scranton, Pa.

No. 977,955. Cutter head for tunneling machines, John Prue Karns, Boulder, Colo.

No. 978,321. Apparatus for handling mine cars, James McEvoy, Toronto, Ontario, Can.

No. 978,236. Process of utilizing coke breeze, Henry A. Tobelmann Warren, Ariz.

No. 978,659. Dynamite thawer, Sheridan S. Scholl, Roanoke, Va.

No. 978,642. Air reservoir for mines, Patrick Quinn, Forbes Road, Pa.

No. 978,586. Rock drill, Charles A. Hultquist, Bisbee, Ariz.

No. 979,086. Method of mining coal, Thomas Nichol, Glen Jean, W. Va.

No. 979,319. Mining starter bit, George G. Mayer and Reese Ashton, South Bethlehem, Pa.

No. 979,253. Operating mechanism for ore jigs, William Alma Bradley, Spokane, Wash.

No. 979,349. Furnace for roasting ores, Xavier de Spirlet, Brussels, Belgium.

No. 979,180. Method of treating ore, Thomas J. Lovett, Chicago, Ill.

No. 978,971. Process of treating fine ores, James N. Whiteman, Philadelphia, Pa.

No. 979,584. Coal washer, Newton A. Smith, Marion, Ill.

No. 980,004. Coke drawer, Andrew H. Reeder, Stonega, Va.

No. 979,597. Mining machine, Spencer Jay Teller, Unadilla, N. Y.

No. 979,857. Apparatus for ore concentration, Theodore Jesse Hoover, London, England.

No. 979,820. Flotation tank for ore dressing, Samuel K. Behrend, New York, N. Y.

No. 979,921. Ore furnace, Charles J. Best, Oakland, Cal.

Mines and Minerals

Vol. XXXI—No. 8. SCRANTON, PA.—MARCH, 1911—DENVER, COLO. Price, 25 Cents

EL COBRE MINES, CUBA

*Written for Mines and Minerals, by E. G. Tuttle**

The island of Cuba was discovered by Columbus on his first voyage of exploration, October 28, 1492. He landed at what is now called the Bay of Nuevitas and took possession of the island in the name of the King of Spain. The island has been called successively Juana, Santiago, Ave Marie, and finally regained its original Indian name Cuba. Diego Valesquez was appointed governor of the island in the year 1511, and to him fell the task of subduing the warlike and savage Caribs and Nahacs.

Old Shafts and Workings That Have Been Pumped Out and Valuable Ore Recovered

It was due to the importance of Havana that the French, Dutch, and British buccaneers during the sixteenth and seventeenth centuries attempted on several occasions to capture that port. While unsuccessful, they succeeded more than once in carrying off valuable booty. When Spain was at war with Great Britain and France, in the year 1762, a strong British army under Lord Albemarle captured Havana after a siege of 2 months. The British retained possession of the island until the following year, when by treaty it was restored to the Spanish authorities on June 6, 1763.

The numerous wars and insurrections which have occurred in Cuba have been a drawback to the development of her natural resources. The early Spaniards were good mine directors, when slaves did the work, even although they never developed their mineral resources in Cuba to any great extent. Copper occurs in many parts of Cuba, and openings have been made near Cienfuegos and Santa Clara, in Santa Clara Province, also in Pinar del Rio Province, but the principal mine is that of the El Cobre in Santiago Province, Fig. 1. The Oriente, as this end of the island is called, is rich in minerals. There are large deposits of manganese and iron oxides in addition to copper and gold. An east and west mineralized belt extends along the southern coast in which the El Cobre mine is located, about 12 miles to the west of the city of Santiago. The iron mines of the Spanish-American Iron Ore Co., near Daquairi, are some miles to the east of Santiago. The new mines of this company are almost directly north of the city of Santiago, at Mayari, near Nipe Bay, where iron ore said to contain 2 per cent. nickel is mined. El Cobre is connected by a railroad 9 miles long with Punta Sal, on the west side of Santiago harbor, nearly opposite Santiago city. It is necessary to cross the bay on a boat from Santiago to Punta Sal to reach the mines by rail, but there is a wagon road from Santiago to Cobre.

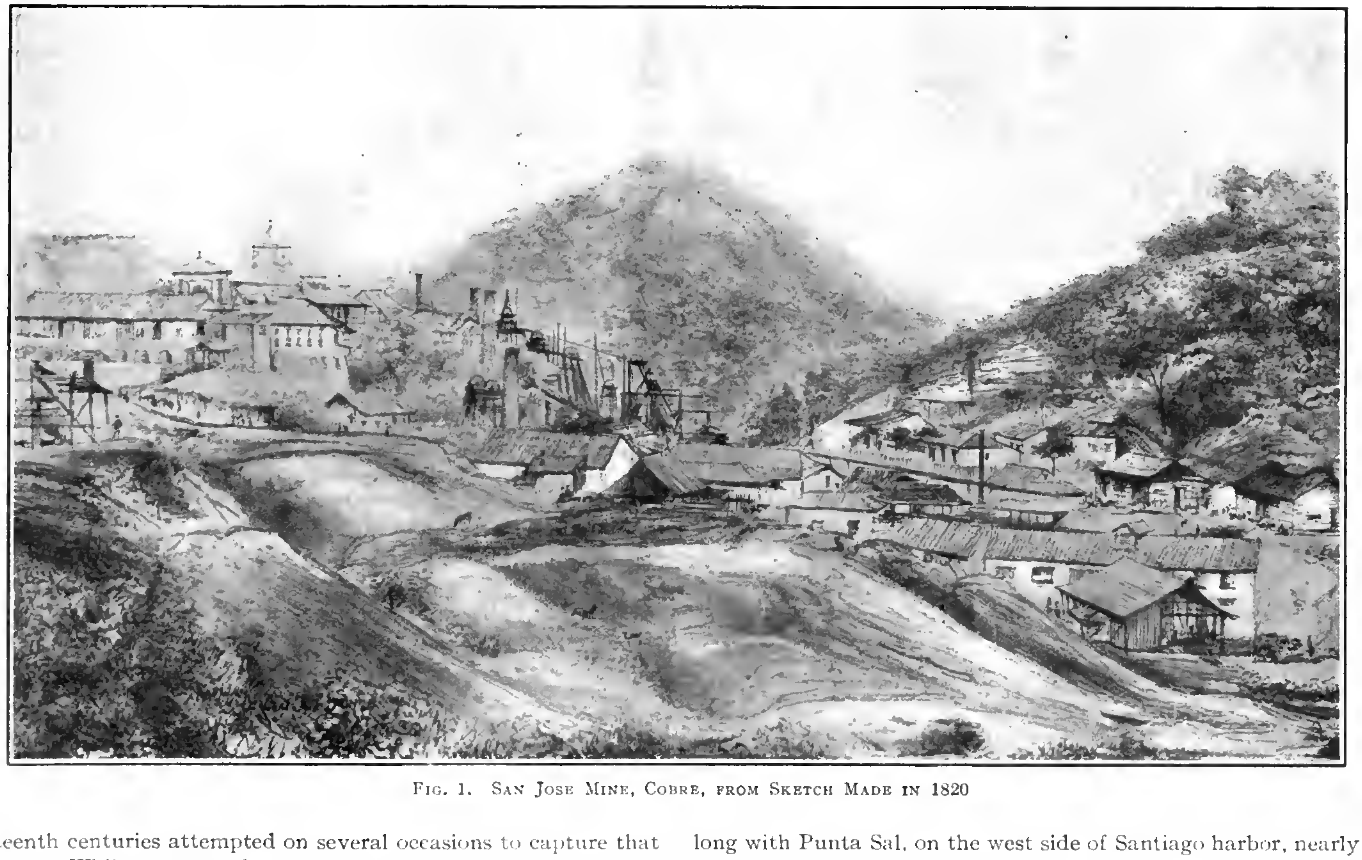


Fig. 1. San Jose Mine, Cobre, from Sketch Made in 1820

The first copper mining in the western continent, provided that of the aborigines is neglected, was done in the Province of Santiago in 1524. The mines at Cobre were worked irregularly from that time to 1832 or 1838. Previous to this time separate and individual property holdings were numerous, then there was a consolidation and several English and Spanish companies operated. In 1834 an English company bought and consolidated the principal properties of El Cobre, and operations increased

* E. M., 30 East Logan Street, Germantown, Philadelphia, Pa.

extensively. They began to assume some magnitude between the years 1840 and 1850 and continued to be prosperous until the beginning of the 10-year war, from 1868 to 1878, when operations ceased and most of the plant was destroyed by the insurgents in that rebellion. The slave insurrections in 1844, also the Lopez filibustering expedition of 1848 to 1852 probably influenced the steady operation of the mines. During the period of activity from 1834 to 1868 the annual average production of copper was about $1,000,000 yearly. The richest ores were shipped to Swansea, Wales, and this required that the ore contain more than 7 per cent. copper. The old records state that large bodies of 3 per cent. and 4 per cent. ore were left in the mines and that lower grades of ore produced in mining were either dumped in the stopes or outside the mine. From some of medium-grade ores matte was produced for shipment. It is stated that some of the shafts obtained depths of 1,200 feet and were equipped with Watt engines for hoisting, and Cornish pumps to remove water, as shown in Fig. 2. From an examination of the old machinery it was evident that it was built substantially and that also the boiler plants were well constructed. Flat hemp and flat wire ropes weie used for hoisting. The seepage water, and especially that during the rainy reason, filtered through the waste in the stopes and resulted in a considerable volume of acid water accumulating, which had to be removed by the pumps. This water attacked the cast-iron pipe columns and deposited copper and ochre to such an extent as to diminish the area of the column pipes. To prevent this as much as possible, long lines of precipitating troughs were constructed underground and filled with iron to precipitate the copper and reduce the acidity of the water. In addition to this, long lines of Californias, as such troughs are called, were constructed outside the mines, as shown in Fig. 3, to recover any copper in the water discharged from the pumps or flowing from the mine.

The concentrating mill was built below the mines for the treatment of some of the sulphide ores. Most of this is in ruins, but from the remains the plant apparently consisted of trip hammers similiar to that shown in Fig. 4, for crushers, and of jigs and buddles. From the arrangement of the plant it was evidently of small capacity and costly to operate, besides the tailing shows considerable loss. One or two attempts were made to reopen the mines after the 10-year war without success, and they practically remained idle from 1868 until 1902. When

Fig. 2. At Left, Watt Engine, Passenger Shaft; in Center What Remains of Old Cornish Pump

the property was acquired by the El Cobre Mines Co., an American concern, it consisted of about 800 acres. The mines were filled with water to within from 100 to 300 feet of the surface, and it was decided to use the Hardy shaft as the main pump shaft for unwatering the property. The old records and maps that were available did not furnish much information about the early surface workings, besides the entrances to them in many instances were caved and covered with thick tropical growth or obliterated by the wash of periodical rains through many years. The data available gave more information regarding the intermediate workings, and this information increased practically with the depth of the workings down to about 1,200 feet, or at the depth attained when operations ceased. Much of the data desired was not obtainable either because records had not been kept, or were misplaced and could not be found. It was also difficult to find the where-

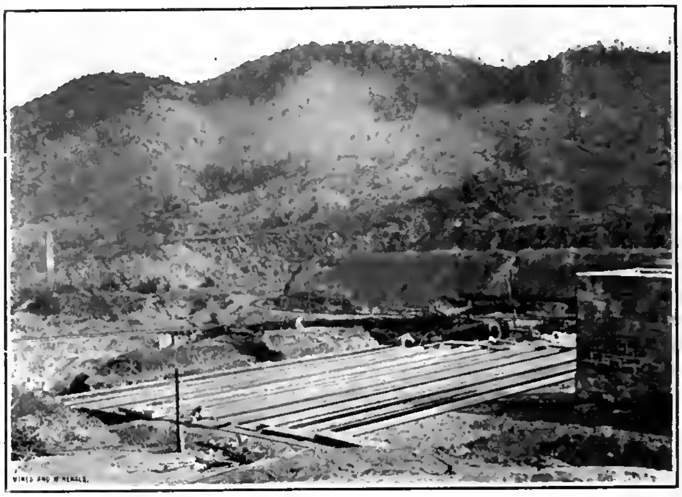


Fig. 3. "Californias" for Recovering Copper from Mine Waters

abouts of any old employes who were able to give verbal information concerning the old workings.

In the early days the ore in the veins was stoped down so far as water could be readily handled, and as the veins were leached from 100 to 300 feet below the surface, it was necessary to do considerable unwatering before any of the ore left in the old workings of moderate depth could be reached. Among the various difficulties encountered before starting up these old mines was the reconstruction of the railroad from Punta Sal to El Cobre. This road was built in the early days, down a valley as shown in Fig. 5, and operated by a Spanish company. Washouts were not infrequent along this railroad, and when the El Cobre Co. bought it many of the masonry bridges crossing the river were washed out. The river, which is apparently harmless most of the time, becomes a menace to the railroad and the mines during the rainy season. After the railroad had been put in working order a portion of it was washed out about a year afterwards and this retarded the work of development. Interruption at this time meant running short of coal at the mines, which in turn would interfere with the pumping which was the principal work in reopening; and most of the expense incurred for labor and material was directly or indirectly for the purpose of lowering the mine water. If the water gained on the pumps or the latter were out of order the work had to be done over again and greatly increased the cost. All efforts were at first concentrated on unwatering the mines to those levels where ore remained in the old stopes, and to this end an expert diver was employed to attend to submerged pumps needing repairs and the removal of obstructions from shafts ahead of the pumps. Another difficulty at the start was in pumping acid waters. Some of the shafts were timbered through old stopes, and the water seeping in these shafts through old fills was very acid, requiring pumps with bronze water cylinders and lead-lined water columns to be installed. The seepage was particulary heavy after the rainy season began, and therefore care was required to prevent the acid waters from corroding the wire ropes, cages, and cars.

Another difficulty which the American company encountered was in determining locations where the ground was sufficiently firm to place foundations for power plants, engines, etc., in other words where it had not been dangerously undermined. In cases it was necessary to proceed without this knowledge

except so far as it could be determined from surface indications, consequently some of the locations proved so insecure later on that parts of the plant had to be rebuilt in other places.

Good progress was made in unwatering the mines, and in about 1 year the shallow workings could be explored, and some new openings were made on ore deposits that were near the surface. During the year 1904 the old workings were unwatered to a depth of about 600 feet and the mud which had accumulated in most all of them, sometimes nearly filling them to the roof, was cleaned out. Explorations were also carried on in new ground which resulted in finding ore. Some large bodies of silicious sulphides containing from 2 to 4 per cent. copper were discovered in the old workings, and were developed in the walls

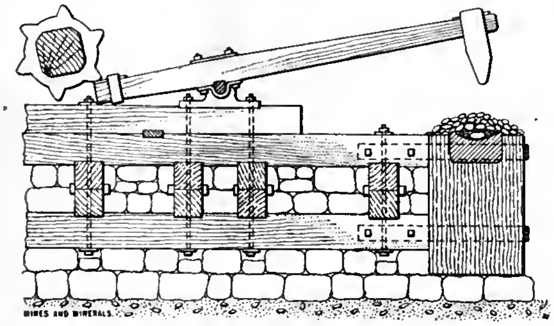


Fig. 4. Ancient Trip Hammer for Breaking Ore

of the stoped veins. Many blocks of higher grade ore were also found in various parts of the mine, besides some parallel veins to the southeast were explored which looked favorable as ore producers.

The high-grade ore taken from new openings and in exploration, beginning in 1903, was shipped to the United States. This averaged from 1,500 to 1,800 tons of 8 to 10 per cent. copper per month. Late in 1905 a depth of about 800 feet was attained at the Hardy shaft where a permanent pump station was installed on the ninth level. This was equipped with 18 in.×28 in. and 14 in.×13 in.×24 in. triple expansion Prescott pumps capable of lifting 1,500 gallons of water per minute. This arrangement of pumps drained the water from all the shafts that were being opened along the vein system to the east and west as shown in profile, Fig. 6.

In 1903 it was decided to build a smelting plant at Punta Sal for the treatment of El Cobre ore. This was completed and put in operation toward the end of 1904. It consisted of a 600-ton concentrating mill capable of enlargement, a briquet plant, and two 200-ton shaft furnaces. The low-grade sulphide ores were concentrated to a ratio of 4 to 1. The mines were equipped so that they could furnish about 750 tons of ore daily, which was to be composed of about 100 tons of 7 to 8 per cent. copper, 100 tons of 5 to 6 per cent. copper, and 550 or more tons of 2 to 4 per cent. copper. By concentrating the third-class ore and some of the second-class ore and mixing them with some of the first-class ore to obtain in all about 200 tons for the smelter charge, a surplus of the first-class ore was left for shipment if desired. It was found necessary to make some improvements to the concentrating mill after starting up. The furnace slag was disposed of at first by dumping it in the water along the shore at Punta Sal as shown in Fig. 7. This plan however was changed and the slag was granulated and hoisted for slacking in piles.

The production of the furnace for January, 1905, was 307 tons of matte containing 39½ per cent. copper; and in February of the same year the furnace produced 271 tons of matte, containing 34.12 per cent. copper.

The vein systems at El Cobre have a nearly east and west strike. The main workings are on what are known as the middle and the north veins shown in the plan, Fig. 8. Near the outcrop, these veins are about 350 feet apart and from 15 to 25 feet in width and occasionally wider. The north vein has a dip of about 55 degrees to the south and approaches the middle vein with depth, as shown in Figs. 9 and 10, which are sections through *A B* and *C D*, Fig. 8. A nearly north and south fault displaces the vein system so that on the east side this system is thrown about 200 feet to the north. The ores are mainly copper sulphide and contain little gold or silver. There are several other parallel veins on which recent developments to the south would suggest that they were of considerable importance.

The geological formations in the vicinity of the mines are for the most part tertiary eruptive andesite, although slates also occur, and exposures of porous calcareous formations are occasionally seen flanking the country rock. Associated with the deposits are large intrusions of quartz porphyry, which where exposed at the outcrop shows extensive weathering, decomposition, and leaching. The ore seems to have been developed near and in the porphyry. To the south there are some very promising copper croppings in a region of slate formation.

The first important opening on the easterly part of the property is the Pozo Verde shaft. Then proceeding westerly, see Fig. 6, the openings are as follows on the south slope of the hill:

Shaft No. 1, Hardy shaft; shaft No. 4; Arrieta shaft, not shown; Richards shaft; Passenger shaft; San Juan shaft; San Jose shaft; and on the north slope of the hill and beyond is the Mina Blanca and other openings. The Trevence shaft is to the south of the San Jose shaft, and is shown on plan, Fig. 8, as are the several tunnels 1, 2, 4, 5, opened on the north side of the hill and driven southerly to the veins. Through these the ores mined near the surface in the early days were trammed.

The Mahon shaft is an old important opening on the north side of the hill near tunnel 5. Some distance to the south of the first-mentioned line of opening and on the northern slope of the mountain is the Santiago mine, and some distance to the west of this is Jueves Santo mine.

Pozo Verde is an old mine of considerable depth. It was reopened to some distance below the fourth level which is 216 feet below the surface, as shown in the profile, Fig. 6. By pumping at the Hardy shaft the water lowered so that from the

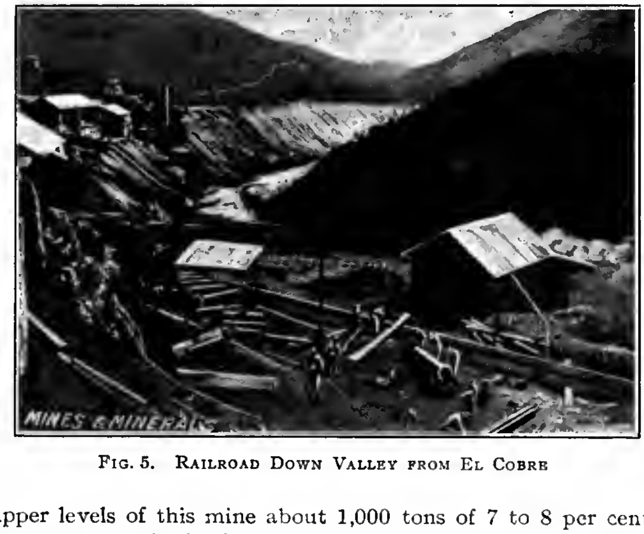


Fig. 5. Railroad Down Valley from El Cobre

upper levels of this mine about 1,000 tons of 7 to 8 per cent. copper was obtained per month. Considerable low-grade sulphide ore was found in the old stopes and faces. It is probable that this mine is on the same vein system as the Hardy mine.

No. 1 shaft is a new opening on a body of oxidized ore that was uncovered by the recent development work, near the surface. The opening was 144 feet deep and dry, as it is in a continuation of one of the Hardy mine veins. About 300 tons

per month of from 11 to 13 per cent. copper ore was obtained from this opening.

During the time of the old operations pumping plants were at the Pozo Verde, Hardy, Richards, San Juan, Santiago, and other shafts. The Richards shaft was equipped with a heavy Cornish pump the ruins of which are shown in Fig. 11.

Through the Hardy shaft the veins on the east of the fault were reopened. What apparently here corresponds to the middle vein is associated with two other veins, see Fig. 8. Considerable 3 per cent. to 6½ per cent. copper ore was found in the mine from the fifth level down. Some of it was in pillars and some in the solid. Above this the veins were extensively stoped. The lower levels connecting with the Richards shaft in a roundabout way also showed good ore remaining near the fault. On the sixth level a rock tunnel was driven toward the Santiago mine. Several of the Hardy mine levels extended to the north vein. On the ninth and tenth levels openings to the north vein showed considerable ore assaying 5 per cent. to 9 per cent. copper; 10 per cent. to 17 per cent. iron; 17 per cent. sulphur, and from 6 per cent. to 13 per cent. lime. The ore produced from this shaft in exploration and mining amounted to about 400 tons of 6½ per cent. copper and 600 tons of 3 per cent. copper per month. The north vein approached closely to the middle vein on the lower levels, as shown in Figs. 9 and 10. Cross-cut tunnels to the north vein were opened at several places, and disclosed that the vein had been extensively mined and filled. Some stopes were open and heavily timbered, and good ore remained in occasional pillars or along the walls.

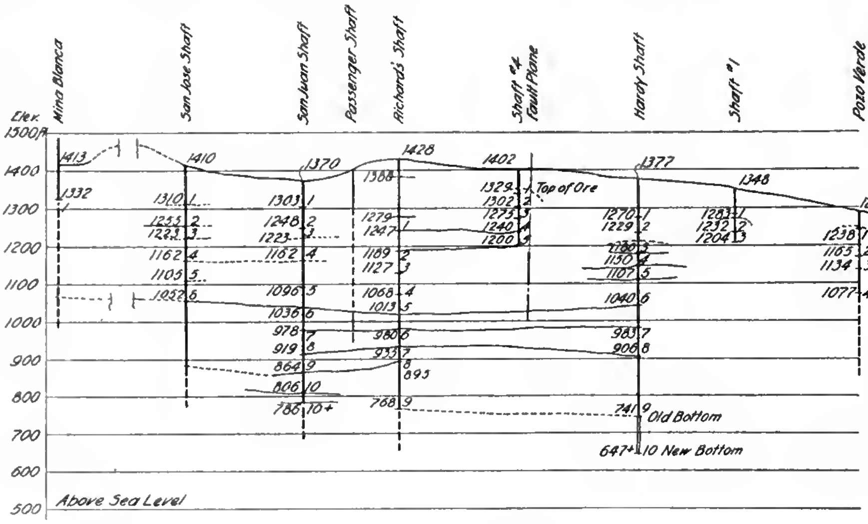


Fig. 6. Profile of Mine Shafts

No. 4 shaft was a new opening on a body of ore near the fault. This shaft is 200 feet deep and is connected with the north vein on the west side of the fault. The mine was dry and the best of the ore came from above the fourth level. The monthly output from here was about 1,600 tons of from 7 per cent. to 7½ per cent. ore and 700 tons of 3 per cent. ore.

The Richards shaft was equipped for hoisting some of the ore from the Hardy and San Juan workings as well as its own. The lower levels of this mine developed the north vein which was very close to the middle vein at this depth. Thick bodies of ore had been extensively mined in the past in the vicinity of this shaft and there was still considerable ore remaining. Some of the stopes were heavily timbered and others were filled.

The Arrieta shaft was not reopened but some of the levels of the Richards shaft extended in its direction. This was once an important mine as large bodies of ore had been stoped. It is located near the intersection of the north vein and the fault. A cave occurred here during the reopening which took in a large area of the surface near the stream and one of the buildings and the lumber in the yard, leaving a deep pit.

San Juan shaft was sunk in the early days on the middle vein, and as in the other cases mentioned, the ore was stoped pretty thoroughly in the upper workings. Good ore remained in the pillars in the lower levels, and high-grade ore was developed in the pillars in the lower levels, and high-grade ore was developed in new explorations. A large body of low-grade quartz sulphide ore was cross-cut in driving in the wall north of the vein on the lower level. This deposit lies between the middle and north veins and is developed for considerable distances to the east and west, and from 200 to 300 feet in height. This deposit was blocked out and as soon as it was removed the space excavated was timbered with square sets. The San Juan shaft was very wet and the water very acid. The monthly output was from 200 to 400 tons of from 9 to 11 per cent. copper ore, and a large tonnage of low-grade ore carrying up to 4 per cent. copper.

The exploration of the San Jose shaft workings was mostly carried on through the San Juan workings. The lower levels were reopened beyond the San Jose shaft toward the Mina Blanca shaft, and the vein was found to have been extensively mined in this direction. A quantity of good ore remained in the stopes and there was a satisfactory amount of unworked ground. Considerable cement copper was found in old Californias at this mine. The old operations, of which there are several on a continuation of these veins to the westward, were

Fig. 7. Punta Sal Smelter, Cuba

not reopened to any extent. The ores in this direction, according to old reports, are said to contain considerable zinc and gold.

The Trevence shaft, which is about 600 feet south of the San Jose shaft, was reopened a short distance and slightly explored. It is on a vein to the south of the middle vein, and cross-cuts were driven from the San Juan shaft in the direction of these workings.

The Jueves Santos mine is an old one some distance to the south on another vein system of some promise. It was reopened through an old drift where there had been some old shafts worked. A small amount of ore was produced here in the exploration of comparatively new ground.

The Santiago mine, which is on a parallel vein a short distance to the southeast of the Hardy shaft, was worked in the early days but probably not steadily toward the last. There was water in some of the workings which may have been difficult to handle. A few parts of what was then a new Cornish pump, Fig. 12, are lying on the ground ready for installation, apparently indicating an intent to further develop the property. A considerable amount of good ore has been obtained from some of these workings. In removing the pump columns from the Richards shaft and elsewhere, some cement copper was obtained where the acid water had attacked the pipe and precipitated copper thereon. Frequently old iron pipe and wire rope were found in the mines which had almost completely turned to copper. The result of the reopening to a depth of 800 feet continued to prove satisfactory, as the deeper and remote workings were penetrated and explored. As stated, the greatest depth attained by the old workings is reported to be 1,200 feet and further reports state that large bodies of low-grade ore, or mundic, as it is referred to, are said to have been opened up at this depth. The north vein is also said to unite with the middle vein and form large ore bodies. The shafts that attain a depth of 1,200 feet are probably the Richards, San Juan, and possibly some others. Among the fairly deep shafts are the Pozo Verde, Pozo Viejo, Hardy, Arrieto, Mahon, Richards, Påssenger, San Juan, San Jose, Mina Blanca, and some others to the west; also the Trevence and Santiago.

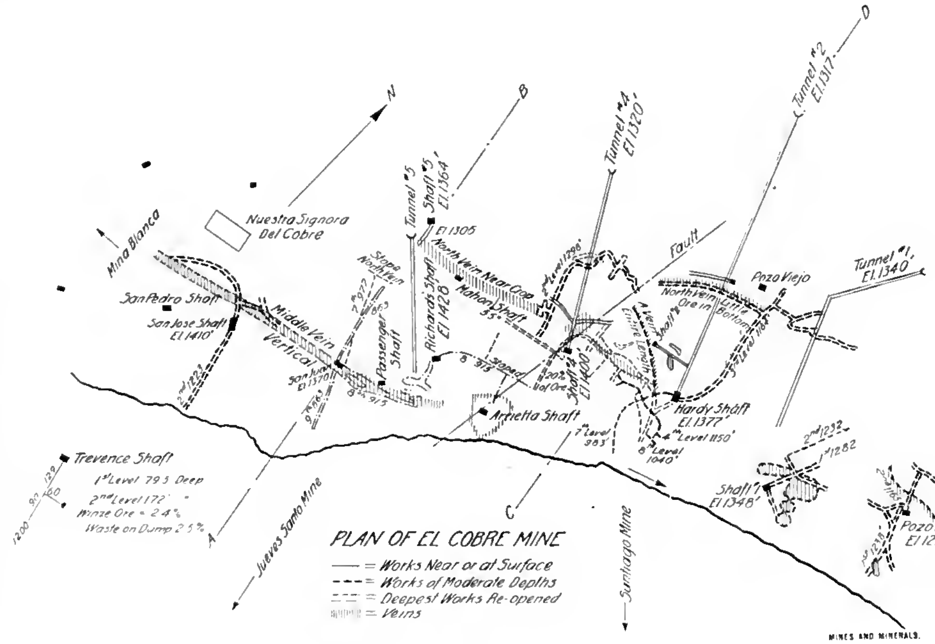


Fig. 8. Plan of El Cobre Mine

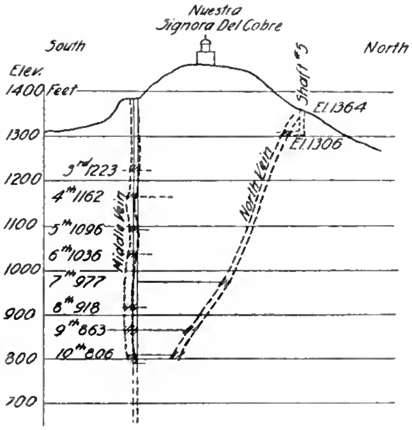


Fig. 9. Section on A-B, Fig. 8

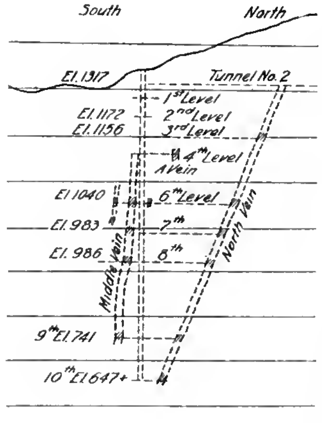


Fig. 10. Section on C-D, Fig 8

Operations were steadily improving when late in the year 1906 a cave-in of old workings occurred between the Hardy and the San Juan shafts. The cave extended from the lower levels to the surface and affected the shafts so that all mining work in this area was closed down and smelting operations at Punta Sal suspended. The extensively mined stopes in the old workings had become weakened through age, and the lowering of the water after so many years' standing probably had some effect thereon. Many of the stopes were opened for wide areas. Several of those that were filled caved during the period of reopening. The country is subject to frequent earthquakes and these occurrences have at times caused the stopes to cave. The explorations of the veins to the south of the old workings resulted in obtaining a large amount of ore for shipment after the old works caved in. Some very good ore is being obtained near or from the Santiago workings, so that the shipment of quite a large tonnage to the United States is kept up. The development in this section may result in some large operations in the future. It is probable that any exploration of the old workings would be better carried on in a shaft sunk in more secure ground to one side of the old veins, or at some distance to the east or west near or on the vein. In the event of such an undertaking it would be preferable to sink deep enough to reach the solid ore below the bottom of the old workings. Cross-cuts might be driven from such a shaft to any of the levels of the old workings if such was warranted.

A most interesting feature of El Cobre and one that has made it widely known throughout the world, is the Sanctuary of the Virgin of Cobre. This stands upon the very apex of the Cobre vein, and its pink dome and square blue towers may be seen from any point in the mountains for miles around. The incident of its erection forms a part of the history of the mine.

Alonzo Ojeda, from San Domingo, wrecked upon the shores of Cuba early in the fifteenth century, was saved from death by an Indian chief, Cacique by name. Ojeda had with him an image of the Virgin, and, in fulfilment of a vow conditional on saving of his life, built a chapel to her in the town of Cuieba. This was the first christian chapel in the island of Cuba. The Indian chief, however, stole the image to try the Lady's charm against his enemies, and it is said that so great was her power that he never lost a battle while he had the image in his possession

In 1660, so legend has it, Juan de Joyos and Rodrigo de Joyos, Indians, and Juan Moreno, a negro, found the image floating in the bay. On a board upon the image was written: "Yo soy la Virgin de la Caridad"—"I am Our Lady of Charity." These three men took the image to Hato, where the mayor erected an altar for her, at the same time notifying the manager of the Royal Mines at El Cobre, Don Francisco Sanchez de Moya, that he had done so. Francisco built a chapel for the Virgin, first at Hato, and afterwards she was brought to Cobre, and a chapel was built for her in the town. As the mine prospered, Our Lady of Caridad became, as it were, the patron saint of the mine, and the sanctuary was built for her protection on the present site. It is a handsome building, 150 ft. × 90 ft., supported by massive columns, and is reached by 430 steps from the village below. Up these steps on their knees the pilgrims passed in thousands to pay their "promesas" to the Mother of Charity, and many wonderful cures of the sick are mentioned. Back of the chancel is a pile of crutches and canes, testifying to the fact that cripples had left the sanctuary without these aids to locomotion.

Fig. 11. Old Cornish Pump—El Cobre

Heroic figures of the saints stood on either side of the entrance to guard the Virgin, who, in her cloth of gold, stood on the altar under the great dome. The figure is about the size of a child's doll and holds a small image of the Holy Infant in her left hand who in turn holds a sphere symbolizing the world. The right hand of the image is raised in benediction.

This celebrated shrine was on the ground that caved in 1906, and while the sanctuary still stands Nuestra Signora del Cobre has been placed in the El Cobre railway station.

The following table gives the average analysis and kinds of ore obtained in the various openings of El Cobre:

Name of Mine	Kind of Ore	Analyses			
		Cu Per Cent.	*FeO* Per Cent.	*SeO*² Per Cent	*S* Per Cent.
Pozo Verde	Sulphides	8.00	22.00	42.80	17.40
Shaft No. 1	Oxides	13.80	11.10	67.70	2.50
Hardy shaft	Sulphides	6.45	15.40	58.50	13.70
	Sulphides, 3d class	3.16	11.60	69.70	8.60
Shaft No. 4	Sulphides	7.50	10.85	63.30	11.00
	Sulphides, 3d class	3.28	11.00	72.40	5.50
Richards					
San Juan	Sulphides	9.35	21.95	47.60	16.95
	Sulphides, 3d class	4.54	11.30	63.90	0.15
San Jose	Sulphides	10.00	10.90	65.35	10.00
	Sulphides, 3d calss				
Jueves Santo	Sulphides	6.00	10.10	91.10	9. 0
Mina Blanca	Sulphides, 2d class				
Tunnel No 2	Oxides				

PHOSPHATE IN OCEANIA

The island of Makatea, which is situated in the extreme northwestern part of the extensive Tuamotu Archipelago, has an area of about 10 square miles, one-fifth of which is covered with deposits of phosphate estimated at 10,000,000 tons, of an average grade of 82 per cent. So far as is known here nothing further has been done to develop the phosphate discovered on Henderson (or Elizabeth) Island, a British possession lying east of the Society Islands. As that island is of an elevated coralliferous limestone formation like Makatea, Ocean, Pleasant and Angaur, it is highly probable that further investigations will show conclusively that the deposits on Henderson are large and valuable.

The world's annual production of phosphate rock is about 5,000,000 tons, the United States being the largest producer, with an annual output of more than 2,000,000 tons. Tunis, which ranks second, produced phosphate of a rather low grade to the amount of $6,117,000 in 1908. In 1909 the great Gafsa Company, which owns its own railroad, mined 907,000 metric tons (metric ton = 2,204.6 pounds). It pays a large dividend on its capital of $7,750,000, as is shown by the fact that the stock of this Tunisian company is selling in Paris at a premium of more than 600 per cent. The Pacific Phosphate Co., of London, which owns deposits of 50,000,000 tons of high-grade phosphate on Ocean and Pleasant islands, is mining some 250,000 long tons a year at a profit of more than 50 per cent. on its capital stock of $1,216,600. A German company has recently begun to mine phosphate to a considerable extent on the island of Angaur, which lies in the western part of the Carolines at no great distance from the Philippines.—*United States Consular Report.*

ACTION OF *CO* AND *H*, OR CO_2 AND *H*, ON IRON OXIDES

A mixture of three volumes of *CO* and one volume *H* was passed at 300° C. over ferroso-ferric oxide. The product contained about 7 per cent. carbon, the remaining 93 per cent. being about one-half ferrous oxide and one-half iron carbide. When steam was passed over this at 400° C., a gas was obtained containing 96 per cent. *H* and 4 per cent. CH_4. If two volumes CO_2 and 1 volume *H* were passed over iron at 1250° C. the issuing gas consisted of 23 per cent. *CO*, 76 per cent. *H* and .15 per cent. CH_4. A mixture of equal volumes of *CO* and *H*, saturated with H_2O vapor was passed over Fe_2O_3 at 250° – 300° C. and the resulting gas was heated to 500° or 600° C. Besides permanent gases a small proportion of a substance condensible to a jelly-like solid, like vaseline, was obtained. These experiments may have a bearing on the geological formation of hydrocarbon gases and petroleum. A. Gautier and P. Clausman. (Comptes rendus, cli, 355.)

Fig. 12. Parts of Cornish Pump

CANADIAN IRON ORE INDUSTRY

Written for Mines and Minerals

A Description of the Principal Iron Deposits of the Province of Ontario

The Canadian Department of Mines, under the direction of Eugene Haanel, Ph. D., issues from time to time advanced chapters of the Annual Report of the Bureau of Mines. Bulletin No. 79, by John McLeish, B. A., chief of the Division of Mineral Resources and Statistics, contains much interesting information for those engaged in the iron and steel industry as well as for laymen. The information in the following article has been abstracted from the Annual Report of the Bureau of Mines, Ontario, and from Bulletin No. 79.

Although iron ores are of wide occurrence throughout Canada, their development has not kept pace with the growth of the manufacture of pig iron and steel, a feature of which reminds one of putting the cart before the horse, were it not that the transportation facilities in Canada must be considered before conclusions are reached. In 1909 the iron mines of Canada produced 268,043 net tons of iron ore, of which number 21,956 tons were exported, and during the same period 1,235,000 tons or 83 per cent. of the ore used in making was imported.

Most of the imported ore came from Newfoundland and the south shore of Lake Superior, although it is probable that the Thunder Bay district in the vicinity of Lake Nipigon is abundantly supplied with iron ores. Heretofore Nova Scotia has been the chief pig-iron producer of the Dominion, but in 1909 Ontario took the lead.

The Oranian iron range is 23 miles northeast from Humbolt Bay in Lake Nipigon.* According to E. S. Moore, the iron formation extends east and west in two bands about 2 miles apart. The iron ore deposits in this district are lean magnetites of the following average analysis taken from specimens: Metallic iron, 50.52 per cent.; ferrous iron, 17.06 per cent.; silica, 26.85 per cent.

The Lake Savant iron range is northwest from Lake Nipigon, and although extensive has had no pay ore located up to this time. The ore is a lean magnetite that contains from 35 to 40 per cent. iron when free from schist.

The Moose Mountain mine is located 25 miles north of Sudbury, in Hutton township. The ore occurs in the lowest Pre-Cambrian rocks termed "Keewatin," which consist of banded iron formation, greywacke and fine-grained gray gneiss, rhyolites, quartz-porphyries, greenstones and green schists. The ore is fine-grained magnetite, hard but comparatively easy to crush. The property is known to contain four different deposits, with probably different analyses, as is usual in such magmas; however, No. 1 deposit is the one at present worked.

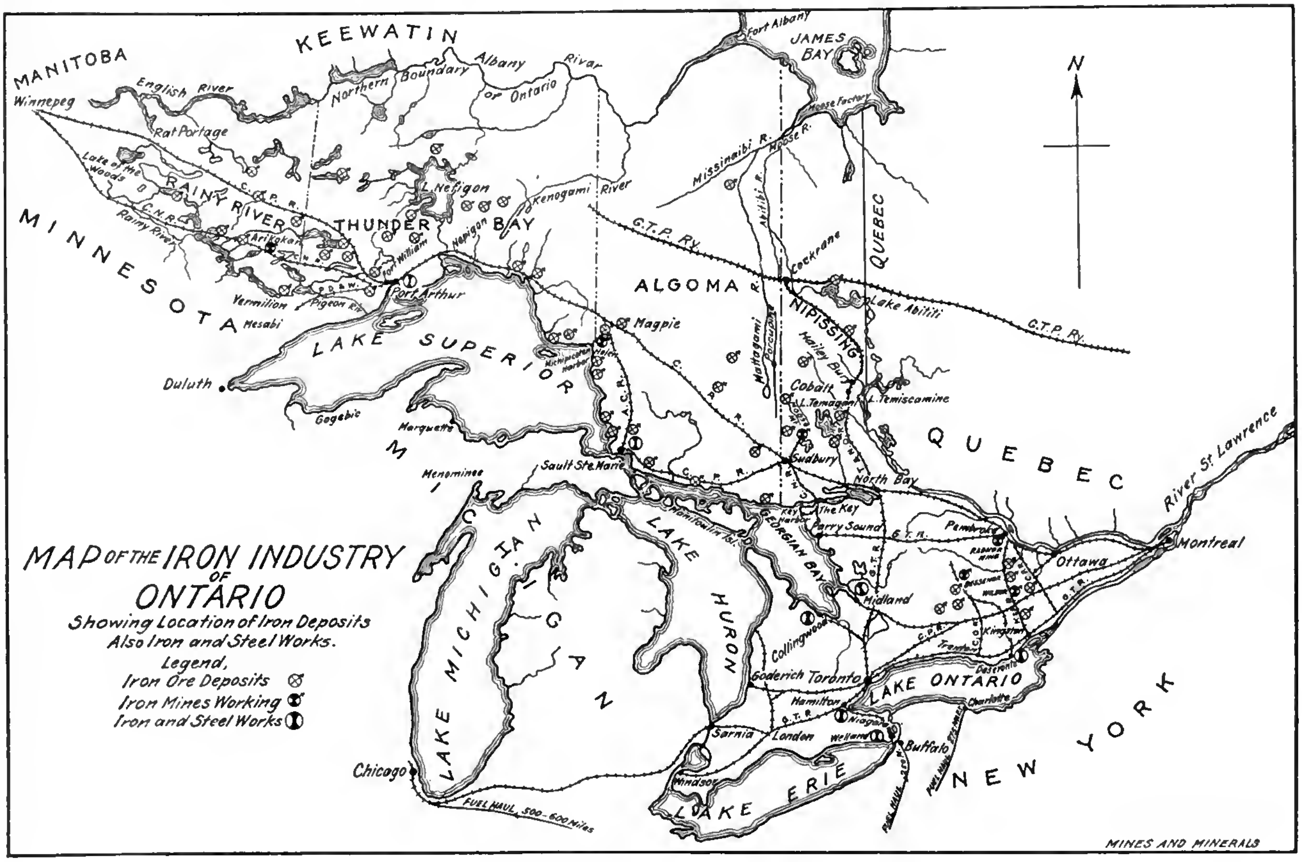


Fig. 1. Map of Ontario, Canada

Nos. 3 and 4 are said to be equally as good although not so well known, while No. 2 is low grade and will require concentration. No. 1 deposit, which is 500 feet long by 150 feet wide, is 210 feet horizontally and 140 feet vertically above the railroad track, as shown in Fig. 2. The ore is mined open-cut, consequently no shipments are made during the freezing months; in the warmer months the ore is shipped 80 miles over the Canadian Northern Railway to docks at Key Harbor, on Georgian Bay. As the ore is broken from the 65-foot stope it falls to the level, where it is loaded in mine cars, trammed to a 30-degree pocket chute *a* and dumped. From the large pocket the ore is fed to the large gyratory crusher *b*, which reduces the pieces to 6-inch diameter and passes it on to the 48″ × 12″ rotary screen *c* having ½-inch perforations. All ore that passes through the screen falls to the elevator boot *d*, but all ore that passes over the screen is fed to the gyratory crusher *e* and is reduced to 1-inch diameter before discharging in the boot. The elevator *f* raises

* Eighteenth Bureau Mines Report, page 196.

the ore in 14″×30″ buckets to the storage bin *g* of 250 tons capacity, from which it is loaded into drop-bottom steel railroad cars and transported to the ore docks at Key Harbor. At the docks the cars are dumped from a trestle on a stock pile, underneath which and in center line of the trestle there is a tunnel through which a 42-inch traveling belt moves and conveys the ore to a similar belt at the docks.

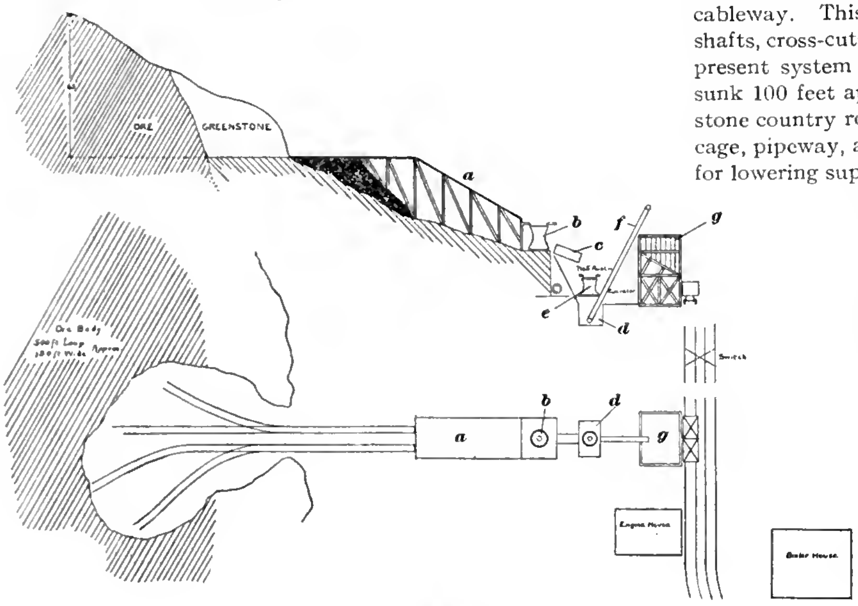


Fig. 2. Moose Mountain Mine

The second belt elevates the ore to the dock trestle 60 feet above water level, but before the ore is tripped off the belt into the ore pockets it is automatically weighed. It is expected that the belts will have a capacity of 800 tons per hour, and also that from the pockets, which have 8,000 tons capacity, ore can be loaded into the holds of vessels at the same rate. As stated, the ore is magnetite, and it may be added that it is of non-Bessemer standard, it being sold on the following guaranteed analysis: Iron, 55.50; silica, 13.29; alumina, 1.21; phosphorus, .10; manganese, .02; lime, 3.60; magnesia, 3.15; sulphur, .011; moisture, 1.

Another producing Canadian iron mine, known as the Helen, is situated 12 miles from Michipicoten Harbor and is connected with that shipping point by a branch line of the Algoma & Hudson Bay Railway. This is the largest iron mine in Canada, as it is capable of producing 1,000 tons of hematite and limonite daily. The first grade ore is a hard, compact, red hematite, 60 per cent. iron or over; the second grade is a porous but hard brown limonite, 57 to 58 per cent. iron; the third grade is a soft brown limonite, 53 to 54 per cent. iron. Samples of the ore taken from the stock pile at a furnace and said to be representative, gave the following analysis: Iron, 57.100; silica, 5.270; alumina, 1.040; phosphorus, .128; manganese, .038; lime, .200; magnesia, .100; sulphur, .053; moisture, 3.410.

Were it not that the presence of phosphorus exceeded the Bessemer limit, which in this ore is $\left(\frac{57.10}{1,000}=\right).057$, the ore could be classed with the best of Lake Superior ores as it is of such composition as to admit of rapid reduction in the blast furnace. The iron formation of the Michipicoten district in which the Helen mine is situated consists of ferruginous cherts, banded cherts and iron ore, and cherty siderites, which contain the iron ore proper. The formation, which is 1¾ miles long and 900 feet wide, strikes east and west and lies in greenish or brownish schists belonging mostly to the lower Huronian. The main ore body extends 1,400 feet eastward from the eastern end of Boyer Lake and has an average width of 400 feet. The highest point at which the ore is found is 100 feet above the lake, and as drill holes have proved the ore to a depth of 188 feet below the lake level, the total thickness is approximately 288 feet.

During early mining the ore was worked as an open cut, broken down by heavy blasts, loaded with steam shovel, and where inaccessible to the latter it was handled by a Lidgerwood cableway. This system of mining was abandoned in favor of shafts, cross-cuts, and stopes. In Fig. 3 is shown a plan of the present system of working. The shafts No. 1 and No. 2 are sunk 100 feet apart, to the south of the ore body in the greenstone country rock. No. 1 is a double-compartment shaft with cage, pipeway, and ladderway. It is used mostly for a manway; for lowering supplies and for hoisting waste rock. It is 450 feet deep. No. 2 shaft has two compartments that are used exclusively for hoisting ore. Both shafts are connected at the bottom by a drift 100 feet long, and from the bottom of the No. 2 shaft a cross-cut *a* is driven 150 feet and continued through the deposit to the northern boundary of merchantable ore, and hence 500 feet a main drift *b* follows the ore to its eastern boundary. From the main drift the levels *c*, *d*, *e*, *f*, etc. are branched out every 50 feet, running to the boundaries of the deposit in each case. Risers are made in these levels every 40 feet and are driven to within 20 feet of the corresponding level above, after which chutes are put in the bottom of each riser as shown in Fig. 4.

The stopes, which are not more than 60 feet high, are worked overhand from the boundaries of the ore body back toward the main level. Boyer Lake is pumped out adjacent to the mine to prevent the water seeping into the workings and necessitating its being pumped to the surface; however, it requires five pumps inside to keep the mine free from water.

Ore is raised in 4-ton skips and dumped directly into a large gyratory crusher capable of breaking 2,500 tons per day to a diameter of 6 inches. When navigation closes, the development work is carried on underground; for instance, new levels will be driven at greater depth and whatever ore is stoped is raised to higher worked-out levels, where it accumulates without freezing and can be drawn off as soon as navigation opens.

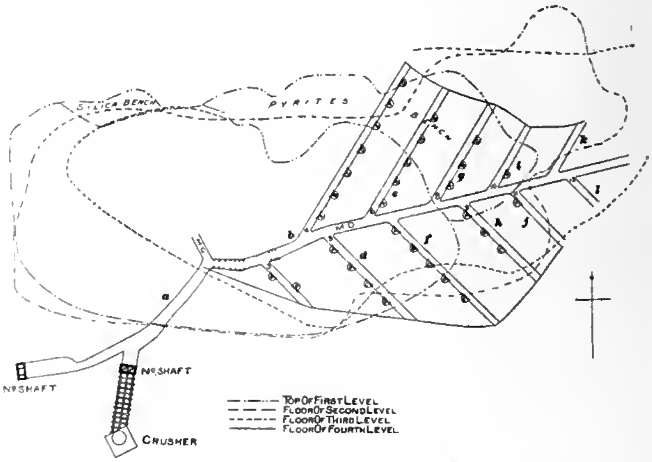


Fig. 3. Plan of Helen Iron Mine

In one instance, a drift run on the third level through a dike found a body of good Bessemer ore; in another case, a drift from the easterly limit of the ore on the fifth level encountered a large body of pyrite, and a large tonnage of this ore has been blocked out for future shipments.

The surface plant at the Helen mine is driven by electric power from the Algoma Power Co., at High Falls, on the Michipicoten

River. An 80-horsepower motor drives No. 1 hoister; a 150-horsepower motor drives No. 2 shaft hoister; a 200-horsepower motor drives a 12-drill compressor; a 100-horsepower motor drives a turbine pump having a capacity of 500 gallons per minute against a head of 400 feet; and an 80-horsepower motor drives the ore crusher. The power is brought to the mines under a pressure of 10,000 volts and is stepped down at the transformer house at the mine to 550 volts. The steam plant is kept in reserve in case there is a break in the power plant. The Lake Superior Power Co., of Sault Ste. Marie, Ont., which owns the Helen mine, is also developing a mine called the "Magpie," 15 miles northeast on the Algoma Central Railroad.

The iron-ore industry in Canada has been encouraged by a bounty and so much as $1.70 per ton was paid for pig iron manufactured from Canadian ores. In the table following the production of iron ores in net tons for 1909 is given:

Kind of Ore	Net Tons 2,000 pounds	Total Value Dollars	Value Per Ton Dollars
Magnetite Fe_3O_4........	74,240	$162,280	$2.19
Hematite Fe_2O_3........	190,473	492,348	2.58
Limonite $Fe_2O_3.3H_2O$..	3,330	4,688	1.41
Totals...............	268,043	$659,316	$2.46

The Atikokan Iron Co., Ltd., of Port Arthur, owns an iron mine 132 miles to the west, near the Canadian Northern Railway. The ores occur in a ridge of chloritic schist striking east and west across a swamp for a distance of 4,000 feet. Parallel bodies of magnetite standing almost vertical can be traced along this ridge.

The ore carries 60 per cent. metallic iron, but is unfortunately mixed with pyrrhotite and pyrite to such an extent that the run-of-mine ore will average 2.5 per cent. sulphur. Like most magnetic iron magmas, the ore carries phosphorus to such an extent it is non-Bessemer; in this case $\left(\frac{62.14}{1,000}=\right).062$ per cent. phosphorus is the limit, whereas from the analysis of the run-of-mine ore given below it carries .12 per cent. phosphorus. Analysis of the Atikokan run-of-mine ore: Iron, 62.14; silica, 7.64; alumina, .75; sulphur, 1.40; phosphorus, .12; manganese, .09; magnesia, 2.18; lime, 2.54. In 1909, 150 tons of ore were shipped daily from this mine to Port Arthur, where it was

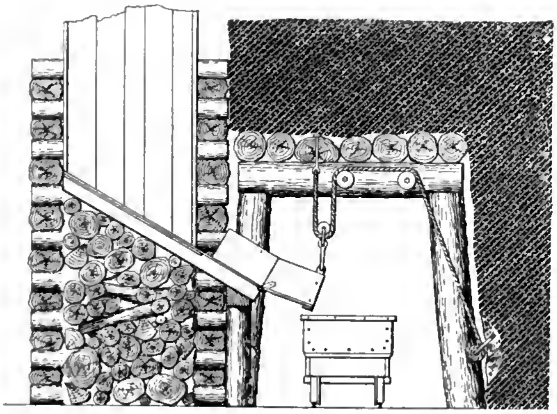


FIG. 4. LOADING CHUTE AT BOTTOM OF RISER

roasted by the waste gases from the blast furnaces of the same company. The ore bodies, three in number, are cross-cut by a tunnel 6 feet to 12 feet wide driven through the hill. The deposit *a*, Fig. 5, is 40 feet wide; deposit *b*, 10 feet wide; and deposit *c*, 16 feet wide. Deposit *a*, the only one worked at present, is developed by "risers," started 85 feet to the east from the tunnel and 65 feet to the west, the object being to understope the ore back toward the tunnel after the risers are holed through to the surface. When ore is broken it is trammed out of the tunnel and fed to a gyratory crusher having a capacity of 50 tons per hour. After crushing, the ore is delivered to 12″×18″ elevator buckets that raise it to a sufficient height to fall by gravity into 50-ton drop-bottom steel cars. No storage pockets are used, but a string of empty cars are kept spotted on the mine siding and allowed to run by gravity to the elevator as required.

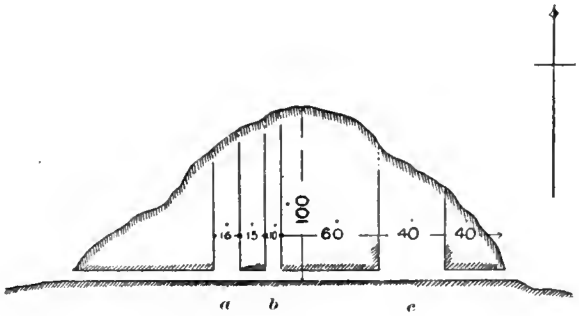


FIG. 5. CROSS SECTION OF ATIKOKAN IRON MINE

In Eastern Ontario several magnetite mines are opened. The Canada Iron Corporation, Ltd., of Montreal, operates the Mayo mines, at Bessemer, in Hastings County, 5½ miles east from L'Amable station on the Central Ontario Railway,

There are five ore deposits opened at Bessemer, which strike northeast and southwest, the usual strike of New York state, and New Jersey magnetites. They dip 60 degrees to the south, and lie in diorite schists and greenstones with one or two intrusions of gneissoid granite with crystalline limestone on the foot-wall. In this respect they differ from the New York and New Jersey magnetites, which are in gneissoid granite with no limestone nearby. The Franklinite deposit of New Jersey is in white crystalline limestone, but with this exception there is probably no other location where magnetite is associated with calcite in the states mentioned. The ore at the Mayo mines is fine grained and will, if cobbed, average 60 per cent. iron. Phosphorus is below the Bessemer limit and the sulphur, which occurs in small stringers of iron pyrite, can be separated by hand picking. An average analysis of the shipping ore from No. 4 opening is as follows: Iron, 54.29; silica, 9.84; alumina, 2.02; phosphorus, .019; manganese, .38; lime, 6.86; magnesia, 1.35; sulphur, .062.

At No. 3 mine the open cut is 36 feet deep and 80 feet square in an ore body approximately 300 feet deep and 800 feet long. The ore from this pit is not all high grade, the proportion being 35 tons high grade to 50 tons low grade. No. 4 open cut, shown in Fig. 6, is 55 feet deep and 500 feet long. At the west end of this cut an 8′×14′ incline follows the dip of the ore. The shaft house shown in Fig. 7 is placed near the shaft collar to receive the ore from the hoisting skip. In order to make a uniform product for the blast furnace the ore is crushed before loading on the cars. It has been found from experience that magnetite is somewhat more difficult to smelt than hematite; however, the No. 4 ore should be more readily smelted than hematite. Magnetite produces less flue dust in the blast furnace than hematite and the best results will be obtained by a mixture of the two ores. The No. 4 ore body has been traced 1,800 feet on the surface and is supposed to be 55 feet wide, between walls. Forty feet of the width is high in iron, the remaining 15 feet is lower in iron and high in sulphur. In this ore bed the pyrite is not spangled through the ore, but is found in fairly large masses, and along the contact of the ore with the country rock.

At the Child's mine 2½ miles east of Bessemer there is an immense body of low-grade magnetite averaging 40 per cent. iron, with both phosphorus and sulphur within Bessemer limits. The gangue of this ore consists of hornblende and calcite with a varying proportion of silica.

The Radnor mine, in Grattan Township, Renfrew County, has been one of the most important iron-ore producers in Eastern Ontario. The ore, which is a coarse crystallized magnetite, occurs in beds from 4 to 10 feet wide, that strike northwest and southeast, with an average dip of 34 degrees. Assorted ore from this deposit carries about 50 per cent. metallic iron, the tailing about 30 per cent.

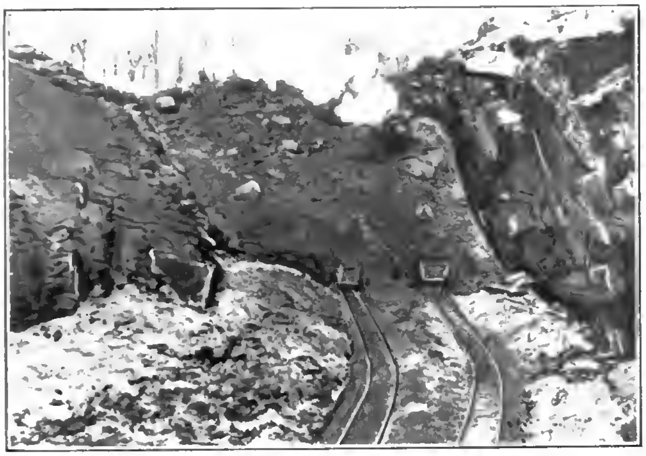


Fig. 6. Open Pit, Mayo Iron Mine No. 4

Another mine in Renfrew County is in Lavant Township, about 60 miles north of Kingston, on the Kingston & Pembroke Railway. This deposit of magnetite, known as the Wilbur mine, is found in detached masses at the contact of crystalline limestone and gneissoid granite, the limestone being invariably in the foot-wall. The strike is a little east of north, with the ore dipping at an angle of 27 degrees to the east. One peculiarity of this deposit is that while the run-of-mine ore carries 50 per cent. iron, the ore is seamed with chlorite and calcite, making it evidently a deposit of second concentration. An analysis of the shipping ore is given as follows: Iron, 55.490; insoluble, 3.450; sulphur, .040; phosphorus, .022.

There are eight openings on the property, the most important being where the ore is mined and hoisted through inclined shafts. Other ore deposits along the Kingston & Pembroke Railway that have been worked intermittently are the Calabogie, Robertsville, and Glendower. In the vicinity of the Central Ontario Railway there are, besides the mines at Bessemer and Mayo, the Belmont, Blairton, and Coe Hill mines.

AN ACCIDENTAL TAYLOR AIR COMPRESSOR

In a recent number of *Science*, E. H. Sellards describes a "Spouting Well" which is of interest as showing how there has been unintentionally constructed what is practically a Taylor air compressor, similar in principle to the large plant at Ragged Chutes, Canada, described in the October, 1910, number of Mines and Minerals. A similar "blowing" of the compressor is there described and a photograph shown. The following is a description of the well:

In parts of Central Florida bored wells are somewhat extensively used for drainage purposes. The wells are drilled through the superficial material and as a rule enter the Vicksburg limestone of Oligocene age, although other porous limestones may serve the same purpose. Many of the wells terminate in cavities in the limestone, while others reach layers of shell or other porous material. Surface water entering the wells is carried into the limestone formation. In some localities in the central part of the state these wells have been found efficient in carrying off surface water and in draining small marsh areas.

One of these drainage wells near Orlando, in Orange County, recently developed the unusual phenomenon of spouting. The well was drilled, in 1907, near the edge of a small lake. It is 12 inches in diameter, has a total depth of 260 feet, is cased 60 feet, and terminates in a cavity in the limestone. The level of permanent underground water at this locality is 33 feet from the surface. The well is intended, by carrying off the surplus water, to prevent the lake from rising above a given level, since to do so would flood the farming land.

When first seen by the writer the water of the lake stood a few inches above the level of the top of the pipe, and the well was receiving water at much less than its full capacity. At intervals of a few minutes the well would reverse itself and spout, throwing a column of water into the air. The spouting comes on gradually. First, the well ceases to receive water and begins bubbling; the column of water follows, rising with considerable force to a height of 20 feet or more above the surface, the spout occurring with tolerable regularity at intervals of 4 minutes. The manager of the farm states, however, that the interval between spouts varies from 2 to 15 minutes.

Although drilled 3 years ago and receiving water more or less continuously during that time, the phenomenon of spouting developed for the first time in September, 1910. The well continued spouting without interruption for a little more than a week and until shut off by the owner.

At this stage of the lake the well is receiving water at less than its full carrying capacity and as the water enters the vertical pipe it forms a suction, carrying a large amount of air into the well, which doubtless collects in a chamber or cavity along the side or at the bottom of the well. As the well continues receiving water the air accumulates under pressure in this chamber until ultimately the pressure under which the air is confined is sufficient to overcome the weight of the overlying water plus the inertia of movement and hence rushes out with considerable force, carrying the column of water with it. The fact that the well when first drilled did not spout and afterwards began spouting indicates that the essential conditions were subsequently developed either by caving or by other changes in the underground conditions.

GRAHAMITE, A SOLID NATIVE BITUMEN

Grahamite was discovered in West Virginia, in about 1860, but there is considerable confusion in literature as to its

Fig. 7. Mineral Range Iron Mining Co., Shaft and Crusher House

distinction from albertite and gilsonite and to a certain extent from manjak. It may be described as a brittle, solid native bitumen, generally pure, but at times mixed with adventitious mineral matter, characterized when pure by a peculiar schistose fracture, known as "hackly." It does not melt, but intumesces; it is soluble in carbon bisulphide and to a small extent in light naptha and yields a high percentage of coke. It closely resembles the "asphaltenes."

TRANSVAAL DRILL COMPETITION, 1909*

During the last 15 years there have been several attempts made on the Rand to determine the efficiency of rock drilling machines, in order to decide their relative merits, to establish the actual cost of machine drilling and compare the costs of of machine drilling with those of hand drilling.

Report of the Committee On the Results Attained by the Competing Drills

In February, 1898, Major L. I. Seymour conducted a test with 31 reciprocating machines varying from $2\frac{1}{2}$ inches to $3\frac{5}{8}$ inches in diameter, averaging 30.9 machines of $3\frac{1}{4}$ inches diameter. On ordinary work they averaged 26 feet 10.4 inches per machine in 6 hours, or .9 inch per minute. The volume of free air used per minute per $3\frac{1}{4}$-inch diameter machine was 81.08 cubic feet, compressed to 69.83 pounds per square inch. The test was for 6 hours; the number of feet of holes drilled by one machine 51 feet 9 inches (9 holes); average, 1.72 inches per minute. The free air used per inch of hole, 90.5 cubic feet, or 1,086 cubic feet per foot drilled.

The next trial was made in 1904 by Messrs. Goffe, Carper, and Docharty. The holes were drilled vertically and great care taken to ascertain the quantity of air consumed by different makes and sizes of air drills.

In these tests there was 105.2 cubic feet of free air required at 70-pound pressure to run a $3\frac{1}{4}$-inch drill 1 minute; 95.6 cubic feet for a 3-inch drill; 88.1 cubic feet for a $2\frac{3}{4}$-inch drill; and 68.2 cubic feet for a $2\frac{1}{4}$-inch drill.

These tests indicated that a small machine $2\frac{3}{4}$ inches in diameter using $3\frac{1}{16}$-inch bit drilled more hole per cubic foot of free air than a larger drill; and also that the cubic feet of free air used per inch drilled was fairly constant at all temperatures.

In 1907 the *South African Mining Journal* inaugurated a competition to test the relative merits of small machines for stoping purposes.

The trials were made on the surface and underground. A report of this test appears in Volume 6, No. 7, of the Journal of the Transvaal Institute of Mechanical Engineers, part of which was abstracted and printed in Volume 30, page 385, Mines and Minerals. At the surface, the Gordon hammer drill with 50 pounds of air pressure per square inch, drilled 1.42 inches per minute of total time, or 1.62 inches per minute of actual drilling time. At 60 pounds pressure 1.8 inches per minute were drilled in the total time and 2.13 inches per minute in the actual drilling time. Underground the Gordon hammer drill, which won the competition, drilled dry holes at the rate of 1.74 inches per minute of total time and wet holes at the rate of 1.82 inches per minute. This drill weighed $72\frac{3}{4}$ pounds, was $1\frac{13}{16}$ inches in diameter, and had a 10-inch stroke. The benefit to be derived by using high air pressure was evident from this trial. Following the trials, the Gordon drill in actual service underground was found to be unreliable. The pistons and anvil blocks had but a short life and the drill used frequently broke or stuck in the holes, and the machine entirely failed to fulfil the expectations of its promoters. As the mining industry on the Rand was expanding and the supply of unskilled labor for hand drilling did not keep pace, the necessity for a good stoping drill was as evident as before.

* Published by Committee, Johannesburg, 1910. Price, 7s 6d.

In February, 1908, the Chamber of Mines and the Transvaal Government jointly offered a prize amounting to £7,500 for the best drilling machine that could be produced for mine work on the Witwatersrand. There were 23 entries for this prize, 19 of which started in the competition.

In the elimination trials, which were carried out on the surface at the Transvaal University College and underground at the Ferreira Deep, Ltd., nine drills were eliminated and 10 entered for the competition of 300 shifts. These were the Holman $2\frac{1}{8}$ inch, Holman $2\frac{3}{4}$ inch, Siskol, Climax Imperial, New Century 00, Konomax, Chersen, Waugh, Murphy, Westfalia.

In the surface elimination test the most rapid rate of drilling was accomplished by the Westfalia machines; namely, 4.996 inches per minute of actual drilling time. The largest air consumption recorded was that of one of the Konomax machines; namely, 125.9 cubic feet of free air per foot drilled.

Fig. 1. Little Holman Drill in 18-inch Slope]!

In the underground elimination air trials the quickest drilling speeds were attained by the Chersen and Holman $2\frac{3}{4}$-inch machines. The former drilled 1.81 inches per minute over drilling and changing time and 1.56 inches per minute over total time, which consisted of three 8-hour shifts. The figures recorded for the Holman $2\frac{3}{4}$-inch machine on this trial were 1.94 and 1.47 inches per minute, respectively.

As the competition proceeded the following machines withdrew: Climax Imperial, Konomax, Murphy, Waugh, and Westfalia. It was the intention to run 300 shifts, but owing to lack of air pressure, 215 shifts were run. The stopes drilled in were 24 to 45 inches wide. The number of feet drilled by each pair of the four leading competing machines were as follows: Holman $2\frac{1}{8}$-inch, 12,779 feet; Siskol, 14,083 feet; Holman $2\frac{3}{4}$-inch, 11,744 feet; Chersen, 11,781 feet. This was drilled in 215 shifts of 8 hours each.

The drilling speeds attained over total times by the four machines are as follows:

Holman, $2\frac{1}{8}$ inches	.742 inch per minute
Siskol	.818 inch per minute
Holman, $2\frac{3}{4}$ inches	.682 inch per minute
Chersen	.684 inch per minute

At the surface elimination trials the average rate of drilling of each pair of machines was as follows:

Machine	Total Time Inches Per Minute	Actual Drilling Time Inches Per Minute
Holman $2\frac{1}{8}$ inches	1.566	2.393
Siskol	2.058	4.337
Holman $2\frac{3}{4}$ inches	1.988	3.110
Chersen	2.515	4.110

The cost of drilling per foot with these four leading machines amounted to: Holman $2\frac{1}{8}$-inch, 9.77d.; Siskol, 9.90d.; Holman $2\frac{3}{4}$-inch, 10.91d.; Chersen, 11.94d. These figures compare favorably with the cost of hand drilling on the Rand.

At the underground elimination trials the rate of drilling was as follows:

Machine	Total Time Inches Per Minute	Over Drilling and Changing Time Inches Per Minute
Holman 2⅛ inches	1.24	1.79
Siskol	.89	1.28
Holman 2¾ inches	1.47	1.94
Chersen	1.56	1.81

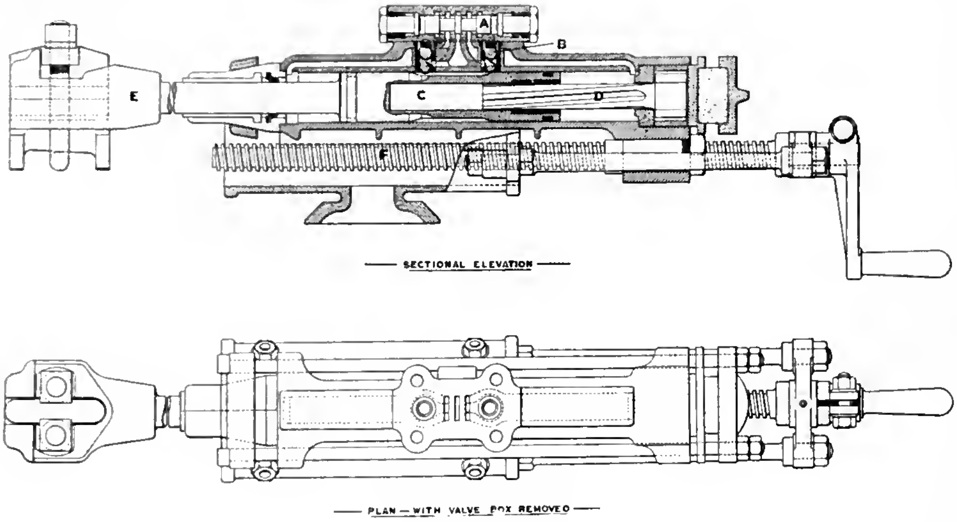


Fig. 2. The Holman 2⅛-in. Stoping Air-Valve Drill

The points to be noted in connection with the four machines are as follows:

The Holman 2⅛-inch seemed a little too small, but it drilled well, was easy to handle, and its maintenance was the lowest of any machine, which fact largely assisted in keeping it in the leading position. This machine, shown in Fig. 2, has a working stroke of 4½ inches. The piston valve is air thrown and is operated by the four steel balls *B* acting as auxiliary valves. The lower ball of each pair is worked by the piston and this actuates the upper pair. The machine worked consistently throughout the trials. Its maintenance cost was low, but its drilling speed was not so high as the Siskol. If the diameter of the cylinder and the length of the piston stroke had been rather more, there is no doubt but that this machine would have exceeded its performance. The miners liked the machine and found that their boys could handle it with ease. Fairly good deep holes could be drilled with it and good arrangements were made for cradle adjustments.

The Siskol machine is shown in Fig. 3. This machine is of the reciprocating type; the diameter of the cylinder is 2⅝ inches, and the stroke of the piston is 7 inches, but its total working stroke probably averages about 6 inches. The valve *A* is of the piston type, operated by air from the main cylinder. The valve chest is fitted on the top of the cylinder and is easily removable. The rotation gear is of the ordinary type. This machine was hampered by the excessive cost of duplicate parts in the earlier stages of the competition, due chiefly to the fact that certain parts, for instance the chuck and its details and part of the front head, were manufactured locally at a much greater cost than if they had been imported. The water in the original design passed through passages *H*, *J*, and *K*. This arrangement was found to be unsuitable for local conditions, and therefore was not used after the underground elimination trials. The details of the front head bushings, designed for water connections in conjunction with hollow steel, were retained because the conditions of the competition did not allow of alterations being made during the competition. The cradle of the machine is faulty in design, in that no allowance is made for adjustment to compensate for wear and tear. During the trial the wear in the cradle had to be taken up by the primitive method of heating and closing by hammering from time to time. The feednuts and screws are on the light side and show considerable wear, thus causing expense for renewals. The valve *A*, which is of the piston type, has not shown undue wear, nor caused much cost for upkeep. The drilling speed of this machine was higher than any other in the competition, showing that both in diameter and stroke it is well proportioned for a machine weighing 100 pounds or under, but if it were made slightly heavier it would be greatly improved.

The Holman 2¾-inch does not drill well in awkward ground and it is liable to fitcher and stick, thus giving considerable trouble. As the travel of its valve is very small it is easily interfered with by dirt. The machine is shown in Fig. 4 as being of the reciprocating type. The diameter of the cylinder is 2¾ inches, the length of the piston stroke 6½ inches. The valve is not of the ordinary operated class, but consists of a hinged flap moving backwards and forwards when the cushioning pressure exceeds the compressed air supply. Valves of this class did not succeed during the competition. The machine was otherwise of the old and well-tried pattern. The cylinder was fitted with a movable steel liner *F* for renewal in case of the wear of the cylinder or piston. This proved a useful feature.

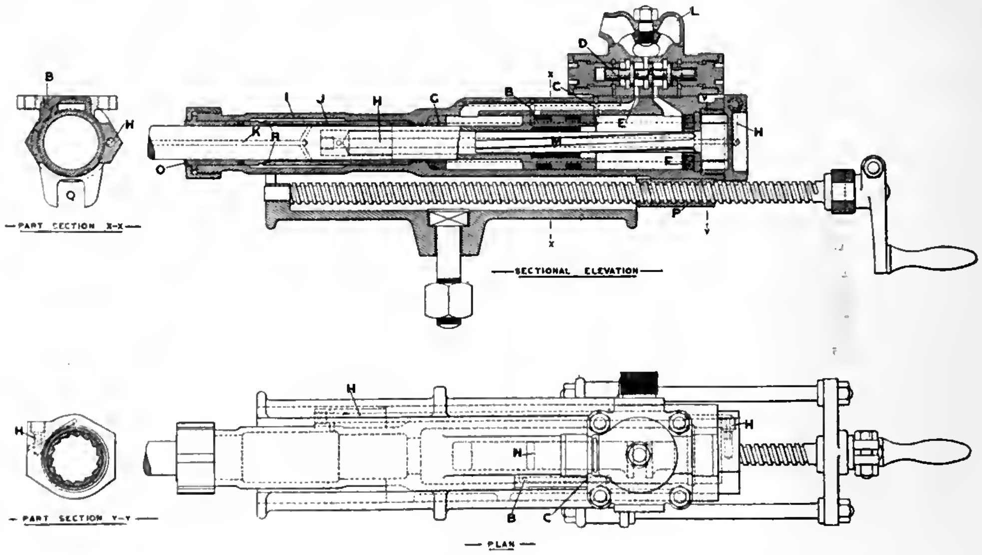


Fig. 3. The Siskol Stope Drill

The Chersen drill, Fig. 5, had a small travel to its valve and was easily interfered with by dirt. This drill did not act well in awkward ground and was liable to fitcher and give trouble. The cut of this drill shows that it is of the reciprocating-piston type, with a piston diameter of 2¾ inches and a

stroke of 6 inches. Compressed air is admitted through the spool valve *A*. It passes through ports *B* and *C* and exhausts through ports *D* and *E*. The motion of the valve *A* on its spindle is caused by compression at each end of the stroke of the piston and is only $\frac{15}{1000}$ of an inch. It was found in practice that unless this travel was kept correct the machine did not drill at its best. Particles of dust or grit easily stopped the proper movement of the valve and temporarily laid up the machine. Considerable trouble was also found in fitchering owing to the want of elasticity in the stroke of the machine. High costs were incurred by piston breakages. The rotation is of the ordinary rifle bar and ratchet type.

The performance of the Chersen was very satisfactory. It suffered, however, from high maintenance cost owing to certain defects in design easily remedied. The only piston drill using water through the machine was the Konomax, and in this case it was found that water leakage could not be prevented internally. Continual trouble in stoppages was thus caused, and finally this machine was withdrawn.

It is interesting to note in these trials, which were carried on for about a year, that the hammer type of machine has not come to the fore, and that the reciprocating type which has been commonly used on these fields has held its position and gained both the prizes. In all cases of hammer-type machines trouble was given by the hollow drills, which crystallized and broke continually. Another trouble was that they frequently stuck in the holes after drilling a certain distance; this being due to the fact that the borings became jammed in a cement-like mass between the drill and the

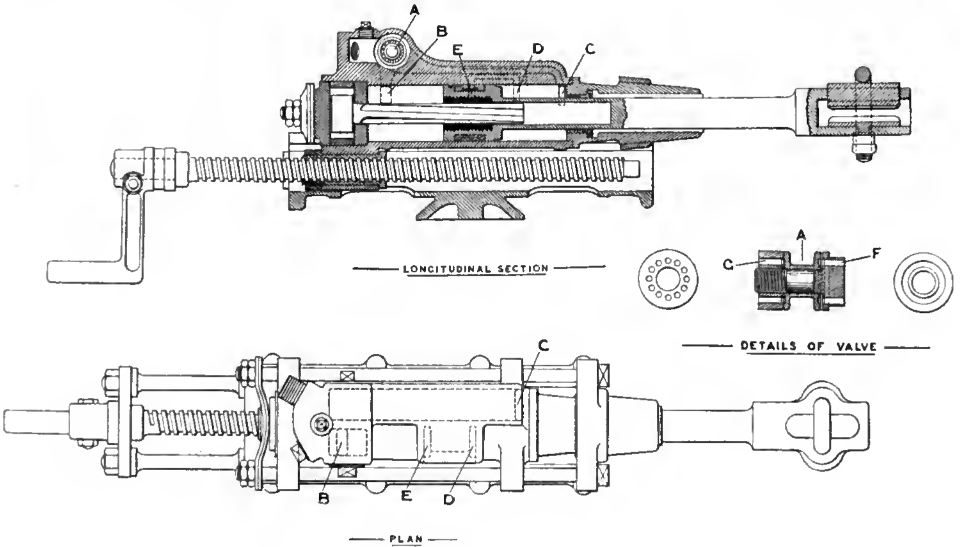


FIG. 5. THE CHERSEN ROCK DRILL

hole. In these cases the holes were lost and fresh holes had to be drilled alongside.

The general deductions made from observations and the results of competition are:

1. That hammer drills are not at present suitable for the general stoping conditions which prevail in the Transvaal fields, that is to say, for putting down holes the reciprocating drill has proved its superiority and there is very little back stoping carried on here.

2. That hollow steel is not at present recommended, the class of material used in its manufacture not being suitable. It is a high-priced article but it appeared to crystallize more rapidly than the cheaper solid steel and gave more difficulty in tempering; which process, it should be stated, was carried out by blacksmiths accustomed to the cheaper solid steel ordinarily used.

3. That no new type of gear was shown to be efficient.

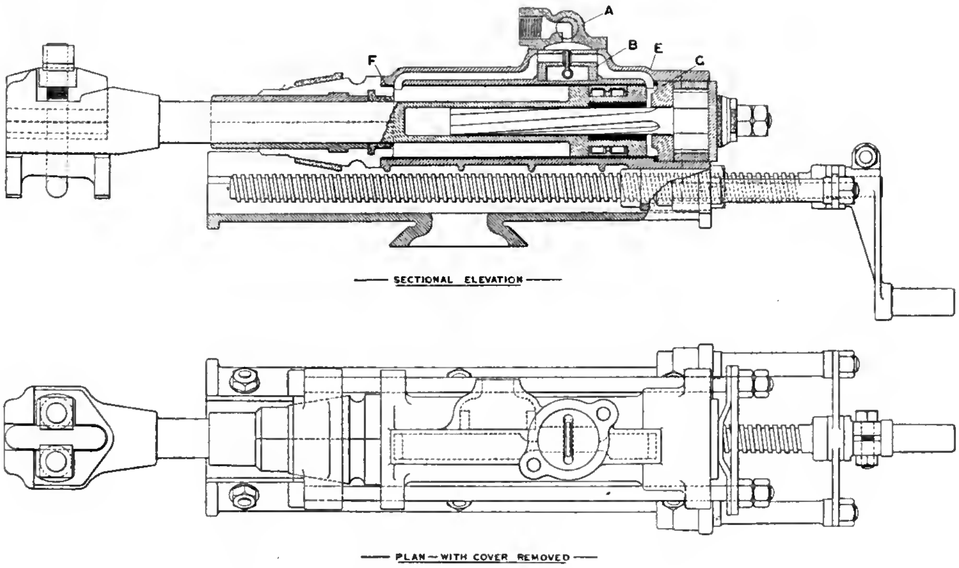


FIG. 4. THE HOLMAN $2\frac{3}{4}$-IN. STOPING AIR-VALVE DRILL

All the finishing machines used the well-known type of jack bar and screw with arms, usually of the Holman type.

4. That any capable miner can supervise efficiently more than two small machines, to which number he was restricted by the rules of the competition.

5. That no two machines finishing in the competition could be efficiently run unless the miner had five native assistants.

6. That it is essential that the arrangements made for the supplying of drills, stores, etc. to the miner should be good, as a short supply of drills often causes the loss of half a shift.

7. That seeing that five boys for two machines weighing 100 pounds are necessary, it would be better to strengthen the machines, adding slightly to their weights but greatly to their durability. Two boys can handle a machine weighing 125 pounds as quickly as one weighing 100 pounds.

8. That under suitable conditions, both as regards mining and efficient working, it is possible to break ground as cheaply and keep stopes as narrow as required with small machines as with hand hammers.

9. That the machines entered in the trials have shown that valve gears operating with a very short stroke are not so efficient as those with a longer stroke, as the longer motion gives more clearance and therefore is less liable to become choked by small particles of dirt passing through the air pipe. This point has been proved by the difficulties that occurred with valves of the type used in the Chersen and Holman $2\frac{3}{4}$-inch machines.

10. That the machines with longer piston stroke have proved themselves in advance of those of shorter strokes in that they have been able to work in broken ground more easily and have also been able to drill deeper holes.

11. That provision for a feed of not less than 18 inches in length appears very desirable.

In the competition, two sets of machines, the Holman 2½-inch and the Siskol have cost approximately 8.3d. per foot drilled plus the cost of steel and drill sharpening, which would come to, at the most, 1.5d. per foot, making 9.8d. per foot. But it has taken 1.282 mine shifts to make one 8-hour shift, therefore the wage cost must be increased in that ratio. Further, instead of assuming 10s. per shift for two machines as the white wage, 25s. per shift were taken for four machines. This alteration means an increase of ½d. per foot drilled, making the total cost per foot of these two sets of machines practically 11.8d. per foot. This is as near as one can get to the actual cost per foot for 28,528 feet drilled by these machines. This cost of 11.8d. per foot compares favorably with the cost per foot by native labor, for which 1s. 1d. per foot would be an exceedingly low figure. The tonnage broken should be as 6 to 5 in favor of the machines. The cost of explosives would be as 6 to 5 in favor of native labor. The tendency is for native wages and cost to increase and this is the heavy item, 10d. per foot, in hammer work, but a much smaller item, only 3d. per foot in machinery. With a proper size of bit a hole 4 feet 8 inches or 5 feet deep is much better as regards tonnage than the ordinary hand-drill hole, for it will carry more explosives and so a greater burden.

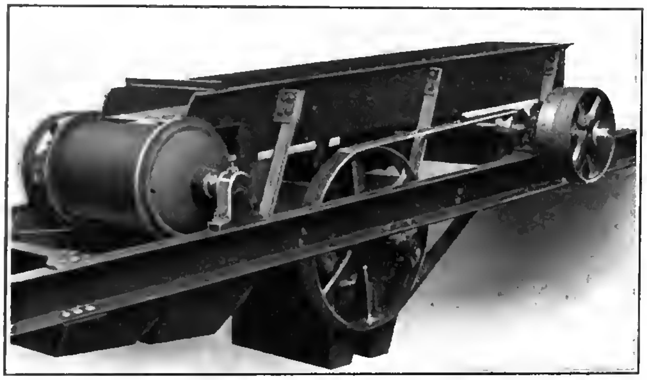


Fig. 1. Two-Pole Magnetic Separator

THE INGERSOLL-RAND-CAMERON COMBINATION

One of the strongest connections of two kindred, yet dissimilar, industries, is the alliance of the Ingersoll-Rand Co., and the Cameron Steam Pump Works. The alliance was effected by the Ingersoll-Rand Co. purchasing a controlling interest in the Cameron Steam Pump Works. The combination is in no sense a consolidation, for each concern will be operated separately and under separate management. The titles of both companies will also be unchanged. For the present the Cameron Steam Pump Works will remain at the foot of East 23d St., New York, but a new and more commodious plant at some other place is in contemplation.

Under the new arrangement, Mr. Geo. W. Fuller, who entered the employ of the Cameron Works in 1883 as office manager, and who has been general manager since 1899, continues as general manager with the additional title and authority of vice-president.

Nearly 50 years ago both companies had their origin and both carried on their operations under the same roof. Later the Ingersoll-Rand Co., through their representatives both in America and in foreign lands, sold Cameron pumps, and the Cameron Pump Works made the castings for Ingersoll compressors, until the Ingersoll-Rand Co. built its own foundry. In this alliance there was only one of friendship and mutual business interests. Both companies were separate and distinct organizations without any financial interest in each other. Naturally when a time arrived that the Ingersoll-Rand Co. could, in a friendly and amicable manner, purchase a controlling interest in a company whose products they had justly indorsed for many many years, the purchase was made. That the high standard of Cameron pumps will be maintained, and if possible improved on, is doubly assured by this combination. Mr. Fuller, as all who have ever met him know, is imbued with the one idea, and that is that in steam pumps, as in human character, the Cameron slogan, "Character is the Grandest Thing," must apply. That he will be supported in this high ideal by the influence of the officials of the Ingersoll-Rand Co. now in the directorate of the Cameron Pump Works, is evidenced by the high character of the products of the former company.

MAGNETIC SEPARATION OF WOLFRAMITE

Written for Mines and Minerals

Although the mining industry of Great Britain is chiefly devoted to coal, the extraction of other minerals is also extensively carried on and an interesting example of the use of electricity in connection with ore dressing is the use of the magnetic separator in connection with the ores mined from the North Tincroft mine of the Carn Brea and Tincroft Mines, Ltd. Here a Wetherill magnetic separator has been installed by the British Humboldt Engineering Co., Ltd., and as this separator is representative of the latest means of separating wolframite, arsenic, copper, and sulphur pyrite from tin ore, green sand from zinc blend, titaniferous iron from diamonds, and copper pyrites from silica, a brief description of its construction and use may be of interest. The material uniformly spread over a belt is conveyed under a magnetic edge, which extracts the magnetic material, and this is conducted out of the field by a cross-belt running under the magnet to the hoppers at the side. A very clean separation is effected through this lifting out and for this reason this separator is very largely used for valuable minerals and ores. It has also been discovered that for ores whose metallic particles are very small and which therefore necessitate fine crushing, this separator gives excellent results.

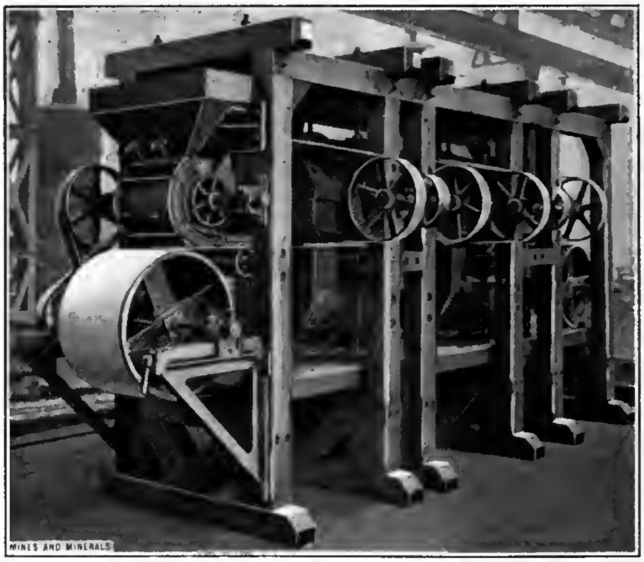


Fig. 2. Six-Pole Magnetic Separator

Owing to the edge shape of the magnets a high concentration of the field is attained, so that this separator can be used for feebly magnetic ores.

The Wetherill separator is built as a two-, four-, and six-poled machine, according to the purpose for which it is required. If only one constituent of an ore is to be separated or the quantity to be treated is small, so that by altering the strength of the

field the material can be retreated, then the two-poled machine (see Fig. 1) is ample. The six-pole machine (Fig. 2) is only considered when four products of different permeability are required. These differences of permeability may be due to the material being ingrained; for example, the six-poled machine is used for enriching slightly roasted copper pyrites. Six different copper products are obtained, that at the first pole being the best, that at the last most ingrained. The last product can thus be further reduced and treated again in order to enrich it. Magnetic separation is especially adapted to copper ores, for wet handling introduces considerable losses due to floating.

For the separation of tin and wolfram minerals the cross-belt is extensively used; the first pair of magnets generally eliminates the feebly magnetic iron, while the second and third pairs extract wolfram. If there is not much wolfram in the crude ore the four-poled machine is sufficient to remove it. These separators are built in two sizes, that is to say with poles 360 and 450 millimeters in width. The latter machine is of much stronger construction, so that the magnetic strength is also greater, the consequence being that both the capacity and efficiency are increased.

The wear of the belts is small; changing can be quickly carried out and the small cost of the belts is soon made up by the increased profits due to the excellent separation. Fig. 3 shows another view of the six-poled machine, and is the type of plant used at the North Tincroft mine.

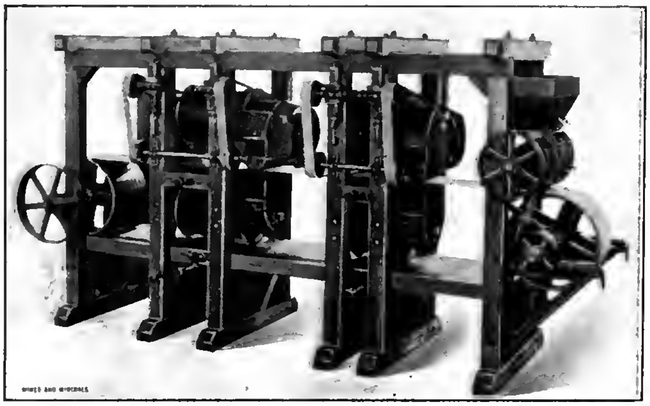


Fig. 3. Six-Pole Magnetic Separator Used at North Tincroft Mines

Personals

H. C. George, B. S. E. M., who for the past 3 years has been Director of the Wisconsin State Mining Trade School, has resigned to become mining engineer for the Wisconsin Zinc Co.

Edward G. Locke, formerly with the United States Geological Survey, has been placed in charge of the mines of the Davis Coal and Coke Co.

H. W. Shields, land agent, and Thos. H. Clagett, chief engineer, will be located in the annex to Norfolk & Western Railway Co.'s office building, at Bluefield, W. Va., instead of at Bramwell, W. Va., after February 1, 1911.

Henry F. De Bardeleben will occupy the position of vice-president of the Alabama Fuel and Iron Co., made vacant by the death of his father.

Miss Kathryn McGough is president of the Phyllis Mining Co., with headquarters at Monongahela City, Pa.

H. C. Parker, of Logan, Utah, is with the Nevada-Utah Co., at Pioche, Nev.

David Cole, formerly assistant general manager of the Green-Cananea Co., is now associated with the Ray Consolidated Copper Co.

Walter Harvey Weed, of the firm of Weed & Probert, has finished his examination of the Butte-Balaklava properties, at Butte, Mont.

Dr. H. G. Torrey, who was chief assayer at the United States Assay Office, in Wall Street, has opened an assay and metallurgical office at 99 John St., New York.

Frank H. Probert, of Los Angeles, recently examined the Ray Central mine at Ray, Ariz.

YUKON TERRITORY OF CANADA

What is known as the Klondike placer mining district, has produced $150,000,000 in gold since 1898, and mining experts estimate the amount yet to be mined equal to that already produced However, as the remaining gravel is of a lower grade, the work must be done by machinery, and for this purpose 20 dredges are in operation this season, as well as a number of hydraulic plants. There are 10 large mining companies operating here, the principal one having 10 dredges and a large hydraulic plant, and it has invested over $17,000,000 in the district. There are also 6 incorporated companies operating for quartz in the district, but little has been milled, although some high-grade ore has been found.

The gold production of the district for 1909 was $3,595,985, an increase over 1908 of $307,321, and the indications are that the production in 1910 will exceed that of 1909. This gold, with the exception of a small quantity sent to the new Canadian mint, was all shipped to the United States, principally by registered mail.

Rich deposits of copper ore have been discovered in the southern part of the district, which is being worked and shipped to a smelter at Tacoma, Wash. The richer ore seems to be on the White River at the Alaska-Canadian line, but this is not mined because of the lack of shipping facilities. The ore extends over a large territory and is claimed to be the richest in the world. A good grade of bituminous coal is found in various localities. An English concern engaged in mining coal is constructing a 10,000-horsepower electric plant for the purpose of supplying Dawson and other towns with heat and light, as well as furnishing power to dredges, etc., within a radius of 75 to 100 miles of the plant.

All but two of the dredges mentioned were manufactured in the United States, as well as three-fourths of the tools used here for mining purposes. Two-thirds of the mining companies are American concerns and four-fifths of the capital invested in mining is American.—*United States Consular Report.*

Obituary

Charles Francis Shelby

Charles Francis Shelby, who went to Peru in 1910 to become general superintendent of the Cerro de Pasco Mining Co.'s smelting plant, was killed January 25 by a railroad automobile jumping the track and striking an iron telegraph pole. Mr. Shelby was a graduate of the International Correspondence School of Metallurgy, having taken up the course while at Aguascalientes, Mex., where he was a chemist. Soon after, he became assistant superintendent, then superintendent of the Old Dominion Copper Co., at Globe. In 1906 he went to Cananea, Mex., and in 1910 to Peru.

Mr. Shelby's aim was to become the best in his line, and in this he did not fall far short. Unlike most metallurgists, he was alway ready to impart as well as to absorb knowledge, as the readers of Mines and Minerals and the *Engineering and Mining Journal* can testify. Mr. Shelby will be buried in Cleveland, Ohio, so soon as Mrs. Shelby and baby can reach that city.

THE PHILIPPINE GOLD MINES

Written for Mines and Minerals, by Monroe Woolley

Recent discoveries of gold in the South Camarines, Province of Luzon, have brought the mineral possibilities of the Philippine Islands into prominence. Probably the richest gold deposits (at least so far as known) are in the vicinity of Paracale, although there are other deposits, as shown on the map, Fig. 1.

Regions and Formations in Which Gold Has Been Found. Conditions Affecting Development

According to Warren D. Smith, Chief of the Division of Mines in the Philippine Islands, the Paracale-Mambulao gold field was recorded in 1571. The Chinese and natives washed gold in a crude way until the Spaniards taught them the use of the arrastra a few years previous to the American occupation.

Native women wash the streams where gold is to be found, being adepts in the use of the pan and batea. A group of women gold washers is shown in Fig. 2, who are partly civilized and who live in the vicinity of Penaranda, where gold washing has been carried on for centuries. The natives at this place resent so strongly the American invasion that constables are required to protect prospectors. The Igorotes and Negritos, uncivilized mountain tribes of the islands who never come down into towns, have been taking gold from their mountains for perhaps centuries. What little they obtained was devoted to ornamental purposes, and it is only recently that they have commenced to sell it to white people. Mindoro, or island of gold mines, has never been systematically prospected or the geology of its interior studied. Placer deposits have been found on the west and north coasts where a few Americans have claims. However, little, if any work has been done on them. From a glance at the map it will be seen that the Paracale mines are not far from Manila, and that between Paracale and Mambulao Bay there is an almost continuous mineralized zone. The Paracale mines were worked in a crude way by Spanish and English, who completely abandoned them. Americans acquired them shortly after the occupation by our troops, and at present new companies have been formed to exploit and prospect these gold deposits in a systematic way. Ore from one mine in this vicinity was shipped to Manila that carried $147.60, and contained besides gold, silver, copper, and zinc. At the present time most of the gold obtained in the immediate vicinity of Paracale is from dredging operations in the rich bottoms of the Paracale and Malaguit rivers. Most of the dredging ground has been staked out on these rivers, although but one dredge is working on the Malaguit, while three are at work on the Paracale, and another is about constructed. Two of the dredges are of the New Zealand type and are manned by New Zealanders; while three are of the Postelthwaite type. The essential difference between these dredges is the absence of a tailing distributor and the rotary screen and side tables on the New Zealand dredges.

Fig. 1. Map of the Philippine Islands

Lode deposits are not receiving so much attention in the vicinity of Paracale as to the west toward Mambulao, nevertheless, development work is being carried forward on several properties in the foot-hills. As the roads and conveniences are not favorable for moving heavy machinery, considerable difficulty is experienced in transporting it to the mines, as may be seen by a glance at Fig. 3. The heavy wagon is pulled up the incline by means of blocks and falls, and the use of a capstan, shown in Fig. 4. Before the machinery is in operation capstans are used to hoist rocks from the shaft.

To the west of the Paracale, near Mambulao, are the ancient workings of the natives and of the Philippine Mineral Syndicate that was composed of Spaniards and Englishmen. Recently two companies—the Tumbaga and San Manricio—have been preparing for immediate operation.

Tumbaga lies 2½ kilometers south of the town of Mambulao.* The ore body is in a brecciated zone between impure sandstone and andesite. The strike of the ore deposit is northeast and southwest. In it there is considerable shattered slate which has been cemented by calcite carrying free gold. Although the mineralized zone reaches a width of 35 feet, the calcite stringers do not all carry gold, consequently the gold is not constant in quantity. Over 100 meters of drifts and crosscuts have been opened in this lenticular ore body, and the ore treated as follows:

Crushing coarse; grinding in a Huntington mill having a capacity of about 21 tons per day; amalgamating on outside plates, and concentrating the pulp that passes over the plates on two tables.

The mines on the island of Masbate, at Aroroy, are not so extensive or so rich as those in the Camarines. However, perhaps a dozen companies are now installing mining machinery. †"In all probability mining was carried on in the Aroroy district before the Spanish conquest." "Old open cuts showing the use of fire in breaking rock, and stone mortars for crushing the ore are indications of the ancient native miners." "The natives say that about 100 years ago an arrastra was worked by a Spaniard and Englishman, and in several places there are underground workings that cannot date back very far."

* Mineral Resources of the Philippine Islands, 1910, page 10.

† H. G. Ferguson, Geologist, Philippine Bureau of Mines.

Near Baguio, the summer capital north of Manila, several Americans and some aliens hold paying claims known as the Benguet mines. A lone prospector of the Benguet country sold his claims to a Hongkong, China, syndicate, for $20,000 cash. A bar of gold weighing 233 ounces, valued at $4,000, was shortly afterwards sent to Manila from these mines, and this, with a smaller one, represented the clean-up of free-milling ore for 1 month. American miners, according to Frank T. Eddingfield,‡ have been prospecting in Benguet for 10 years, "but only in the past 3 years has successful mining been carried on by organized companies; previous to this the veins had been worked by Igorotes, Filipinos from other provinces, and Chinamen." "Some of these old workings are quite extensive, and have been a great help to prospectors in locating veins." "When driving a drift an old stope was found in which the waste rock was cemented as hard as wall rock. There are four mills, 12 mining companies, and four prospecting companies reported in this district from which 279,600 pesos were taken in 1908 and 243,500 in 1909. These figures, while showing the mining activity of the district, do not give an idea of the development work and prospecting which is being done on some 300 claims. The principal producers are the Benguet Consolidated and the Bua. The latter is working two veins, one of quartz .3 to 1 meter wide, the other a rhodochrosite vein 1 to 2 meters wide.

FIG. 2. WOMEN WASHING GOLD

The ore is mined by overhead stoping, the excavation being filled with waste broken during mining. The ore is dumped in a bin at the mouth of the adit, from which it is trammed for a distance of about 1 kilometer to a second bin which feeds into a two-bucket gravity aerial tramway, stretching across the Antamok Valley to the mill. The mill, which is run by water power, consists of six stamps with two amalgamating plates; the tailing from these is cyanided and then discarded. A custom mill designed to treat the ore in this district is greatly needed. Igorotes are used for surface laborers and cutting timbers, Filipinos from other provinces as miners and mill men, as they are steadier and better workmen; Japanese are used as timbermen and carpenters because of their greater strength and adaptability.

‡ Mining Engineer, Bureau of Mines, Philippine Islands.

A number of men from Manila formed a company and tested the placer ground in the vicinity of Penaranda with a drilling machine. The drill tests were reported as showing 33 centavos per cubic yard; while pan tests in other places gave gold varying from 50 centavos to 1 peso per cubic yard. One of the interesting features in connection with this Nueva Ecija district is the presence of platinum in the placers. The natives who have been washing for gold in this district for ages, ignorant of the value of platinum, threw it away. The only other place in the Philippines where platinum has been found is 40 miles south from Penaranda.

According to Mr. Eddingfield gold has been found on Catanduanes Island with evidences of placers having been worked by Spaniards or natives. Old Spanish reports of 1877 speak of placer mines in Mindanao.

Filling Stopes

In large veins ore is sometimes lost by settling upon the timbers of the first few sets above the sill floor. The weight of the ore crushes the timbers and forces them out of line to such an extent that cars can no longer be trammed through the levels. When this occurs, bulkheading and filling must be quickly done to prevent collapse.

When a case of this kind, occurred some time ago at the Anaconda Mine, Butte, Mont., and the timbers on the sill floor were forced out of line, the stope was filled with waste and timber as rapidly as possible and the level abandoned. A lateral drift was then run in the foot-wall 60 feet from the vein; and from it at distances of 60 feet and directly opposite the chutes in the abandoned stope a square set was placed. At each such station a raise was put in and carried up on an incline to the ore body above the filled stope. The drift was in solid granite, but as the raises approached the vein the rock was found decomposed, and required careful timbering. The raises were timbered with square sets, though had the foot-wall been solid no timber would have been required. Through these raises nearly all the ore lost by settling was recovered.

FIG. 3. HAULING MINING MACHINERY IN PHILIPPINE WILDERNESS

FIG. 4. SCENE AT PARACALE MINES

GEOLOGY OF THE LA VETA COAL FIEI D

Written for Mines and Minerals, by Arthur Lakes

The difference between the geological conditions surrounding the coal fields of the Rocky Mountains and those of the Eastern States is very marked.

An Illustration of Volcanic Action Upon the Quality and Location of Coal Seams

In the East the coals are mostly of the Carboniferous period of the Paleozoic system and found in the Appalachian system of mountains and to the west of them. In the Rocky Mountains, taking Colorado as a type, the merchantable coals belong to the Mesozoic system and are mainly of Cretaceous period. There are a few inferior and unimportant Tertiary coals, and none workable of Carboniferous period.

The western coals underlie the horizontal plains for an unknown distance or occur between horizontal strata of lofty tablelands bordering on the mountains, or lie in a highly tilted position in the sandstones of the foot-hills, or within the parks of the granitic ranges. The coal measures may be capped with flows of lava, or intruded by horizontal sheets and vertical dikes of igneous rock emanating like the tentacles of an octopus from some center of eruption, such as from an old volcanic neck or a laccolite, as shown in Fig. 1.

Eastern coals are mostly of the bituminous, semibituminous, or anthracite type. Western coals grade from brown lignite to black lignite, to lignitic-bituminous, to full and coking bituminous, and locally to anthracite.

Some of the eastern coals are of the coking variety, others are anthracized by metamorphic heat connected with the mechanical folding up of the mountains.

Western coals are locally coking, sometimes over considerable area and occasionally anthracized more locally and on a smaller scale; both conditions and qualities are due to their proximity to igneous rocks.

A striking example of the characteristic geological conditions in the West is shown in that part of the great Trinidad coal field of Southern Colorado, which is dominated by the Spanish Peaks as a volcanic center, with La Veta and Walsen-

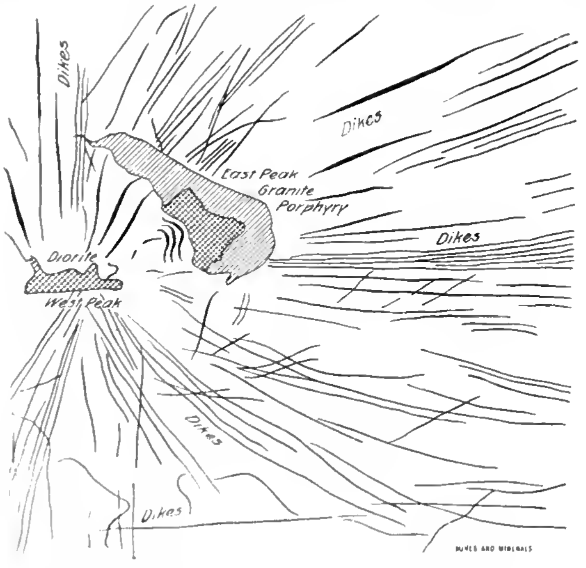


Fig. 1. Spanish Peaks, Showing Plan of Dikes

burg as the principal railroad coal towns. Here in the heart and center of the coal fields, and greatly influencing the character and quality of the surrounding coals is the most extraordinary volcanic eruption in the Tertiary period of the Rocky Mountains, the monuments of which are two isolated lofty and prominent mountains called the Spanish Peaks, rising from the prairie and foot-hill zone to a height of 13,623 feet above sea level, a few miles east of the main Sangre de Cristo Range. These symmetrical peaks represent the center of a series of eruptions which for a long time affected the southern part of the state for hundreds of miles. From this center emanated a system of dikes radiating from the central peaks like the arms of a cuttlefish. The singular dike walls, Fig. 2, can be followed across the foot-hills and over the prairie for many miles, giving

Fig. 2. La Veta Region, Showing Dikes

rise to such names as Crestone, Stonewall Park, Muralla Wall, or Dyke Mountain and the like. The entire region is traversed in every direction by a network of dikes and "sills," and it would seem as if the country at the time of these eruptions must have literally sweated igneous matter.

The traveler can wander amongst these dike walls, rising at times in the vicinity of the peaks a hundred or more feet above him like ancient fortifications, pierced here and there by natural embrasures, or eroded into battlemented forms. The walls are occasionally utilized by stockmen as shelter for their flocks and herds, or as forming convenient sides for corrals. The faces of these cliff-like walls betray their igneous origin in rounded outlines, suggestive of molten matter retaining the form of the soft shale mold in which it was cast and through fissures in which it ascended and was congealed in the form of dikes. Fragments of shale altered by heat into slaty hardness still cling to the walls.

As the sedimentaries approach the central cores of eruption against which they are tipped, they show the evidence of heat in being metamorphosed.

The influence of these centers of igneous intrusion has been on the whole beneficial to the surrounding coal field, changing the lignitic-bituminous coal, characteristic of the more northern fields of Colorado, into a coking bituminous coal. The destructive effects on the coal have been local. Intrusive horizontal sheets have locally changed the coal into a worthless species of natural coke or graphite, Fig. 3. The effect of the dikes passing through coal seams has generally been confined to a width commensurate with the thickness of the dike. Dikes and intrusions, if numerous, are sometimes annoying, but not insurmountable obstacles in the development of a coal mine, as they can be readily cut through.

Despite the geological disturbances to which the La Veta region has been subjected, faults of importance are singularly few in the coal measures. Along the eastern border from Walsenburg north to the Huerfano River, the coal plateau is abruptly terminated by erosion, displaying a line of low sandstone cliffs containing several workable coal seams dipping gently southwest into the plateau, which here is in the form of a broad synclinal fold, which further west, near Oakdale, is followed by an anticlinal, and that in turn by another synclinal whose western limb highly tilted against the Sangre de Cristo Range, raises the coal to verticality. Here, at the Occidental mine there are two seams 8 and 4 feet thick.

The intervening country of Huerfano Park, between the Sangre de Cristo and La Veta Mountains is doubtless underlaid by coal whose outcrops are obscured by a large overlying deposit

of Eocene-Tertiary lake beds, 1,000 to 2,000 feet thick. These coarse sandstones and conglomerates were deposited in a lake occupying one of the great trough-like folds parallel with the Sangre de Cristo and Greenhorn Mountain ranges.

For a detailed account of the eruptions of this interesting region we are indebted to the researches of R. C. Hills.*

The prominent feature connected with the overflows of lava covering the Raton hills in Southern Colorado, is that they

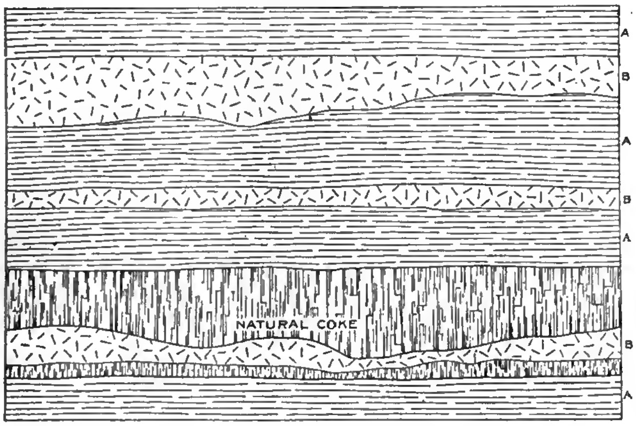


FIG 3. NATURAL COKE AND SILLS OF IGNEOUS ROCK
A, Shale; *B*, Igneous Rock

contain most of the coking coal of this part of the state. The dikes, radiating throughout the region, and the Veta, Silver, and Black Buttes mountains, are northward prolongations of the eruptions of the saddle-shaped twin mountains known as the Spanish Peaks.

The Spanish Peaks and Silver Mountain, Fig. 4, are a similar and connected eruption, as shown by their component rocks, and they are directly connected with the great system of diverging dikes. The upheavals are due to intrusions of igneous rock into the Cretaceous shales. The intrusions were upwards of 2,000 feet thick and extended in wedge-like form along the shales 8 to 12 miles. The sedimentary rocks overlying these intrusions were arched and fissured, allowing molten lava to ascend from the main igneous reservoir through these vents, giving rise to two systems of dikes.

Dikes radiating from the West Spanish Peak are most numerous and cut through the sedimentaries lying on the upper flanks of the mountain.

The laccolitic masses of the Spanish Peaks are composed of a succession of consecutive intrusions of different varieties of lava.

At Silver Mountain the upheaval was greatest immediately under the crest of the mountain, which is the focus of intersecting and diverging dikes, so numerous as to exclude all other rock matter. These dikes do not descend below the laccolitic horizon in the Colorado shales.

The igneous material composing the La Veta, Black Buttes, and Silver Mountain is from one magma. Another system of intrusions and dikes having no connection with the former appears in the Colorado shales from the state line to the Huerfano River, 50 miles. These intrusions produced in the overlying coal measures a system of fissures through which lava exuded. All these dike systems were formed from the reservoirs of molten rock underlying the region and from which on the surface these dikes were the manifestations.

In the South Veta and Sheep Mountain is a second group of laccolites unconnected with any corresponding system of dikes. Five intrusive bodies belong to this group in the northwest portion of the area between Cuchara and the Huerfano River. The Veta Mountains consist of two distinct eruptions of equal size forming North and South Veta peaks. North Veta belongs to the Silver Mountain and Black Buttes systems.

* United States Geological Survey, U. S. Folio, No. 71, 1901.

South Veta is of hard white porphyry, unlike the coarse, hornblendic lavas of Silver Mountain and the Spanish Peaks. Veta Mountain is of later eruption than Silver Mountain.

Upheaval was followed by subsidence due to the extravasation of so much molten matter and its ascending into fissures caused by upheaval and in them solidifying as dikes. Each lava injection was a pulsating movement giving rise to independent systems of dikes.

The alteration effects on the coals, changing dry coal to coking, was due to radiation of heat through the medium of water.* More pronounced alterations, such as to natural coke, semianthracite, or graphite were due to the direct contact or action of steam or very hot water.

At the close of the Laramie coal-making period general upheavals accompanied by folding formed long trough-like depressions parallel to the main ranges that formed the basins of subsequent Tertiary fresh-water lakes. Less pronounced folds eastward also formed minor lake basins.

One of these greatest trough-like depressions received the Tertiary lake deposits, known as the "Poison Cañon" series. This lake covered the depression between the Greenhorn Mountains and the Sangre de Cristo, receiving sediments washed down from the neighboring highlands. The thickness of this series of rock beds is between 1,000 and 2,000 feet, and it covers an area of 250 square miles. These beds are folded and metamorphosed against the West Peak. Among the carnivora and rodents that once inhabited the shores of this semitropical lake was one the size of a little black bear, termed "Tillotherium," whose jaws were armed with enormous incisors; another has been called "Coryphodon." Bones of the diminutive Tertiary horse, or Pliohippus, have been found; also the huge carapace of gigantic turtles.

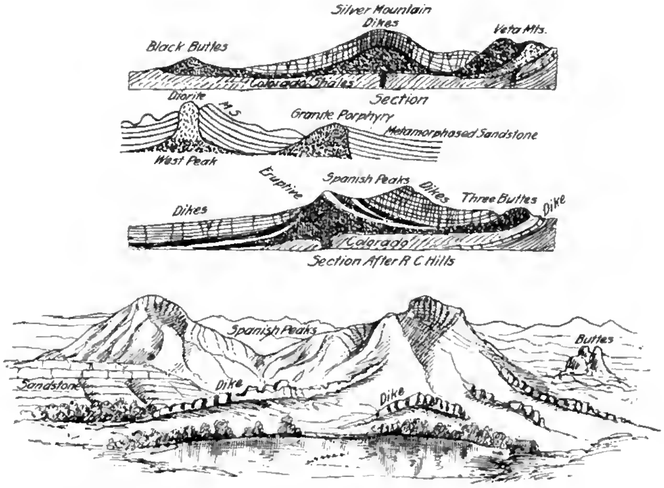


FIG. 4. SPANISH PEAKS AND SILVER MOUNTAIN

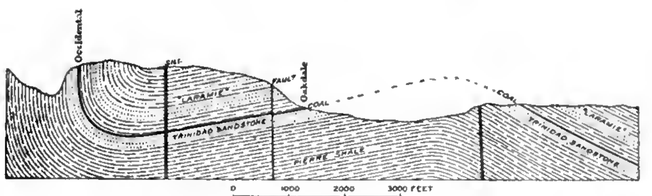


FIG. 5. SECTION OF STRATA NORTHWEST OF LA VETA, COLO.

Previous to the last eruption the sedimentaries of the West Peak were riven in an east-west direction, and a mass of igneous rock ½ mile wide, was forced up into the vent. At the same time another portion of the igneous mass was intruded in lenticular form beneath the sedimentaries of East Peak. At a later date final eruption was initiated by the production of a U-shaped fracture, a prolongation of the old vent across the base

* The editor was not there, and is unable to disapprove these two sentences.

of East Peak, and thence to the West Peak. This was accompanied by displacement of strata along the line of fracture as much as 5,000 feet. Sediments included between the **U**-like arms were upturned in a spoon-shaped trough. The effect of this eruption on the previously formed trough was the production in its bottom of a great upward bulge, rent and faulted at the summit, warped by the intrusion of huge masses of igneous rock and ribbed by a system of dikes.

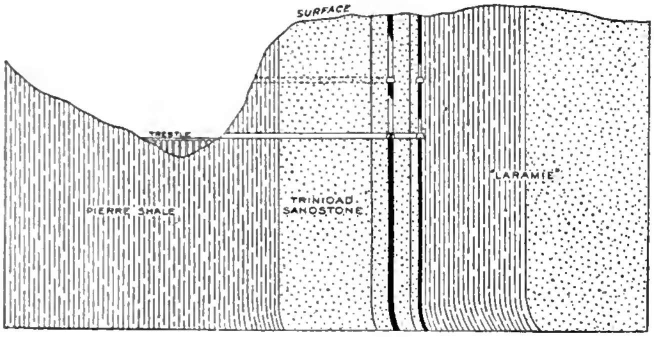


FIG. 6. SECTION AT OCCIDENTAL MINE

The igneous rocks of this region, identified by R. C. Hills, are: Monzonite porphyry; early lamprophyre; late monzonite porphyry; late lamprophyre; augite diorite; augite granite porphyry; granite felsophyre; basalt. Eruptions continued intermittently nearly up to the present time; the varieties of rocks observed may be due to differentiation from one common magma.

Coal Seams of the La Veta-Walsenburg Area.—The Walsenburg area embraces the coal beds and outcrops on the northeastern extremity of the Trinidad field. The coal plateau here terminates in abrupt cliffs of yellow Trinidad sandstone facing the prairie. Below this, in Colorado, well-known bedrock of coal, are the non-coal-bearing Cretaceous shales, which largely underlie the eastern prairie. Above the sandstone, within the first 200 feet, are usually two to three workable coal seams. The cliffs represent the upper limit of a synclinal and the commencement of a broad anticlinal stretching toward the prairie. The crest and remainder of the anticlinal has been entirely eroded on the east, so that there is no more coal east to the Kansas boundary of Colorado. The coal outcrops are very clearly seen along the cliffs, and several small mines and openings are on them. Near Black Buttes, on the north, the Laramie coal-bearing series is overlapped by the "Poison Cañon" Tertiary beds. Along the eastern margin the coal dips gently inward to the plateau southwest and becomes almost horizontal in the bottom of the succeeding synclinal trough. Only the lower group of the Trinidad coal-field series is worked at Walsenburg, principally three prominent parallel beds locally known as the Cameron, Walsen, and Robinson. The coal beds in this field cannot be persistently correlated or identified over large areas, the coal horizons changing.

A prominent feature of the Walsenburg area is a great dike extending east and west for many miles. It cuts through the coal about a mile northwest of Walsenburg. The Maitland and Pictou mines have long been worked on these seams. North of Maitland is the Pinon, and still further north along the D. & R. G. R. R., the Tioga and Sunnyside. South of Tioga are horizontal intrusions of igneous rock, and the position and character of the coal should be tested by drilling. Between the Big Four and the Oakdale mines, on opposite sides of the synclinal basin, the coal outcrops are obscured by Tertiary beds. It is probable that the coal measures extend beneath this basin to Huerfano Park. The depth and thickness of the coal should here be tested by drilling.

The La Veta district includes the area contiguous to the coal outcrop along the northwestern margin of the Trinidad coal field. In the center and southern part of this basin the continuity of the coal is interrupted by those great igneous intrusions already mentioned. The coal measures in this area dip northeast from 10 to 65 degrees. At northwest end the rocks are folded into anticlinals and synclinals. The Laramie formation is overlain for a considerable area by the Poison Cañon Tertiary formation. This Veta field offers an inviting area for investigation by drilling. It has not been thoroughly prospected, largely owing to the local covering of Tertiary rocks. Only the lower group of coal seams have so far been found and worked.

In the subsidiary basin north of the Denver & Rio Grande Railroad, there has been considerable development work. Here coal is cut off at the north end by the intrusion of the Veta Mountains, Fig. 5. On the west side of the basin the coal stands vertical; on the east side the dip is 15 degrees and to the southwest.

The Occidental mine is connected with the Rio Grande Railroad by a long tramway, and the coal is reached by tunnel 300 feet long (Fig. 6). The coal is in two seams 45 feet apart, one 4 feet, the other 8 feet thick.

The Oakdale mine, on the east limb of the synclinal, was working in 1908 on a 7-foot seam.

Outcrops of narrow coal seams occur along the main west margin of the Spanish Peaks syncline west of La Veta, but are rarely of workable size.*

THE MAIKOP OIL FIELD

The Maikop oil field is situated in the Kuban Province, in Southern Russia, on the western fringe of the Caucasian Mountains. As at present known, it has a total length from east to west of approximately 25 miles, while its greatest width from north to south is about 20 miles.

The field only came into prominence during the latter part of 1909, owing to the sinking of a 240-foot well, which is stated to have thrown upward of 30,000 tons of petroleum in 6 days, and after defying all attempts at control for weeks continued to discharge oil in large quantities. This marked the commencement of the commercial history of petroleum in the district, for although the region has been known for centuries to be oil bearing, no serious attempt had previously been made to test the territory by means of boring.

It is the most favored petroleum-producing region in Europe, being only about 50 miles from Tuapse, on the Black Sea, with which export point it will in time be connected by both railway and pipe line.

At many points, particularly on the hillsides, large outcrops of impregnated oil sands have been noticed, and in other places exudations of petroleum are very frequently encountered. To the east of Khadijinskaya is an asphalt mountain, from almost every spot of which oil oozes, which, after losing its volatile constituents, leaves a tarry residue spread over the surface. To the southwest of Nephtanaya, indications of the petroleum deposits have been evident from time immemorial. It was here the Black Sea Co.'s spouters were struck and not only are the most prolific oil sands encountered at shallow depths, but exploration generally can be conducted at considerably less cost than in the Baku region.—*United States Consular Report.*

RULE FOR FIGURING CAPACITY OF ROLLS

By Charles F. Spaulding

Let S equal speed of roll stull in feet per minute; F equal face of roll stull in feet per minute; O equal opening of rolls in feet; C equal factor for irregular feeding (usually 2 to 4).

$$\text{Tons per hour} = \frac{S \times F \times O \times 60}{C\ (2\ \text{to}\ 4) \times \text{cu. ft. in ton}}$$

Taking $C=3$ for average work, and 20 cubic feet to a ton, the above equation becomes:

$$\text{Tons per hour} = S \times F \times O$$

* For further information on this district, see U. S. Folio Map 71, and G. B. Richardson's Report, Bulletin 381 of the United States Geological Survey.

ASBESTOS

*By H. R. Edgecomb**

Amphibole and Chrysotile. Sources of Supply and Uses to Which Asbestos May Be Applied

Take a certain kind of glass, soften it by heat, draw it out into fine, flexible thread, weave it with care into a fabric incorruptible as to decay and the action of most acids, but not as to heat, and you have approached the limit of man's effort to produce a mineral fiber. Nature makes a shrinkage crack in a rock, fills it with water, dissolves a little of the rock in the water, and crystallizes the whole crack full of threads which are much finer, more flexible, and stronger than the glass fiber, and in addition thereto capable of withstanding a temperature of 2,000° to 3,000° F. Remembering that the water combined in these fibers runs as high as 14 per cent. of the whole weight, one is forced to the conclusion that some of nature's processes are "past finding out."

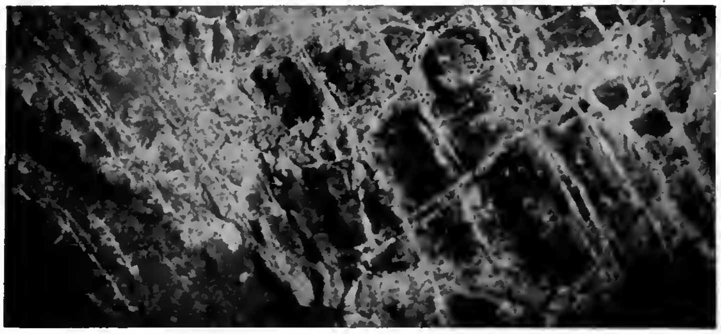


Fig. 1. View Showing Asbestos in Walls of Drift in Bell Mine

The Greeks named this fiber "asbestos" (unquenchable, unconsumable); the Romans made cremation robes of it, and Charlemange astounded his gaping courtiers by casting his soiled table cloth into the fire and withdrawing it clean and white for another feast. Nor has it been many years since an otherwise respectable and industrious lumberman was accused of witchcraft and run out of a camp in the Canadian woods because he persisted in washing his socks in the stove instead of by more conventional methods.

Because of shrinkage, the outer surface of the earth has become wrinkled and cracked, folded, and broken, worn and torn, and the fissures filled by rock from within. In the vicinity of Thetford, Province of Quebec, one of these intrusions of rock occurs. It is, more correctly, a series of rocks, including peridotite, pyroxenite, gabbro, diabase, and others. The peridotite when altered by hydrating is called serpentine. During the cooling of this peridotite it is supposed that fissures formed and that the hydrating process which caused the rock to take water into its structure widened these fissures. During the hydrating action the water carrying some of the serpentine in solution collected in these cracks, and eventually the dissolved mineral formed thread-like crystals, usually building up from opposite walls of the crack and meeting or forcing past each other at the center. The cracks are generally straight, and do not occur in parallel planes, but cross each other in all directions.

Broadly, asbestos may be classified as amphibole or hornblende and chrysotile, or serpentine asbestos. The amphibole, hornblende asbestos, does not possess the fineness of fiber, the tensile strength, the elasticity, or the flexibility of the chrysotile, although it has approximately the same heat-resisting qualities. Amphibole differs from chrysotile chemically in having lime combined with its magnesia, and while the chrysotile, a silicate of magnesia, is always hydrated, amphibole is frequently anhydrous.

*Reprinted from *The Electric Journal.*

It will be noted in Table I that the composition of chrysotile is practically identical with that of the serpentine rock from which it is formed.

Table I. Typical Analyses of Rock and Fiber

Chemical Constituents	Serpentine Rock	Italian Chrysotile Fiber	Canadian Chrysotile Fiber	Amphibole Fiber
Silica	40.34	40.30	41.90	61.82
Magnesia	43.32	43.37	42.50	23.98
Alumina	1.32	2.27	.89	1.12
Ferrous Oxide	1.23	.87	.69	6.55
Lime				1.63
Water	14.17	13.72	14.05	5.45

There can be little doubt but that there is a definite relation between the softness of the fiber and the quantity of water contained therein; 14.38 per cent. of water has been found in very silky fiber, while a harsh, brittle sample showed only 11.7 per cent. This possibly accounts for the extreme brittleness of amphibole asbestos, one sample of which, as indicated in the table, contained 5.45 per cent. of water. The effect of high temperatures on very soft fiber also demonstrates this fact. When part of the combined water has been driven off by excessive heat, the fiber loses its flexibility and becomes harsh and brittle, and the variations in strength and silkiness in various deposits of the mineral are best explained by assuming that the water content was originally nearly the same in all cases, and that the movement of associated rocks or the injection of igneous rock have furnished sufficient heat to drive off part of the water.

Physically, chrysotile has some interesting properties. In the absence of reliable data as to the diameter of its fiber, some microscopic observations were attempted. At about 90 diameters the subdivisions of the fibers appeared to be unlimited. The bundles broke up and branched off into numberless finer collections of lines. At 900 diameters magnification, fibers appeared which were barely discernible and which were estimated to be five one-millionths of an inch in diameter. There was, however, nothing to indicate that these were not capable of still further subdivisions, and, as suggested by A. Kingsbury, who very kindly made this examination, it would not require much elasticity of imagination to believe that the ultimate asbestos fiber is just one molecule in diameter. This seems reasonable, moreover, as in the case of mica, the cleavage is such that specimens have been split to a thickness equal to one-half wave length of distinctly violet light.

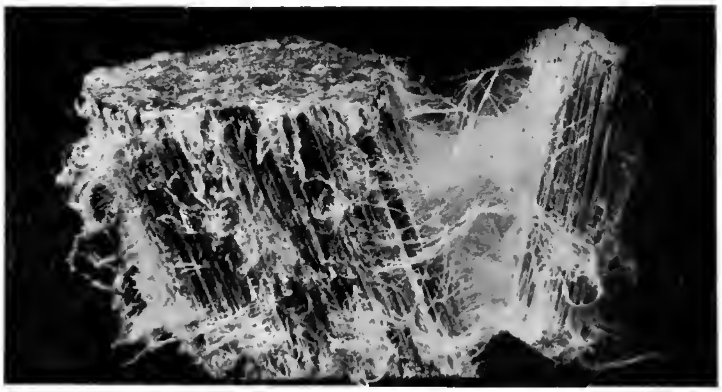


Fig. 2. Asbestos from Bell Asbestos Mines

Wool, cotton, silk, and other fibers, all have rough surfaces, and when spun into a thread this roughness aids in holding the mass together. While its fibers are almost infinitely finer than these, chrysotile has an absolutely smooth, glossy surface, that for a long time prevented the successful spinning of what was otherwise an ideal fiber. The overcoming of this difficulty

has made possible the spinning of a fairly strong thread weighing approximately an ounce per hundred yards of length.

Chrysotile has the following tints: Pure white, yellow, pale green, grass green, blackish green, blue, gray, brown, salmon, greenish blue, and lavender blue. When fiberized the color generally disappears, in most cases the flossy crystals appearing entirely white.

As a material in engineering, asbestos is unique. Soapstone has great heat resisting qualities and is a good electric insulator, but it can only be used where massiveness is permitted. Mica also withstands heat and provides excellent dielectric resistance. Its brittleness bars it from use as an insulator in many cases. Asbestos is to mica and soapstone what the line is to the plane and solid, and later references to its use will bring out this fact more in detail.

At least 75 per cent. of the world's supply of chrysotile is mined in the eastern townships of Quebec. The asbestos-bearing serpentine is scattered through a relatively narrow belt running nearly northeast from the boundary of Vermont to within about 40 miles south of Quebec.

On account of the general distribution of serpentine rock throughout the world, it is not surprising that deposits of chrysotile should be discovered in widely distant places. If the quality were always good, the available useful supply would be much greater than it is. Unfortunately, most of the fiber is dead and not suitable for spinning and weaving. The United States has many deposits, but they are generally of inferior quality. The largest supply of usable asbestos has come from the Sall Mountain district in White County, Ga.; Dalton, Mass.; New Hartford, Conn.; Pinto Creek and Grand Cañon, Ariz.; Polk, Mitchell, Wilkes, and Yancey counties, N. C.; Stevens Point, Wis.; Gaspar, Wyo., and Orleans and Lamoille counties, Vt., have all produced more or less of the fiber. The deposits in Vermont form a part of the great serpentine region extending across the Province of Quebec.

Fig. 3. Carded Asbestos, No. 1 Quality

Newfoundland, in the vicinity of Port au Port Bay, has large and promising deposits, but is prevented by its inaccessibility from participating in the rapid development now in progress in the Canadian field.

Italy, up to 1877, produced practically all the asbestos, but its use was made almost prohibitive by the high prices charged. So soon as the Canadian product was exploited the Italian industry received a decided set-back; nor is it surprising, because Canada has a much greater percentage of good fiber that is more easily spun and woven than the Italian mineral.

There are extensive deposits in the Ural Mountains of Russia. At Ekaterinburg chrysotile of fine and silky fiber is found, which is valuable for spinning on account of its tensile strength. The work in these mines is done by Russian peasants, who receive for their labor from 33 to 38 cents per day, together with free sleeping quarters.

Mongolia, Siberia, Finland, Queensland, South Australia, New South Wales, and New Zealand, have deposits which have not as yet been extensively worked. A lavender-blue fiber is mined in South Africa, which differs from other varieties, not only in color but in specific gravity, which is lower. This fiber has great strength and other good qualities and is becoming a strong competitor of the Canadian product.

The production of Russian chrysotile has been pushed forward at even a greater rate than the African fiber. In 1902, 45 tons were shipped from the South African mines, and in 1909, 2,000 tons were exported, at an estimated value of $135,000. In 1907 the Russian mines produced over 10,000 tons, as compared with 1,000 tons in 1900, and they are rapidly increasing their annual output. The startling growth of shipments from these two sources of supply is furnishing food for thought on the part of the Canadian producers, whose annual deliveries have only about trebled in 8 years.

According to a recent estimate, the world's annual consumption is not over 100,000 tons of all grades of chrysotile. Since 1877, when the Canadian industry was founded, the Province of Quebec has supplied at least 75 per cent. of this, and asbestos mining is without doubt the most important mineral industry in the province. To within a very short time, the mines have been owned and operated by a considerable number of separate concerns. Recently the Amalgamated Asbestos Corporation, Ltd., has drawn together a large majority of these concerns, and with a capitalization of $25,000,000 controls from 50 to 70 per cent. of the world's supply of asbestos. In the opinion of H. Mortimer Lamb,* this merger has grossly overcapitalized a fine industry. The rapid growth in foreign fields is cited as an indication that the Canadian producers may not always enjoy their recent monopoly.

The immediate result of the merger has been unprecedented activity, and the opening of new mines and the erection of new plants is going rapidly forward. Some of the larger mines are working 24 hours daily, using electric searchlights for illumination at night. An Asbestos Bureau has been instituted, its object being to keep the public advised of the new uses for asbestos, and to report the development of asbestos properties the world over.

Excepting a few underground workings, asbestos is mined in open pits. When consistent with thorough work, the barren rock is left standing and the fiber-bearing rock removed. In many cases, however, the useless rock is carried to the dump and the pit left unobstructed. While tunneling work can be continued throughout the year, regardless of weather, it is not well adapted to asbestos mining on account of the irregularities of the deposits and the amount of waste made necessary in order to have suitable pillars and supports. The beginning of an asbestos mine is no small undertaking, it sometimes being necessary to remove a layer of soil 15 to 25 feet deep before actual quarrying can be started. In some cases the steam shovel is furnishing excellent results as to speed. It may also be suggested that an opportunity is offered for the use of electrically operated shovels where electric power is available or may be developed. In quarrying, the serpentine is attacked with air or steam-driven drills, that make holes from 8 to 15 feet deep. About ½ pound of dynamite is used for each foot depth of hole. Three cents worth of explosive brings down about 1 ton of rock. Electric batteries are used for firing the shots. The broken rock is next sorted and lifted from the pit in boxes and placed in cars, the barren rock going to the dump and the fiber and "fines" to the mills for further treatment.

* In the *Canadian Mining Journal* of January 15, 1910.

To convey some idea of dimensions, a typical mine might be referred to. This mine is 700 feet long, 200 feet wide, and 165 feet maximum depth. Cable derricks are placed along one side, and the transport boxes of rock are hoisted and carried to the side of the mines by these derricks.

The dressing of asbestos consists in removing the adhering rock, by cobbing the long fiber, and machine treatment of the shorter stuff. No. 1 crude chrysotile fiber measures over three-quarters of an inch in length, and is worked up by men using 6- to 7-pound hammers. The sorting work is usually done by girls, who break and pick out the fiber.

Putting chrysotile fiber into suitable condition for spinning is by no means an easy problem. To find methods of separating each fiber from its neighbor and from the small pieces of rock has taxed the ingenuity of asbestos engineers. A number of plants were failures because complete fiberization was not accomplished. Successful extraction of fiber from the ore dates back barely 15 years, and now the greater quantity of Canadian asbestos is fiberized at the mines.

As a preliminary step the ore is heated sufficiently to expel all water adhering to the fiber. When dry, the mineral passes through crushers which break and grind the bundles of fibers until they are pulled apart. These crushers are, essentially, magnified coffee mills. For still finer separation the fiber passes between rolls giving direct pressure to the bunches of fiber which at this stage are rarely more than three-quarters of an inch in size. After these crushing operations, the fiber is mostly in the form of small lumps, which must be further broken up and put into a flossy, feather-like condition. This is accomplished either by cylindrical beaters or "cyclones." The beaters have rapidly revolving arms with teeth at their ends mounted within a cylinder. The rapidly moving teeth tear the fiber apart and the finished floss is passed out of the cylinder at an opening opposite its entrance point. The cyclones have propeller-like beaters placed opposite each other and revolve at 2,000 to 2,500 revolutions per minute. The lumps of fibers are fed in so as to strike these beaters, and are hurled against each other with sufficient force to be torn apart and fiberized almost immediately. An exhaust fan draws out the fibers and carries them to a shaking screen, which takes out the remaining lumps. Suction fans carry the finished fiber to collectors and settling chambers. It is sometimes found desirable to pass the crushed ore under magnets to extract small particles of iron.

Materials of a fibrous nature enter very largely into the useful arts. As organic fibers, generally used for such purposes, must be kept at a safe distance from fire, it is natural that intensive engineering should welcome asbestos which assists in space economy and in fuel economy as well. Probably the most striking example of its value is the use of boiler and steam pipe coverings made from asbestos and plaster. A conservative estimate places the fuel saving resulting from this protection at 25 per cent.

The modern fire fighter can accomplish very much more than was formerly possible, because of his fireproof equipment. His garments, boots, gloves, helmet, mask and respirator (all asbestos), permit him to remain in direct contact with the flame for a considerable time. Asbestos theater curtains have become a necessity, and asbestos rope makes possible the saving of property and oftentimes life as well.

Rubber sheeting, strengthened by a web of asbestos, makes ideal steam packing, while for the insulation of cold storage chambers and ammonia pipes, asbestos felt, both plain and corrugated, is used. For general heat insulation around stoves and furnaces, asbestos mill board is available. This is made by one manufacturer as follows: Asbestos fiber is placed in beating engines, with water, where it is thoroughly worked up with a suitable binding cement, and stored in tanks, from which it is fed to the paper machines. These machines have cylinders of fine wire gauze, upon which the pulp is collected, excess water pressing through the gauze. The thin coating of pulp passes from the gauze cylinder on to another drum, and when a sufficient thickness has collected it is cut across and removed. This moist square of mill board is pressed by hydraulic machinery and dried.

Asbestos enters into many materials of architecture. Fireproof brick, wall plaster, floor tiling, shingles, roofing felt, and so-called fireproof paints all have asbestos in their composition, while gas grates are faced with clear asbestos fiber.

In the field of electrical insulation, asbestos gives the fireproof fiber necessary to hold together certain other insulating materials. The gums, such as asphalt, rubber, shellac, and Bakelite, are molded with asbestos, and the fiber adds strength to what would otherwise be too brittle for practical use. Asbestos wood or lumber is being used for switch bases and small switchboards. When properly waterproofed by impregnation with asphalt, this material is entirely suitable for these purposes. The unimpregnated lumber has remarkable arc-resisting qualities, and gives excellent satisfaction when used in switch boxes.

In addition to its use in molded insulating pieces, asbestos is used to insulate copper conductors in coils where the temperatures are relatively high. Field coils for railway and other motors have their life decidedly increased in this way, as a greater temperature rise may be allowed than is permissible when ordinary cotton insulation is used. For this purpose round wire is wound with cement, which makes a fairly homogeneous covering. This wire and flat ribbon similarly insulated are used to some extent for armature windings. Field coils made from edgewise or flat wound copper straps are insulated by winding strips of asbestos paper between turns of the bare conductor.

FIG. 4. SPINNING DEPARTMENT, THE UNITED ASBESTOS COMPANY'S WORKS, LONDON, ENGLAND

The fire underwriters require the employment of asbestos as a non-inflammable covering for conductor cords used as leads for electric-heating apparatus. Copper cable is loosely insulated with asbestos thread and one or more strands thus protected are encased in a cotton sleeve. The destruction of this outer sleeve would not in any way damage the insulating qualities of the asbestos.

Notwithstanding the uses just indicated, it must be said that asbestos has very serious limitations as an all-around insulating material. Asbestos fiber is not as strong mechanically as other fibers ordinarily used for insulating purposes. It cannot be used as an outside covering when it will be subject to abrasion. Mass for mass, the insulating value of asbestos is relatively low compared with the best insulating materials, such as mica or

varnish-treated cloths. It is also true that molded insulating materials containing asbestos do not compare favorably in dielectric strength with those which are made without it. A possible explanation lies in the fact that the relative conductivity of asbestos is greater than that of other fibers. Another explanation is that a compound of two materials having a wider divergence in specific inductivity capacity will not be as good an insulator as a compound of two materials having capacities more nearly coincident. Asbestos fiber is more likely to collect moisture than the other fibers, thus reducing the resistance of the mass of which it forms a part. It is probable that all three of these conditions combine in making asbestos a less valuable electrical insulator than many other materials used for such purposes.

Fig. 5. Process of Drawing, Doubling, and Combing Asbestos, Mill of Keasbey & Mattison, Ambler, Pa.

Notwithstanding its limitation as an insulating material where high voltages are in question, asbestos will continue to be used to advantage for moderate voltages. In the application of asbestos to electrical work as well as in its common application as a heat insulator, great progress has been made in the last few years, and without doubt new needs will continue to develop and the demand for this valuable mineral will be greatly increased during the next few years.

EARTH COLORS OF SARDINIA

Earth colors are mined in Sardinia, at Capo Rosso, Capo Becco, and Cola Figu. The annual production of red oxide, both light and dark, ranges from 3,000 to 4,000 long tons. The selling price of the light variety is $7.72, while the dark-red oxide sells for $9.65 per long ton f. o. b. Carloforte. These prices are for shipments not exceeding 100 long tons, when sold in bulk. The price is higher when the earth is packed in sacks. The chief exportation of these red oxides is to England, though some small quantity finds its way to Germany.

Thousands of tons of dark yellow and light yellow oxides are also mined every year and sell at the same price and on the same conditions as the red oxides. These yellow oxides when treated with calcium are said to give a fine quality of red. The same mines yield a jasper that has the same color as yellow, red, and violet ocher, and is, at times, extracted in blocks that measure a cubic yard. Violet ocher is also mined, the selling price of this at times reaching $77.20 per long ton. Manganese is also obtained from the mines at Capo Rosso and Capo Becco, to the extent of 3,000 long tons per annum. This product contains from 35 to 45 per cent. of metals, 10 to 12 per cent. of silica, and 10 to 12 per cent. of iron.—*United States Consular Report.*

JOPLIN ZINC ORE NOTES

Three concentrating plants, each having a capacity of 500 tons per day, are under course of construction in the Granby, Newton County, Mo., camp, and considering the large amount of development being done in that camp, the production of zinc blende, calamine, and lead ores should be materially increased during the ensuing year. Two of these large mills are being built by an eastern company, represented in this district by Axel Ulhorn, who closed a deal recently whereby the Mascot and the Homestake mines were bought from Neosho, Mo., bankers, for $75,000 cash. This was the largest deal of many months in the Joplin district. Both of these mines are large producers of zinc and lead ores, despite the fact that they have been equipped with mills of only 100 tons capacity per day. The Mascot is on a 30-acre lease of the Granby Mining and Smelting Co.'s land, while the Homestake is on a 30-acre lease of the Davis land adjoining. Operations are conducted at a depth of 200 to 250 feet. The third of the big mills is the one being built by the Granby Mining and Smelting Co. to take the place of its large custom mill destroyed by fire the latter part of January.

The annual report of the M., K. & T. Railroad shows that $25,000 was paid in 1910 for mine tailing from Galena, Kans., zinc and lead properties. The tailing was used in making better roadbeds for the railroad.

The Golden Rod mill in Galena, Kans., is to be moved to a lease of the Granby Mining and Smelting Co.'s land in the west part of Joplin, where it will be placed in operation by the Postal Mining Co., a new concern of Galena.

Henry Weymann, of Joplin, Mo., has purchased the 320-acre tract of the Bonanza Mining Co., at Galena, Kans., and several new companies are starting work on this property. The ore here, for the most part, occurs near the surface and is mined as a rule by small concerns or individuals, whose capital is limited. It is one of the typical "poor men's" tracts of the Galena camp.

The name of the Little Jew Mine in the West Joplin camp has been changed to the Little Jewel by Clarke Marshal, who recently purchased this old-time zinc producer. Mr. Marshal has remodeled the mill, increased its capacity, drifted east instead of west as formerly, and encountered better and more extensive ore bodies. The mine is now ranking well up toward the top of the producers of the Joplin camp, the weekly production ranging from 80,000 to 120,000 pounds of blende concentrate.

Fig. 6. Weaving Department, United States Asbestos Company's Works, London, England

THE CALAMA COPPER DISTRICT, CHILE

*Written for Mines and Minerals, by F. A. Sundt**

Large Deposits of High Grade Copper Ore. Conditions Under Which It Is Mined

The Chuquicamata copper deposits in Antofagasta, Chile, are reached by taking the Bolivian Railroad to Calama, thence to Punta de Rieles, from which point it is 3½ kilometers to the mines. The mineral zone at Chuquicamata has a length of about 4 kilometers, with southern end but little worked, although it is a very important part. In Chuquicamata there are veins containing copper and also irregular mass deposits termed "Llamperas," meaning brittle mineral deposits. Both kinds of deposits are being prospected and worked.

As a rule the fissure veins strike north and south and are parallel, but come together at intervals along their course. The average width of the veins is 1.51 meters, but where two or more unite the combined width may be 8 or more meters; and, what is more unusual, at their junction the ore continues as rich as elsewhere, and sometimes increases in its percentage of copper.

After joining, the veins separate again and as before take the same general direction until the next junction point is reached; and only in exceptional cases are they known to cross each other. From this and other features it is presumed that the fissures and vein filling were contemporary. The hill through which the veins pass is composed of an acid white rock in which the quartz and monoclinic feldspar can be detected with the naked eye. Basic minerals, particularly the dark iron-bearing ones, are seen only occasionally, which would intimate that the rock was a sort of micaless pegmatite. Judging from its appearance, the gangue accompanying the copper is of the same general composition as the wall rocks, although it is softer owing to the weathering and former circulating ore solutions. The copper ore is found first on one wall side, then the other, and sometimes it occupies the entire width of the vein, and then only a part; but it is continuous and not in bunches here and there along the vein. Occasional but unimportant faults are met, being short, horizontal, and normal, and as the resistance of the rock increases with depth, the faults are not found in deep veins.

In the Panizo Mine of the Compania Poderosa de Chuquicamata, there are two faults. One is produced by a diagonal vein which is barren to the west and mineralized to the east; and the other originates through a change in the nature of the rock. All veins from east to west in contact with the main veins are mineralized for a distance of 20 meters, but those branches from the west to the east of the main veins are barren. Some of the Panizo veins are contorted into the form of an S, that has rich ore in one curve and poor ore in the other. The richest veins, called San Gregorio and San Manuel, continue from the Panizo Mine into Rosario del Llano Mine of the Compania de Minas y Fundicion de Calama.

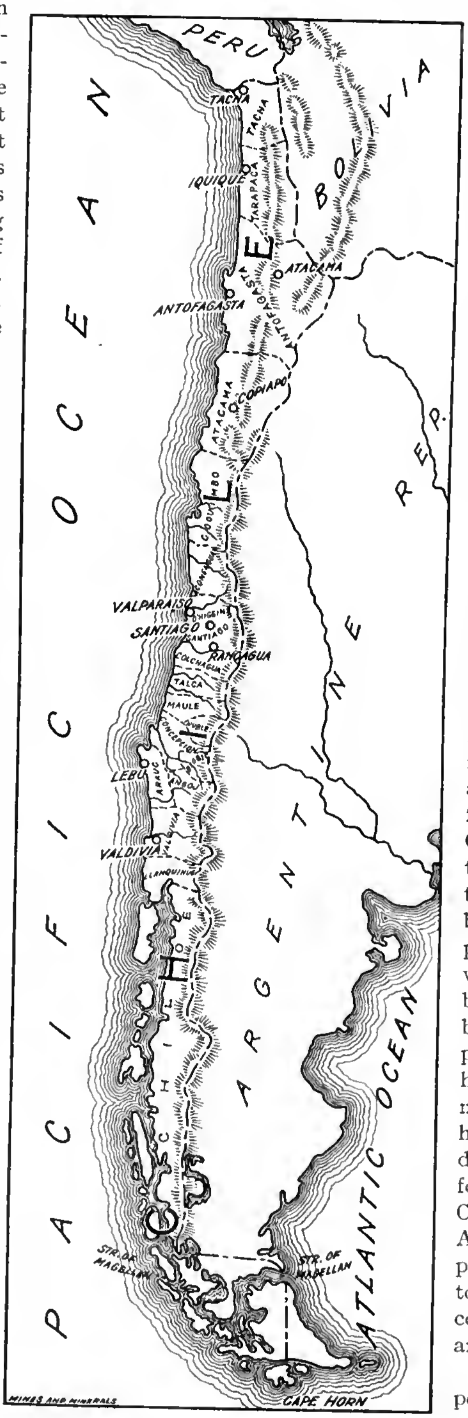


FIG. 1. MAP OF CHILE

Most of the Chuquicamata copper minerals at present worked are oxidized secondary enrichment ores near the surface. Atacamite $CuCl_2+3H_2O_2Cu$ and brocantite Cu_4SO_4+3Aq are found in abundance. Owing to their similarity in color and fibrous structure these two minerals are easily confused. A rosy-colored copper mineral having a similar structure and orthorhombic crystallization is found in some places, and it may be that all three minerals are derived from each other. Copper sulphates, both simple and complex in composition, are found near the surface, and as they are soluble in water, they are a loss when concentrating other copper minerals with water. Copper carbonates are not abundant in this district, although silicates of copper are found quite frequently. At depth the copper sulphides are encountered, but the zone of oxides and that of sulphides is not well marked, in fact one vein may carry sulphides and a parallel vein oxides at the same depth, although both have similar vein materials.

The sulphides most commonly found are chalcocite Cu_2S and covellite CuS. The gangue in the latter instance carries considerable oxide of iron. The chalcocite is purer than the covellite. Bornite and the purple sulphides are not found, at least so far as work has proceeded, although in both the Panizo and Rosario del Llano mines iron pyrite poor in copper appears at a depth of 160 meters. Chalcopyrite free from pyrite has not been found in Chuquicamata, which is peculiar, owing to there being large deposits of pyrite containing less than 6 per cent. copper in both of the mines. Pyrite of this kind is particularly valuable in this district, where matte smelting is carried on and blister copper made, owing to its furnishing both the iron and sulphur needed in the process. None of the Chuquicamata mines have been worked to any great depth, most attention being given to their horizontal extension; consequently at a depth of 180 meters no water has been found, except in small quantities in the Constancia and Flor del Bosque mines. At these mines the water contains sulphates of copper and is removed in carts to a mill where it is employed for wet concentration, it not being suitable for any other purpose.

It is to be remembered that the copper district under discussion is in a desert and all water has to be brought to the mines, where it sells for $10 per ton.* The little veins of water, whose origin is obscure, not being surface water, disappear at greater depth. However, the small quantity found is of sufficient value in this district to be saved.

Those mines which belong to the Poderosa company have been developed systematically by a series of shafts and levels in order to block out the ore. The San Filipe shaft, at the Poderosa Mine, has a cross-section of 2×3 meters, and a depth

* Injeneiro de Minas, Professor Estraordinario de Metalurjia en la Universidad de Chile, Santiago, Chile.

* The translator is not sure whether the prices given are in gold or silver.

of 170 meters. Like the other vertical shafts of this company, it is located so as to cut the vein at certain depths and then deviate from the vein, which must then be reached by driving cross-cut levels from the shaft. At the San Filipe shaft the vein is cut at a depth of 40 meters. That the men may not be subjected to accidents in going in or coming out of the mine, special shafts are put down that they are compelled to use, and this plan reduces hoisting accidents to a minimum. So far the Cia. Poderosa has confined its mining to developing and exploring its property, by which method it has kept up shipments and has 30,000 tons of ore blocked out that carry more than 10 per cent. copper. When rooms are worked, underhand stoping is followed, and this is entirely feasible and safe because no timbering is required except for ore pockets and chutes, and often for these uses timber is unnecessary, as the ore can be worked down the stope floor to the cars as needed. Dynamite and gunpowder are used for breaking rock. So far rock drills have not been used; however, the management appreciates the advantages to be derived from their use, and it is due to adverse conditions that a rock drill plant is not installed. Transportation is carried on by cars wherever feasible, as there is nothing much more expensive in the district than manual labor.

The Panizo Mine has a steam hoisting engine of the two-cylinder single-expansion type. It has a double cylindrical drum and is capable of hoisting with 700-kilogram cages from a depth of 300 meters.

Steam is furnished by a Babcock & Wilcox boiler such as is shown in Fig. 2, working under an average steam pressure of 4 kilograms per square centimeter, although the maximum pressure may reach 8.5 kilograms per square centimeter. Steam is generated by bituminous coal that is bought in sacks containing from 65 to 70 kilograms and costing $3.50 per sack. Four sacks, approximately 560 pounds, are used daily. The boiler consumes 600 liters of water in 10 hours, and probably what is obtained by condensing the steam. Peat was tried for fuel but did not furnish satisfactory results.

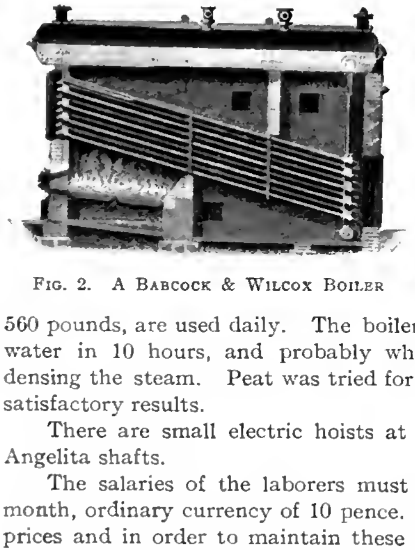


Fig. 2. A Babcock & Wilcox Boiler

There are small electric hoists at the San Filipe and the Angelita shafts.

The salaries of the laborers must not go above $300 per month, ordinary currency of 10 pence. This is the basis of the prices and in order to maintain these figures a careful inspection of the pay roll is made and a rigid watch is kept over the laborers. Piece work is allowed only when it is of absolute necessity, and then the piece workers are carefully watched, and a duty of 15 per cent. of the production is collected from them.

The manager of each mine receives $500 per month; the laborer, $275; the mine boss, $280; the drillers $6 per day; and some $80 per full length meter; the carriers, $5.50; the outside workers, $6. The cost of production fluctuates about $50 per ton.

In July, 1909, the production was as follows:

Ponderosa	266.6 tons, with	38.32 tons of copper
Panizo	275.0 tons, with	30.00 tons of copper
Totals	541.6	68.32

The average copper was 12.6 per cent.

The ores are shipped to England when the copper exceeds 18 per cent., and when the copper is 14 per cent., they are sold to the American Smelting and Refining Co., at Tacoma, Wash., U. S. A. Whenever it is possible, the ores of lower copper percentages are concentrated.

The gangue is softer than the mineral, and that is the reason why the ground ores are superior. When they have a value of 6 per cent. or more, they are sent to the concentrating plant of Huamachuco, and from there they are exported.

Compania de Minas y Fundicion de Calama owns the Rosario del Llano Mine in Chuquicamata. This mine has been one of the greatest producers; its system of working is irregular, however, and it seems that the work has been carried out without sufficient forethought. Its maximum depth is 160 meters, and the horizontal area worked measures 200×200 meters. In some parts of it, iron pyrites with 3 per cent. copper have appeared, but in general the quality has decreased at that depth.

According to the manager's report, the production of this mine during the year 1908 amounted to 13,814 tons of ore, carrying 12.6 per cent. copper, or 1,741 tons. The value of the production of the last 6 months was $581,082.82.

According to the above-mentioned report the cost of the production, including the money spent in new works, amounted to $52.18, which is cheap.

The factors that increased the expenses were:

1. Greater development, exploration, and preparation.
2. Increase of the exploitation of the sulphides on account of the needs of the smelter, and therefore a more elaborate mining system was required, which practically forced the management to use the slicing and filling method of mining.
3. Mining by contract or lease, which is not entirely developed in an efficient form on account of the mine not having been prepared for it.

They have succeeded in reducing the amount of this system of mining until at present it corresponds to 54 per cent. of the total exploitation.

In order to keep down the cost, the following is practiced:

1. To develop as much as possible the practical mining, either by the company or by giving contracts per ton.
2. To prepare the deposits which are not yet in a condition to be worked, and remove them economically.
3. To limit the contract work to the points of difficult access where a mineral of high value lies, paying tariffs in accordance with the market value of the metal, and protecting themselves by charging the contractors a high fee for the right of way to the mines.

The reserves amount to 3,385 tons of minerals of 12.7 per cent. copper.

The price per longitudinal meter in the last 6 months of the year 1908 was $93.

Llamperas Deposits.—Besides the deposits of ore in vein form in Chuquicamata there are other copper deposits of irregular forms termed "Llamperas." "Llamperas" is derived from "Llampos," which means brittle mineral, because the mineral is reduced to a fine powder with a greater facility than the gangue. The Llamperas are found in one of the sides of the mineralized hill of Chuquicamata, at a generally higher elevation than that of the veins. Up to the present time no connection between the Llamperas and vein deposits has been found. An excavation of the Iberia Mine of the Compania Explotadora measuring 80 meters in depth over the brittle formation has not traversed it, nor has it cut through the hard rock passed through by the veins.

It is generally believed that the mineralization of the Llamperas proceeds from the veins with which they are in communication at some depth.

The ground where the Llamperas are found is broken and disintegrated, and inside of the cracks the copper ore is found in the form of colored minerals of brochantite, acatamite, soluble sulphates, etc. The ground is broken with heavy charges of gunpowder, and by means of wedges the mineral is separated from the gangue and crushed afterward to a fine powder. The rock averages 3 per cent. in copper. The crushed mineral is sifted, giving 15 per cent. copper. The sieves have

four openings per linear centimeter. The pieces of mineral containing only 2 per cent. copper are consigned to the waste pile.

The rock of the Llamperas is composed of quartz and monoclinic feldspar seemingly pegmatite in a semidisintegrated state.

The copper appears in the fissures of the rock and is not deposited with regularity. The rock forms strata of uneven and irregular inclinations and directions.

The Llamperas mines could furnish abundant mineral for concentration on vanning tables. The lixiviation of these minerals, as well as that of the poor minerals of Chuquicamata, is a problem to be solved, although concentration has given good results.

LIME REACTION IN CYANIDING

*Written for Mines and Minerals, by Theo. P. Holt**

In comparing the practice in different cyanide mills one cannot help being impressed with the large variation in the amount of lime used in different districts. In the gold mines of the Black Hills for example, from 3 to 5 pounds per ton of ore is considered good practice, in some of the Mexican plants on the other hand as high as 30 pounds per ton, and 20 pounds is not an uncommon figure. In fact, Mexican practice has been criticised by metallurgists of other districts for what they term "an excessive use of lime." They insist that it not only impairs extraction, but needlessly increases the expense of frequent treatment of filter leaves to remove the resulting deposit of calcium carbonate. But, as is usual with established mill practice, it is safe to assume that where there is a "way there's a reason."

The physical property of lime, which coagulates the slime of the ore and causes its rapid settlement, may or may not be an aid in treatment. This depends on the process used. In the Black Hills, where the separate treatment of sand and slime is in vogue, it has been found that the addition of lime makes the classification of sand from slime much more troublesome. The sand particles are coated with slime which makes percolation in the sand tanks difficult. In the "all-sliming plants," where no separation of sand and slime is made, and the rapid settlement of all the solids in the solution is the object sought, the coagulating effect of the lime is desirable. The decanted solution if not perfectly clear may be passed through a sand filter before going to the zinc boxes. The solid particles in such a solution will be attracted to the surface of the sand grains in the filter even though the interstices between the grains are many times the diameter of the particles.

The above conditions in some measure determine the small or excessive use of lime in cyanide mills. Combination sand and slime plants in general use low lime, while the all-sliming plants use high lime. It happens that mills treating silver ores are of the latter class and it is this, probably, rather than the considerations given above that are the chief reasons for wide variation that exists. There is a decided difference in the relative solubility of gold and silver in cyanide solutions of varying alkalinity. Silver is more difficult of extraction than gold, and in addition to finer grinding and stronger solutions, often requires the introduction of special chemicals such as lead salts to hasten the reactions. Lime is an active agent in a chemical way, increasing the solubility of most of the silver minerals. On gold it has the opposite effect. I have observed this fact in connection with experimental work on a large number of gold-silver ores. An increase of alkalinity will increase the silver extraction and decrease the gold extraction in almost every case.

These facts are clearly presented by a few results obtained on three gold-silver ores. The extraction of gold for different tests is represented in the graph, Fig. 1, by full lines, and the silver by dash lines. Samples No. 1 and No. 2 are slightly acid, while No. 3 is strongly so, consuming all the cyanide when less than 3 pounds per ton of lime was present in the test. The number of tests in each case are indicated by the points on the graph. A larger number of determinations would tend to smooth out the irregularities in the curves, but the principal characteristic is evident. For the solution of the gold values in an ore it is best to have only sufficient lime for protection. In other words, just enough to prevent the loss of cyanide by its union with the acid to form salts in the ore. More than this not only adds to treatment costs but decreases the amount of gold recovered.

The effect of an alkali on the solution of native silver is the same as on gold. The reason for this is not apparent. Mr. B. L. Gardiner suggests* that in a special case "it was due to the formation of $Fe(OH)_2$ which is a powerful reducing agent, robbing the solution almost immediately of its oxgyen and thus preventing the solution of the gold." He suggests that "the only remedy for this condition seems to be to remove the $FeSO_4$ by means of water washes preliminary to cyanide treatment, and the use of solutions of as little protective alkalinity as possible, even at the expense of increased cyanide consumption." It is easy to conceive how such conditions might exist in a leaching vat, but in an agitation tank with

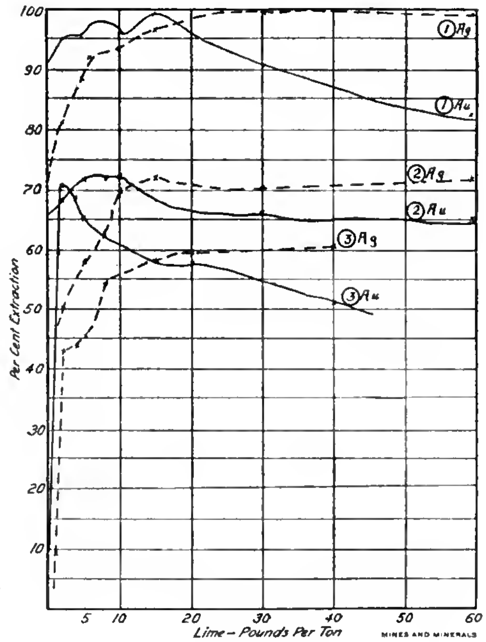


FIG. 1

the constant contact of the solutions with the air such conditions could hardly exist. The $Fe(OH)_2$ formed is quickly oxidized to $Fe(OH)_3$, and the solutions are immediately supplied with the oxygen of the air to supply that removed. A more tenable explanation seems to be that the lime forms a protective covering on the metallic surfaces.

The increased extraction observed for the silver minerals is in accordance with the well-known Mass law. Lime is an effective precipitant of soluble salts which otherwise accumulate in mill solutions and prevent the dissociation of silver compounds. A good grade of lime is soluble in the average cyanide solution to the extent of about 4 pounds per ton. Lime added in excess of this amount is carried in suspension as milk of lime but dissolves whenever the solution falls below the saturation point.

The most economical amount of lime to add in any given case is a matter of experiment. When the values in the ore are divided between gold and silver it often complicates the problem, as one metal can only be secured at the expense of an increased loss in the other.

* Mining Engineer, Salt Lake City, Utah.

*Journal Chamber of Mines, West Australia, October 1907

METALLURGICAL SLEUTHING

Written for Mines and Minerals, by E. B. Wilson

PITTSBURG, PA.

"DEAR EDITOR:—Enclosed find the description of a new metallurgical process which it is said will revolutionize metallurgy. Several of us have about decided to go into the manufacture of 'Borindum,' but I thought I would obtain your opinion before final decision. Kindly send me your advice, together with your bill, and oblige."

Showing the Value of the Application of Deductive Reasoning to Solve a Metallurgical Mystery

The above letter, which, except for the last sentence, reminded the editor of many others that he had received, was read with interest, and he turned then to the enclosed prospectus which was as follows:

BORINDUM

An inexpensive secret compound called borindum has been invented for the treatment of gold, silver, copper, and lead ores and for giving toughness to steel and iron in connection with the Buckenlach inventions of automatically operating machinery. By the various Buckenlach processes a recovery of 97 per cent. to 98 per cent. is effected without the making of slimes. After pulverization the ore pulp passes automatically into a tank containing a solution of borindum, from whence it is poured through a series of gold-saving taps, having been previously scoured by the solution, thus leaving the metals in a state to be readily taken up by the mercury which is rendered extremely powerful by the presence of borindum. Not only is the affinity of the mercury for gold and silver greatly increased but the tendency to flour, foul, or oxidize is prevented, so that waste or loss of the mercury is impossible and its action is made continuously "quick" by a current of electricity.

When used in connection with cyanide, chlorine, and other well-known chemical processes the dissolving of gold and silver and their precipitation become rapid and continuous, with increased percentage of recovery.

The Buckenlach method of recovery for copper ores containing gold and silver provides for the rapid dissolving of the values in a borindum solution and their simultaneous precipitation by both scrap iron and electric current, the copper being deposited in one place, the gold in another, and the silver in a third place automatically.

For use in connection with iron and steel, by a chemical treatment of the iron in the cupola furnace, borindum will produce homogeneous castings free from all blow holes and with its tensile strength increased 25 per cent. This combination will be invaluable for car wheels and axles where great tensile strength is needed. It greatly purifies the iron of titanic oxide and sulphur and phosphorus. It makes clean iron of great toughness and strength and of very close grain. The same advantages may be had in steel and steel castings.

REPLY

DEAR SIR:—Your communication regarding the inexpensive secret compound called borindum, with the request that I look into the merits of the process and inform you whether it can be used at your lime mine is received.

I congratulate you on the wisdom you show in first consulting a metallurgical sleuth before investing in the new process. The compound being secret we must approach it cautiously and ascertain if possible by deductive reasoning whatever it is. All the more cautiousness is needed because it is stated that it is for the treatment of gold, silver, copper, and lead ores, and for giving toughness to steel and iron in connection with the Buckenlach inventions of automatically operating machinery. By not running risks, we learn that borindum has not the entire responsibility in this matter, and that there are various Buckenlach processes which recover 98 per cent. without making slimes, the other 2 per cent. being graft most likely on the part of borindum. We found from reading about borindum that "after pulverization the ore pulp passes automatically into a tank containing a solution of borindum, from whence it is poured through a series of gold-saving taps, having been previously scoured by the solution, thus leaving the metals in a state to be readily taken up by the mercury which is rendered extremely powerful by the presence of borindum." This being somewhat involved, we again were obliged to proceed cautiously, and by this method we ascertained that borindum was a beast.

Secret-process sleuths know that an inventor, after inventing, invents a name which will apply to his invention, and since the inventor's invention is uppermost in his mind, the chances are the inventor furnishes a clew to his invention when inventing a name for his invention; for example, should you go into a drug store and ask for "blockade runners" you would not be handed a gunboat. Dissecting now the name borindum, and correlating, we find that the money put in secret processes is called "*mud*," and the investor claims the inventor has been "*robin*" him. If now the first and last syllables are spelled backwards, and the last syllable prefixed to the first syllable, and the middle syllable suffixed, the word "borindum" will spell mud-rob-in.

No self-respecting secret-process Sherlock considers his work finished with one simple piece of mind reading, and therefore we look up the word *bor*, and find it is derived from the Gaelic *boar*, and means a he hog, so called because he bor-in-mud. Arrange the syllables *ad libitum* and a second time we have deduced the word *mudrobin*; moreover, we have discovered it is a beast, for the inventor says that after pulverization the ore pulp passes automatically into a tank containing a mudrobin, who proceeds to recover 98 per cent. without making slimes. Nothing but a hog would eat 98 per cent. of the values; but hist! why is a tap "automatically scoured?"

Any secret-process inventor who fully explains his invention leaves the investor in doubt as to his reliability; besides most inventors follow the poet's advice, to wit:

"When reaching for a cactus bud
A prick or two pertains."

The inventor leaves the investor and investigator in doubt as to whether the metal in the previously scoured gold-saving taps is readily taken up by the mercury or the metals in the pulp; however, he assures the reader that mudrobin renders mercury extremely powerful, and that "not only is the affinity of the mercury for gold and silver greatly increased, but the tendency to flour, foul, or oxidize is prevented, so that waste or loss of mercury is impossible, and its action is made continuously 'quick' by a current of electricity."

The matter apparently is becoming involved, for (1) the inexpensive mudrobin is said to do the recovering; (2) the mercury is said to do the recovering after the pulp has run through the previously scoured taps in the presence of mudrobin; (3) it is the continuously kept "quick" mercury by a current of electricity that does the recovering.

We find then that the secret process is merely a secret in name, and that mercury is the main and only recoverer of the 98 per cent., which makes the inventor resemble that beautiful stanza:

"When coyote gambols on the plain
The partridge seeks the lea;
The fool beast in chasing after him
Gets a cactus prick in his knee."

Having disposed of the first paragraph, we turn to the second, which reads as follows: "When used in connection with cyanide, chlorine, and other well-known chemical processes the dissolving of gold and silver and their precipitation becomes rapid and continuous, with increased percentage of recovery." In regard to this statement, individually I would not accept it without making the inventor disclose and prove what the other well-known chemical processes were.

We now come to the following: "The method of recovery for copper ores containing gold and silver provides for the rapid dissolving of the values in a borindum solution and their simultaneous precipitation by both scrap iron and electric current,

the copper being deposited in one place, the gold in another, and the silver in a third place automatically." Apparently the mud-rob-in has his own sweet way in this case until he meets and has a scrap with the iron, which takes the copper away from him automatically, after which the electric current swipes the gold and silver. Evidently the action affected by mudrobin is equally acid and basic, similar to that produced by feeding hogs on clams for a considerable period, when they will automatically swell up and collapse as the tide rises or falls.

When a mudrobin can swallow or dissolve the values, and simultaneously precipitate them, the reaction is bound to be continuously automatic, or the circulation would be stopped and mudrobin choked to death instead of Buckenlach. Of course you are aware that all precipitation is not automatic, and that it requires the mixing of different solutions, or a solid substance, or possibly an electric current, or something else either manual or mechanical. You are also aware that eruginous microbes are in solutions of copper containing gold and silver and that they are poisonous, and this naturally raises the question concerning the expense connected with the inexpensive secret; in other words, what is the longevity of mudrobin.

The inventor does not furnish sufficient data on the number of gallons of saliva required to dissolve and simultaneously precipitate, without which the life of a mudrobin is indeterminate. You should obtain his figures.

Much criticism was given the mill men on the Comstock because they used tobacco juice in their pans. The mill men never answered their critics, but laughed up their sleeves at the wiseacres who were unable to understand that tobacco juice looked better in the pans than on the mill floor, and was more efficacious, since saliva will dissolve anything.

If what the investor relates in the last paragraph were true, then the United States Steel Co. must go out of business, and this is too good to be true. This person Buckenlach is the first inventor to invent an invention that so closely approaches the *vade mecum* of the metallurgist, and that can be adapted to fire or water. To place this automatic continuously-kept-quick hog before intelligent men shows his homogenous cast-iron brass is full of blow holes with 25 per cent. increase tensile strength in the blast. My advice to you, sir, is to let borindum severely alone; for a solution that will only greatly purify iron of titanic oxide and sulphur and phosphorus and will dissolve all other metals, is without doubt devilish. If you intended to invest $25,000 in borindum before consulting me you can send your check for $250; if, however, you did not intend to invest in borindum, but simply consulted me out of curiosity, you can send your check for $2,500; my time is valuable and not to be trifled away.

Yours very truly,

来 来

SOLUBILITY OF GOLD IN POTASSIUM FERROCYANIDE

Finely divided gold dissolves slowly in potassium ferrocyanide solution at the ordinary temperature, and forms a distinctly alkaline solution; solution is very slow even at boiling point. Apparently potassium aurocyanide is first formed, and the liberated ferrous ions are oxidized by the atmosphere to ferric hydroxide, which is precipitated, in amount corresponding to the amount of gold dissolved. Prolonged action of a strong solution of potassium ferrocyanide seems to give ultimately potassium auricyanide.—*Monats. Chemie.*

SMALL AIR COMPRESSORS AT MINES

Written for Mines and Minerals

Nearly every mine manager has desired at times an air compressor that might be used to tide him over temporary difficulties; and nearly every mine manager has wished for an air compressor to do work that is done under difficulties by manual labor.

That such machines are not in their equipment is due to the size, weight, and cost of the machines; the expense that must be incurred in their installation, their upkeep, attendance, etc. Realizing the desire that there is for small air compressors among mining men for lifting ore gates, running small pumps intermittently or an occasional rock drill, for running a fan or possibly for ventilation in bad places, for running an engine intermittently at the shop, etc., manufacturers of air compressors are working out the problem. Small machines such as shown in Figs. 1 and 2 are not as apt to receive the same careful attention as larger and more costly machines, while the work demanded of them is no less wearing. The manufacturers realizing this have worked along the lines of the three cardinal virtues of machinery, strength and hence durability; simplicity in construction and parts, features that appeal to the users; and efficiency, which in compressors involves the construction of the air cylinders, valves, and the cooling arrangements to a greater extent probably than the power end of the machine. Fig. 1 is the sectional view of a smaller steam-driven air compressor which mounted on its own bedplate can be moved about and set up without great expense; further, in case it is to go into the mountains, it can be readily taken apart for transportation and almost as readily be assembled. It will be understood that the air cylinder is on the end and that the cylinder ends are water-cooled in addition to the cylinder. Both the inlet valves at the bottom of the cylinder, and outlet valves at the top are poppet valves arranged in a radial direction to the axis of the cylinder and placed close to the ends of the cylinder. This position minimizes the losses due to clearance and the air trapped by comparatively long ports after compression in the cylinder. To remove the valves it is not necessary to remove the cylinder heads as they can be reached by unscrewing the plugs shown in Fig. 2 on the outside of the air cylinder. This feature is one that appeals to users of air compressors as the entire valve can be removed and the parts thoroughly examined without trouble. The valves are so constructed that in case of breakage it is impossible for broken parts to enter the cylinder.

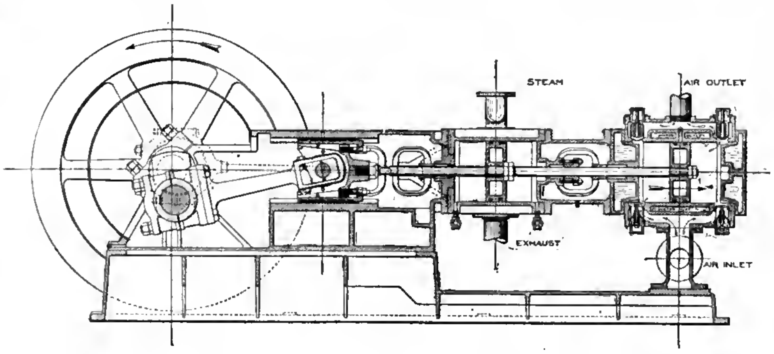


FIG. 1

Fig. 2 shows the steam-driven compressor in elevation, and the steam regulating device which places the throttle valve under the influence of the air-receiver pressure.

Fig. 3 is the section of a belt-driven compressor which may be operated by electric motor, gasoline engine, oil engine, or steam engine.

The power-driven machines are fitted when desired, with an unloading device placed on the air inlet of the compressor. This device is set to shut off the incoming air to the compressor when the receiver pressure exceeds a predetermined point. This enables the compressor to run at regular speed, an essential in power-driven machines, compressing no air until the receiver pressure again falls below the point set. This secures important reductions in the amount of power used when the compressor is furnishing air intermittently. The device is smooth and even

in its action, throwing the load off and on the compressor without sudden jar or strain.

The writer desires to acknowledge the courtesy of the Sullivan Machinery Co., of Chicago, Ill., for the illustrations and information so kindly furnished for this article.

DETERMINATION OF ALKALINITY

Written for Mines and Minerals

Among the various operations performed in the laboratory at a cyanide plant is one which deals with the alkalinity of the cyanide solution. A certain quantity of alkali must be added

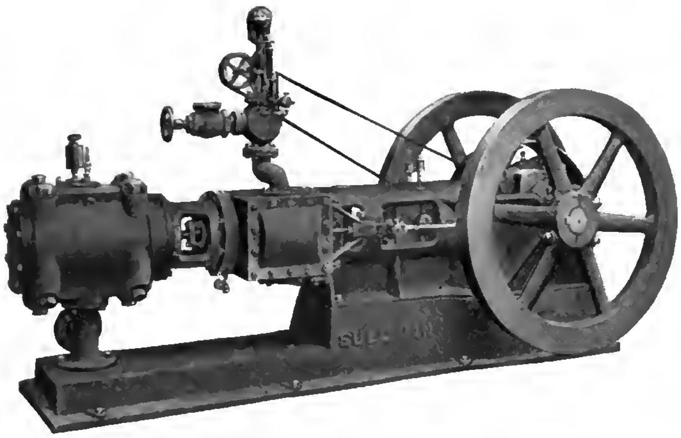


Fig. 2

to cyanide solutions to protect them from acid which may be in the ore. A normal acid solution is one so prepared that 1 liter of the solution contains 1 gram of replaceable hydrogen. Accordingly the normal sulphuric acid solution would contain

$$H_2 + S + O_4$$
$$2 + 32 + 64 = 98 \text{ grams}$$

were it not that there are two replaceable atoms of hydrogen in the sulphuric acid, hence $98 \div 2 = 49$ grams per liter, or .049 gram per cubic centimeter.

A normal alkali solution must contain the chemical equivalent, weighed in grams, of the hydrogen replacing metal or group of elements. For instance, the normal sodium hydrate solution must contain

$$Na + H + O$$
$$23 + 1 + 16 = 40 \text{ grams per liter}$$

or .04 gram per cubic centimeter, because the valency of sodium and hydrogen are the same. Equal quantities of any normal acid and normal alkali solution neutralize each other, and the quantity of any solution of a single acid or alkali that is neutralized by 1 cubic centimeter of a normal solution contains .001 gram of hydrogen equivalent of its active constituent.

To test the alkalinity of a cyanide solution, take 29 cubic centimeters of the solution and place in a beaker, next add silver nitrate solution until there is a slight turbidity. Phenolphthalein is then added and the solution titrated with standard sulphuric acid solution to the end point. Each cubic centimeter of the acid consumed is equivalent to 40 ounces Troy, or 53.9 ounces avoirdupois, of sodium hydrate to the ton. This is the equivalent alkalinity of the solution, and to protect the cyanide from "acid cyanicides," a trifle more alkali should be added. If too much is added, the white precipitates will be increased in the zinc boxes, a feature which should be avoided so far as possible. Caustic soda being expensive compared with lime *CaO*, the latter is used in practice, as it forms lime hydrate with water and neutralizes acid. It is customary to calculate the lime needed for cyanide solutions from the quantity of sodium hydrate used in determining the acidity in the ore. To test for acidity, an assay ton or 29.166 grams of finely pulverized ore is shaken in warm water for ½ hour, after that it is filtered and the solids on the filter washed with water. If the ore is acid the filtrate will turn blue litmus paper red; if it is alkali it will turn red litmus paper blue. Ores to be treated with cyanide solutions are usually acid, and it is therefore necessary to add lime. The molecular weight of sodium hydrate is 40, that of lime 56, but the valency of sodium is 1, while that of calcium is 2, hence $\frac{56}{40 \times 2} = .7$; that is, it requires seven-tenths as much lime as sodium hydrate to neutralize an acid. Assume the following weights of acid solution are taken for testing, and the following quantity of sodium hydrate is used in titration, the quantity of lime needed would be calculated as follows:

	Grams
Weight of beaker and acid	59.396
Weight of dry beaker	30.230
Weight of acid solution	29.166

	Cubic Centimeters
Final reading of burette	15.9
First reading of burette	13.4
Normal solution used	2.5

From the assumption, 29.166 grams of acid solution contains 2.5 cubic centimeters of standard acid, because normal acid and the alkali soda are equivalent, therefore each cubic centimeter of sodium hydrate corresponds to 49 ounces Troy, or 53.8 ounces avoirdupois, of sodium hydrate per ton of ore. The pounds of lime required to neutralize the acid properties in 1 ton of ore will be

$$\frac{53.8 \times 2.5 \times .7}{16} = 5.88 \text{ lb.}$$

SCANDIUM

Scandium has been found to the extent of more than 1 per cent. in the mineral wiikite from Finland, and in small amounts

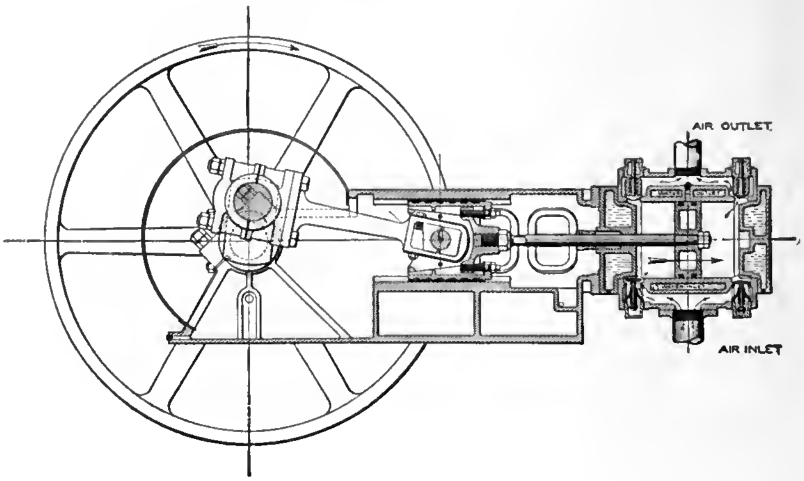


Fig. 3

in many other minerals. By a systematic series of fractionations it is possible to separate scandium from most associated elements; ytterbium is the most difficult to separate because its nitrate is decomposed almost as readily as scandium nitrate. In view of the atomic weight relationships, and of the frequency with which these elements occur together in nature, it is suggested that yttrium and scandium are degradation products of ytterbium. Scandium can be separated from ytterbium by precipitation as metanitrobenzoate. Sir W. Crookes. (Phil. Trans. Roy. Soc., A209, 15.)

TITRATING FOR PROTECTIVE ALKALINITY

*By H. L. Sulman and Frederick Reade**

The authors, under the heading "Errors in Testing Cyanide Solutions," disagreed with Bede Collingridge as to the effect of potassium iodide in titrating cyanide solutions for protective alkalinity, and experimented themselves. They found that the presence of potassium iodide obscured the end point. In presence of silver cyanide and potassium iodide, phenolphthalein fails to give a sharp end point; the indistinctness is a double one, due, principally, to the constant return of the phenolphthalein color indicating alkalinity after its discharge by standard acid to an apparent neutral point, and, to a minor extent (owing to the employment of unnecessary amounts of potassium iodide and silver solution), due to the masking of faint alkaline colorations by the iodide of silver suspension.

When using a solution one-tenth normal, the end point of the reaction is sufficiently sharp to give accurate results, but to determine small amounts of protective alkali in small quantities of highly dilute cyanide solutions (such as may be derived from the treatment of slimes by decantation) standard acid should be employed of a dilution commensurate with requirements, say .04 normal. Under such conditions the error becomes noticeable, and is further accentuated by the use of silver nitrate in excess in the first (or cyanide) titration. If, however, no potassium iodide be used, the results with the weak acid are accurate and the reactions normal.

DEL CARMEN MINE, WORKINGS ON HILLSIDE AT LEFT

The phenomena are explained as follows: "In the titration for cyanide, with silver all potassium cyanide (which would otherwise give its full alkali value on titration with acid) must first be converted into the double cyanide *KCN, AgCN*, a compound which is not decomposed by dilute acid in which the combined moiety of *KCN* does not therefore show alkalinity on titration with standard acid and an indicator.

"The end point of the cyanide reaction is, however, necessarily marked by the precipitation of silver cyanide as a cloudy suspension. This precipitated silver cyanide now reacts with potassium iodide, forming yellow silver iodide, and free potassium cyanide, of course with its equivalent alkalinity."

The authors conclude by restating Mr. Collingridge's proposition thus: "If small amounts of protective alkali are required to be measured by dilute standard acid solutions, the use of potassium iodide as an indicator for the cyanide reaction point will give rise to serious error in regard to protective alkalinity present.

"Further, the error will be greater in proportion to the overdoing of the cyanide end reaction with silver-nitrate solution, in presence of an equivalent addition of potassium iodide; this is due to the reaction between argentic cyanide and potassium iodide in producing free potassium cyanide."

*Bulletin No 67 I M M

BOQUILLAS ZINC DEPOSITS

Written for Mines and Minerals, by Carlos Moser

Only 4 miles south of the Rio Grande, in the state of Coahuila, Mexico, is a deposit of zinc carbonate, which has lately attracted the attention of all men interested in this metal. The western foot-hills of the Sierra del Carmen are very much changed, broken and distorted by a porphyry dike several miles long and from 100 to 200 feet wide, along which caves and fissures were eaten out of the lime-rock mountain by solutions and afterwards filled with lead and zinc ores of remarkable purity. About 20 years ago these deposits were first discovered and mining started by the old Consolidated Kansas City Smelting and Refining Co., which, of course, was only interested in the lead ores, which they at first shipped to their El Paso plant, but eventually smelted at the river's edge in the village of Boquillas, about 8 miles from the mines.

ZINC CARBONATE AT DEL CARMEN MINE

The lead ore was first discovered on top of the Puerto Rico hill, where it appeared in the form of galena in a chimney some 30 feet in diameter. Later the same class of ore was found in the Zaragoza hill, where the deposit at the surface was 40 ft. × 120 ft. in thickness. After some 8 years of mining these surface lead bodies were exhausted and the mines abandoned. Some 5 years ago the writer examined these abandoned properties and to his astonishment found the old stopes filled with carbonate of zinc ores and great masses of it reaching to the very surface on three sides of the chimney in Puerto Rico mine. Since then the properties have been developed to such an extent that it is claimed more than 50,000 tons, averaging 46 per cent. throughout, with less than 3 per cent. iron, and no other injurious minerals, are actually blocked out. As mentioned before, these deposits appear in chimneys and caves, the chimney in Puerto Rico being 220 feet deep, the area being about 30 feet in width by from 40 to 80 feet in length. At its bottom this chimney connects with a cave 700 feet long, running northwest, which was practically filled with lead ore and where the east wall is covered with the zinc ore spoken of. To the southeast a regularly formed fissure contains a mixture of iron, zinc, and lead of little value, until it reaches a deep arroyo, also some 700 feet distant from the main chimney. On the other side of the creek rises the Zaragoza hill with large deposits of

iron ore unexplored for nearly a mile, till the Zaragoza lead mine, now mostly caved, is reached.

The very steep slope of the main mountain to the east of the Zaragoza is covered with boulders of high-grade zinc ore, but on account of the high walls, rising in short terraces to 5,000 feet above the foot-hills, the source of this phenomenon has not yet been discovered.

Further, north of Puerto Rico, are strong outcroppings of zinc and small stringers of lead ores, but also not explored. Apparently there are a number of veins parallel to the porphyry dike, but their contents are unknown as they are principally barren, or nearly so, on the surface. The greatest drawback to the full development of this region is the long distance from a railroad. The nearest railroad point in Mexico is Cuatro Cienegas, over 200 miles away, and, of course, it is out of the question to haul the product from these mines such a distance. Last year a Leschen patent aerial tramway was constructed to transport the ores into the United States direct. This tramway is 6 miles long, 2½ miles in Mexico and 3½ miles in Texas.* From the Texas terminal of this cable it is exactly 80 miles to Marathon. A splendid highway connects these points, but common wagon freight and the high duty on zinc ores make it rather an expensive affair to make shipments now. There is

Puerto Rico Hill, Boquillas Zinc Mines

a movement on foot to build a railroad either from Marathon, Haymond, or Paisano on the Sunset line to Boquillas.

The lower section of Brewster County, in which Boquillas, Tex., is located, promises large freight resources, as ores of many kinds, cinnebar, lead, zinc, iron, and silver, also strong beds of first-class cannel coal; immense quantities of sotol; guayule, the rubber plant; and mariala, for the manufacture of linoleum, besides the productive wax plant, locally known as candelilla.

WISCONSIN ZINC ORES

The mining rights of the Jarrett Lead and Zinc Co., at Pt. Rose, Wis., have been transferred to the M. & D. Mining Co., of Madison, Wis. Drills will be put to work to prove the ground, and an attempt made to get the big plant built a year ago at a cost of $30,000, into commission once more. A fine showing of ore was made in the Jarrett during the progress of shaft sinking, and much was expected of this property. It lies along the same range as the Board of Trade and other producers in this district. Results were obtained from milling the stock pile, and further underground work was disheartening.

Another profitable producer is the Henrietta, and a good showing is being made by the Peaceful Valley Mining Co. With the Baxter mine in shape again by April 1, a good output of all grades of ores may be expected the coming summer.

The Frontier Mining Co. has purchased the power and concentrating establishment built for the Hoosier Mining Co., and a transfer will be made to one of the new Benton prospects under development. Much was expected from the Hoosier but the ground failed to make good. The Frontier has a very likely producer in the new Sedgwick and the plant may be rebuilt.

W. L. Thorne, the expert driller, is operating with two Keystone steam drills upon the mining allotment of the Fields at New Diggings. This mine is operated by R. N. Hunt & Co., of Chicago, under the direct personal management of R. S. Landers, of Galena. A new shaft is being sunk upon the Crawhall land adjoining. More extensive mining operations have been planned for the coming season.

The mill, machinery, and mine, at Dodgeville, formerly owned by the McKinley Zinc Mining Co., has been purchased at foreclosure sale by McKinley Bros., of Dodgeville, and they will continue to operate this property.

The New Jersey Zinc Co. continues its buying program, the last deal involving the Carrow farm, at Linden, the consideration being $25,000, which carries with it the mining rights to the allotment. The Rajah Mining Co., of Milwaukee, is located upon this land and this latest news would seem to indicate that the proposition has failed under the Rajah management. The surface equipment was brought over from the Trio mine and was known for a time as the Racine-Linden. The plant failed to give satisfaction and the development of the ground while showing zinc ore in disseminated formation made it plain that better facilities for handling the ore must be had before results could be obtained. More capital was required which the Rajah management tried hard to raise but failed. The Linden-Mifflin Mining Co. was none other than the New Jersey Zinc Co. and operated what was known as the Mason mine with a big power and milling plant. This was closed up some time ago on depleted ranges and it was believed that the surface equipment would be removed. Except for a portion of the machinery the plant remained intact and it is more than probable that this plant will now be made to do service for the Carrow mine.

The Drumm mine is shipping about a car of ore daily. With heavy shipments of ore for more than 6 months it begins to look as if this property would yield some measure of profit to the owners of the mining rights, the Benton Mining and Developing Co. The surface equipment is installed by the Wisconsin Zinc Co. and must of course be paid for out of the first net earnings of the corporation.

The American Zinc Ore Separating Co. finds the electrostatic method, the Huff process, of separating ores so profitable that additions are planned in the near future. It is proposed eventually to bring the capacity of the plant up to 1,000 tons weekly. Machinery is being installed with all possible haste at the Cleveland Victory Lead & Zinc Co. dry process plant on the Klondyke allotment. This process is entirely new and the outcome is being watched with much interest by mining men.

The Dodger Mining Co., at Mifflin, is installing a new hoisting and air plant. Drilling operations are in progress at the Cruson mine with variable results. Late reports from the Dall Lead and Zinc Co. are to the effect that the main ore body has been picked up once more east of the new shaft and that the outlook is now better than at any time since this portion of the mine has been developed. The Dall is one of the big producers of the field and at one time was one of the strongest dividend payers. The ore runs have been badly broken up, accounted for by the fact that the first mining was done in a draw where the ground is productive of numerous crossings and spur ranges. Sooner or later the Dall is destined to develop a large permanent deposit of zinc ore. The power and milling plant is one of the best in the field and is valued at $35,000. A new shaft has recently been completed.

* Described in Mines and Minerals for Feb., 1910, page 437.

COAL NOTES

Intensive Working.—The *Mining Journal*, of London, England, when commenting upon the Hulton colliery disaster of December 21, 1910, has this to say concerning intensive working: "Certain facts have been mentioned, such as the excellent ventilation of the Hulton pits and the up-to-date appliances for rapid working, as if they represented safety factors in the working of a mine. The better the equipment of a colliery for rapid production, the greater the amount of dust produced in the workings; and the freer the ventilation, the greater the opportunities for an explosion of dust, owing to the added dryness of the workings. These are matters which will, no doubt, be admitted theoretically, but are often overlooked in practice, and we mention them to remind readers that there is nothing surprising in the fact of such explosion overtaking new and up-to-date collieries; and that, in fact, the more intensive becomes the system of colliery working, the greater the liability to serious disasters."

A combination of intensive working, free ventilation, and freshly mined coal dust, which is more explosive than that which is weathered, is so common, that quotations such as given at the outset are explicable.

Vacuum Coal-Dust Removers.—The vacuum carpet sweeper is to be introduced into the coal mines of Colorado to remove the dust. It will now be necessary to have a new official, preferably of the fairer sex, to supervise the work. It is unfortunate that this idea was not evolved before the commission to revise the mining laws of the state had made its most excellent preliminary report, as some recommendations covering the duties of the new official should be made to the legislature. If a suggestion is permissible the inspector should not be allowed to wear hobble skirts. However, the plan has been tried on a small scale with ordinary room sweepers and satisfactorily removes the dust, including pieces of coal up to ¼ inch in size, from the rough floor of a boiler room. The idea is revolutionary, of course, but there seems no reason why it should not work if proper modifications are made to adapt the machine to underground conditions.

Bellevue, B. C., Disaster.—Mr. James Ashworth, when giving evidence before the coroner's jury in the case of the Bellevue, Alberta, B. C., mine disaster, said: "The Mt. Killner mine, in New South Wales, was standing idle. Three men were in the mine, one at pump and two in the shaft, when a great rumbling was heard, after which the mine blew up, and was found to be completely filled with afterdamp. The jury that sat upon the case rendered a verdict of spontaneous combustion, but when the mine was explored it was found that the roof had fallen and there were no traces of fire in the mine." Mr. Ashworth stated that he was of the opinion that the cause of the Bellevue accident was due to a fall of rock. The inference is that the heat brought about by the concussion due to the fall of rock might be sufficient to cause a dust explosion. Referring to an explosion in a mine in South Wales he stated that on the upper side of the gangway where the rooms were wide the men were all able to walk out after an explosion, while those who were working on the lower side in narrow rooms were all killed.

Coal at Elkton, Mich.—Well drillers on a farm 3 miles northeast of Elkton, Mich., drilled through 6 feet of bituminous coal. Samples were reported to be entirely free from sulphur, which is the case in the Sebewaing mines, 20 miles west of there. Just how far the deposit extends is not yet known. Oil prospectors have secured options on thousands of acres of land north and east of Elkton, and while they are secretive regarding the matter, it is expected that active drilling and developing will begin in the early spring.

More Coal Found in Canada.—Coal, said by experts to be of better quality than any yet found in the West, has been struck by well drillers at Salvador, a station 9 miles west of Luseland, Sask., in the territory being colonized by a land company of St. Paul. It was found at a depth of 260 feet, and the bed was 5 feet wide.

Birch Grove Colliery.—Development work at the Birch Grove Mine, Sydney, B. C., is proceeding very quietly. Both slopes are now down a distance of 400 feet. They have passed the dip in the seam and are now ascending to the southern outcrop. When the slopes were started, they were driven at an average of 43 degrees for about 200 feet, when the trough of the basin was reached. At this point they became level and continued so for another 100 feet, when they again took a rise of about 5 per cent.toward the southern outcrop. A carload of coal has been sent from the mine to the washing plant and chemical laboratory at the steel works, where it is being tested to determine its value for coking purposes; its value in that respect is believed at present to be quite high.

Big Muddy Coal, Wyoming.—It is reported that a new bed of coal has been discovered at the Big Muddy Mine, Basin, Wyo.; also that good coal has been found between Bodie and Wellington, Nev.

Coal Mine Owner Badly Burned.—Thomas Wanless, owner of the Wanless coal mines, and two miners were burned by an explosion in the mine at Providence, Ky. Two miners were shooting preparatory to leaving the mines, and the owner was down inspecting the work. Four or five shots went off at the same time, and it is supposed that a blown-out shot must have occurred, igniting the powder supply. The mine immediately became a mass of flames, which died out after 20 minutes. The men, when found, were unable to speak. Wanless is in a critical condition. The other two men will recover. The mine is a mass of debris. It is a slope colliery, and 20 men were employed.

He Had Cold Feet.—On Sunday night as the curfew bell was ringing out the old and ringing in the young, the staff of the Fernie Hospital was thrown into consternation by the loss of a patient, one Xenophon Pouplos, a Greek who had acquired some frosted pains at Elko and had been brought there for treatment.

Like his lamented and illustrious namesake of Elea, this sport believed that phenomena was merely an illusion and that cold was a chimera. So he proceeded to demonstrate his belief by wandering out of the hospital, clad in a nightie, one sock, and one slipper. It was about 10 degrees below zero at the time.

But if consternation reigned in the hospital, she reigned there as over a tributary state, her real throne and imperial dominion being over at the coke ovens, where Jimmy McLean's gang of nihilists were down on their prayer handles, worshipping a luminous being in flowing robes that was toasting its shins over a glowing vent above one of the ovens.

One of the Russians called for the priest, but an M. F. & M. man thought he said "police" and phoned for the "force." Xeno was persuaded to go into camp temporarily in the car shops and was taken later in a pung to the hospital. Strange to relate, he was little the worse of his adventure.

A Nova Scotia Mine Tragedy.—An explosion January 3 set fire to Sydney No. 3 mine of the Nova Scotia Steel and Coal Co., at Florence, N. S., and eight men were entombed. The explosion occurred about 4 o'clock in the morning, when Under Deputies Ferguson and Purchass went into the mine accompanied by six workmen, making preparations for starting the mine. For 8 hours Manager John Johnstone, assisted by Inspector Nicholson and others fought the fire, and at last found the charred bodies of six shift men who were clumped together and so burned that it was impossible to recognize one from the other. Then failing to reach the bodies of Deputies Ferguson and Purchass, owing to the deadly gases, they were compelled to retreat to a place of safety.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

Western Editorial Office: 72S Equitable Building, Denver, Colo.
Chicago Office: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to Mines and Minerals, Scranton, Pa.

CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER.......... Manager
EUGENE B. WILSON, Scranton, Pa.......... Editor
GEORGE F. DUCK, E. M., Denver, Colo.......... Western Editor
P. G. MOORE.......... Circulation Manager
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., Ltd., Dryden House, 43 Gerrard St., Soho, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 Pitt Street, SIDNEY,
SOUTH AFRICAN AGENT: E. HERBERT BYSSHE, CAPE TOWN, Mansion House; JOHANNESBURG, Trust Building; DURBAN, Natal Bank Chambers; BLOEMFONTEIN, Florence Buildings.

SUBSCRIPTION RATES

Single Copies.......... 25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada.......... 2.50
To all other countries.......... 3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always gives your FORMER as well as your present address.

TABLE OF CONTENTS

El Cobre Mines, Cuba, by E. G. Tuttle.......... 449
Canadian Iron Ore Industry.......... 455
An Accidental Taylor Air Compressor.......... 458
Transvaal Drill Competition, 1909.......... 459
Magnetic Separation of Wolframite.......... 462
Personals.......... 463
Obituary.......... 463
The Philippine Gold Mines, by Monroe Woolley.......... 464
Geology of La Veta Coal Field, by Arthur Lakes.......... 466
Asbestos, by H. R. Edgecomb.......... 469
The Calama Copper District, Chile, by F. A. Sundt.......... 473
Lime Reaction in Cyaniding, by Theo. P. Holt.......... 475
Metallurgical Sleuthing, by E. B. Wilson.......... 476
Small Air Compressors.......... 477
Determination of Alkalinity.......... 478
Titrating for Protective Alkalinity, by H. L. Sulman and Frederick Reade.......... 479
Boquillas Zinc Deposits, by Carlos Moser.......... 479
Wisconsin Zinc Ores.......... 480
Coal Notes.......... 481
Editorial—
Proposed Colorado Coal Mine Law—Colliery Warnings—A New Book Catalog.......... 482
Book Review.......... 483
Correspondence—
Mining Around Gas Wells—Steam in Air-Current—James Blick.... 486
The Proposed Colorado Mine Law.......... 487
First-Aid Work at Coal Mines, by John H. Ketner.......... 490
First-Aid Graduates, Brier Hill, Pa.......... 491
Combustion of Coal in Boilers.......... 492
Electric Shock in Mines, by Sidney F. Walker.......... 493
Methods of Removing Coal Pillars, by F. W. Cunningham.......... 495
Gravity Determination of Coal, by Abraham Blakeley and Edwin M. Chance.......... 499
Liability for Industrial Accidents, by Sion B. Smith.......... 501
Mining Industrial Expositions.......... 503
Bavarian Graphite Industry, by George Nicholas Ifft.......... 504
Data of Petrodynamics, by R. Dawson Norris Hall.......... 505
Cement-Lined Pumps and Column Pipe.......... 506
Electric Circuit Problems.......... 507
Trade Notices.......... 508
Examination Questions—
Pennsylvania Bituminous Mine Foremen's and Fire Bosses, April, 1910 509
Catalogs Received.......... 512
New Inventions.......... 512
Squibs—
Phosphate in Oceania, 454—Action of *CO* and *H*, or CO_2 and *H*, on Iron Oxides, 454—Grahamite, a Solid Native Bitumen, 458—The Ingersoll-Rand-Cameron Combination, 462—Yukon Territory of Canada, 463—Filling Stopes, 465—The Maikop Oil Field, 468—Rule for Figuring Capacity of Rolls, 468—Earth Colors of Sardinia, 472—Joplin Zinc Ore Notes, 472—Solubility of Gold in Potassium Ferrocyanide, 477—Anthracite and Bituminous Output, 485—Coal in Uruguay, 485—Volume of Air Delivered by a Fan, 486—Export of Coal, 494—Parrafine Oil in Ohio Mines, 499—To Prevent Dusting of Concrete Floors, 500—Discovery of Coal in Brazil, 500—United Mine Workers' Expense Account.. 504

PROPOSED COLORADO COAL MINE LAW

ON ANOTHER page we have criticized severely the vicious and ridiculous proposed Colorado coal mine law.

The first rational mine law enacted in the United States was framed by Mr. T. J. Foster, then editor of this journal, which was at that time known as The Mining Herald. Incomplete as that first law was, it was a step in the right direction and its enactment in the state of Pennsylvania was largely due to the then editor of this journal. From that time on, every coal mine law enacted in the United States has to a greater or less extent been influenced in its character by the criticisms and suggestions made by the practical men connected with the editorial management of Mines and Minerals. In no instance has it opposed any law which would tend to increase the safety and conserve the health of coal miners, but, on the contrary, has strongly advocated the passage of such laws. Naturally, from time to time in Pennsylvania and in other coal-mining states, proposed acts or amendments have been introduced in the legislatures that in some instances were merely attempts to make trouble, or were the mistaken ideas of men who were honest in their desire to remedy certain conditions, but whose proposed remedies would have made those conditions worse rather than better. Such acts Mines and Minerals has always opposed, and in most instances has succeeded in defeating them.

The proposed law now before the legislature of Colorado is one that we do not believe can be tinkered into rational shape, and it should be thrown out bodily; otherwise the state of Colorado will be worse off than if it had no law. We do not believe that any half dozen intelligent, conscientious miners in that state would, on its merits, favor the passage of such a bill as the one in question, and we know that no mine manager or mine owner will commend it.

If the proposed law is really the work of the commission appointed by Governor Shafroth, we are greatly disappointed in our previous estimates of the practical mining knowledge possessed by its members.

COLLIERY WARNINGS

APROPOS of the increasing interest displayed in explosions of coal dust and the causes leading up to them and the suggestion sometimes heard that the government be requested to send out warnings of changes in temperature and relative humidity, that the mines might be more thoroughly watered before an approaching dry spell, an editorial in a recent issue of *The Mining Journal*, of London, England, is of interest. Commenting upon the Hulton colliery disaster it says: "The Royal Commission on Coal Mines points out that there is no longer any controversy that coal dust is the agency to which alone great explosions are due, and that the only question at issue now is what means are to be taken to prevent them." Putting this more broadly

the Commission might have said: "Regardless of the initial cause, whether a blown-out shot, or mine fire or explosion of firedamp, there has been no accident within the past 30 years, and presumably not in history, involving an explosion and the loss of as many as ten lives which should not be attributed to the agency of coal dust and to that agency alone."

In England, of the initial causes leading up to an explosion of coal dust perhaps the most usual is the firing of a pocket or feeder of marsh gas, with blown-out shots second, and mine fires or "crossed" electric wires third, while in the United States the order of the first two causes would probably be reversed. It has long been known that changes in atmospheric pressure, as measured by the rise and fall of the barometer, have some effect upon the entrance of gas into the workings of a mine, but it has seemed to us that the importance of these barometric changes have been much exaggerated. The barometer has been the "bogey man" of the mining engineer for many years, and it has always been believed, in the United States at least, that the so-called "colliery warnings" sent out in England were official and had the sanction of the government. But it seems that we were mistaken and we are glad to rectify the error.

Continuing, the same editorial says: "There is one other small matter which invites a word of observation at the moment. *The Times* (London), of Friday's date says: 'A disaster has followed immediately upon a colliery warning which appeared on Monday in newspapers circulating in the various mining districts; it is now 30 years since the warnings were started, and the interval has witnessed an enormous diminution both in the frequency of explosions and in the loss of life caused by them.' For ourselves, we have never been able to understand the meaning of the so-called 'colliery warning,' and we would venture to direct our contemporary's attention to the following passage in the report of the Royal Commission on Coal Mines: 'Our attention has been drawn to the issue in the newspapers from time to time of the so-called colliery warnings. These warnings are based upon observations of barometric changes, and are supposed to indicate the need for special precautions being taken against firedamp in collieries. In some cases these colliery warnings have been quoted in connection with the occurrence of fatal explosions. We were informed that these warnings are not issued with the authority of the Meteorological Office, and that they have no official sanction. *They are misleading, and, so far as we can see, their publication serves no useful purpose.*' (The italics are our own.) Perhaps, if our contemporary is in possession of information denied to the Mining Commission, we shall at length be able to discover the reason for a faith in these mysterious warnings which to us, at any rate, have always seemed in the nature of a shibboleth."

The Royal Commission on Coal Mines states the case with true British conservatism when it says of these warnings: "Their publication serves no useful purpose." In Great Britain, where the miners are almost entirely natives, they may do no particular harm, but in the United States where perhaps as many as 90 per cent. of the mine workers are of non-English speaking birth, such warnings might be a source of actual danger. Few men, and particularly those of the class composing our foreign mining population, are cool in the face of impending danger and are then very apt, through sheer nervousness, to be guilty of acts imperiling their safety, which would be improbable, if not impossible, under ordinary mental conditions. An illustration of the effect of the ill-advised statements of public officials has recently occurred in one of our western states. There, the officer in question while in no way connected with the mining department and obviously ignorant of the first principles of the profession, appeared in print as saying that three mines which he specifically named, were due "to blow up." When this item of alleged news was distributed around the mines, it caused an exodus of men from the workings, and they remained out until the State Mine Inspector reported that the mines in question were absolutely safe. If a minor state official with no particular standing, and in no way connected with a mining department, can cause so much harm by erroneous statements, how much more would be induced by "warnings" issued officially by the national government and coming with authority? If these warnings were sent privately to the officials of the mine who are responsible for the safety of the workings they would, at least, do no harm, but falling into indiscriminate and too often ignorant hands, they would more frequently be a source of serious menace.

A NEW BOOK CATALOG

FOR the convenience of our readers who may want the most up to date and reliable books on Mining, Mineralogy, Metallurgy, Prospecting, and Assaying; and on Mechanics, Electricity, Chemistry, etc. as related to mining and metallurgy, we have prepared for the Technical Supply Co., ,of Scranton, Pa., a comprehensive and convenient catalog of *"books worth while,"* which will be sent free on request to any reader on application to either Mines and Minerals or the Technical Supply Co.

The prices quoted in the catalog are strictly publisher's prices and any books ordered from the Technical Supply Co., will be sent postpaid. Orders received by Mines and Minerals will be promptly filled by the Technical Supply Co.

BOOK REVIEW

Electric Circuit Problems is the title of a book by Ellis H. Crapper, B. Eng., M. I. E. E., and head of the Electrical Engineering Department at the University of Sheffield, England. The book deals with the applications of electricity to mining

operations, which are now so extensive and varied as to form an important part of their equipment.

When an electrical system has been properly installed at a mining operation and qualified electricians are placed in charge the dangers attending the use of electricity in mines may be reduced to a minimum, provided that adequate and systematic inspection is forced.

The contents are: Units of Measurements; Insulation; Circuit Testing; Cables; Alternating and Polyphase Currents; and Polyphase Current Circuits, which with the index cover 159 pages. There are 57 illustrations, and the text is replete with formulas and examples. The book is printed by the Colliery Guardian Co., London, England. Price 3s. 6d.

The Mineralogy of Arizona, by F. R. Guild, Professor of Chemistry and Mineralogy, University of Arizona. According to the author, in certain respects many of the minerals in Arizona are sufficiently unusual in their mode of occurrence and in their composition to warrant special attention. The order of treatment, according to minerals in this book, is the same as under Dana's classification. The elements found as minerals are considered first, then the sulphides and others as outlined. There are seven chapters devoted entirely to minerals. The publishers are the Chemical Publishing Co., Easton, Pa. The price of the book is $1.

Heaton's Annual for 1911. This is the Seventh Annual Commercial Handbook of Canada and Boards of Trade Register. It is published by Heaton's Agency, Toronto, Canada, and contains much valuable information concerning Canada, its banks, resources, mining and general information. The price is $1.10, postage paid.

Mechanical World's Pocket Diary for the Year 1911. This is the twenty-fourth year of this publication. It contains a great deal of useful information for mechanics and mechanical engineers. It is printed by the *Mechanical World*, London, England. The price is 6d. net.

The Mechanical World's Electrical Pocketbook for 1911 is printed by the *Mechanical World*, 20 Bedford St., London, W. C. Price 6d. net.

A Manual for Assayers and Chemists, by W. H. Seamon, B. S. A., chemist and metallurgist. This book is the result of the experience of 14 years as a teacher of chemistry and assaying, and 15 years of practical work in the mining districts of the southwest. On reading over the work one is particularly impressed with the system followed, in that when a subject is taken up the various methods by which it may be assayed are given. For instance, mercury. First the ores are mentioned, then the method of the detection of mercury; the method of determining mercury, the method of decomposition, the titration of mercury, the gravimetric method, and, finally, the valuation. There are four parts to the book. The first part treats of metals; the second of non-metallic elements, chlorine, phosphorus, selenium and tellurium, silica, and sulphur; the third part is miscellaneous, and the fourth part contains tables which are much used by assayers and metallurgists. The book is 12mo, contains 255 pages, and the price is $2.50. It is published by John Wiley & Sons, New York, N. Y.

Elementary Crystallography, by W. S. Bayley, Ph. D., Associate Professor of Mineralogy and Economic Geology, University of Illinois, Urbana. The book does not pretend to be a treatise on crystallography. It is a guide for those attempting to gain an insight into the fundamental principles underlying the science. It is the direct result of the need felt by the author in his own classes for a simple discussion of crystals and a series of simple statements of crystallographic conceptions. It is divided into three parts, Geometrical Crystallography, Physical Crystallography, and Chemical Crystallography. Pentagonal hemihedrism of the positive and negative dyakisdodecahedral class are both illustrated and given the positive and negative symbols of the diploids. Right orthorhombic bishpenoids and left orthorhombic bisphenoids are illustrated under the hemihedral division. From any pyramid it will be remembered two of these forms are produced. They are enantiomorphous, and are therefore known as right and left forms. The book contains 241 pages, and is published by the McGraw-Hill Book Co, New York.

Pocketbook of Mechanical Engineering, by Charles M. Sames, B. Sc. This is the fourth edition of this pocketbook, which has been revised and enlarged. It has a flexible cover 4 in.$\times$6⅝ in. and contains 220 pages. The price is $2, and is for sale by C. M. Sames, 542 Bramhall Ave., Jersey City, N. J. The contents are mathematics, chemical data, materials, strength of materials and machine parts, energy and transmission of power, heat, and steam engines, hydraulics and hydraulic machinery, and electrotechnics. The *Engineering News, Scientific American, Engineering Record and Machinery*, make favorable comments on this work of Mr. Sames.

Corrosion and Preservation of Iron and Steel, by Allerton S. Cushman and Henry A. Gardner. The authors are well known as investigators of physical and chemical phenomena, Mr. Cushman having been Assistant Director of Chemists in charge of chemical and physical investigations in the United States Department of Agriculture, and Mr. Gardner, Director of the Scientific Section of the Paint Manufacturers' Association of the United States. The authors have made the effort to include or mention the results of all recent investigations and original researches touching on the corrosion of iron and steel which have appeared up to the time of this book going to press. The book is written mainly to elucidate the electrolytic theory of corrosion and includes an appendix presenting a discussion before the American Institute of Mining Engineers on the "Corrosion of Water Jackets of Copper Blast Furnaces." No two men are better qualified to write a book of this description. It is dedicated to the memory of Charles B. Dudley, who did so much in the line of investigations for the prevention of corrosion and wear of steel. The price of the book is $4; it contains 363 pages and index, and it is published by the McGraw-Hill Book Co., New York.

The School of Mines and Metallurgy, University of Missouri, Rolla, Mo. The bulletin of this school, issued in December, 1910, will be found exceedingly interesting to mineralogists. It contains an article on "Some Relations Between the Composition of a Mineral and Its Physical Properties."

The Cement Age and Concrete Engineering, the latter formerly of Cleveland, Ohio, have been consolidated, and will hereafter be published under the title "*Cement Age*." The magazine will be larger than the present size of *Cement Age* and will have a type page 6 in.$\times$9 in. The headquarters of the publication will be 30 Church St., New York, N. Y. Allen Brett, Editor of *Concrete Engineering* for the past two years, will take the position as associate editor of the new publication, and Arthur E. Warner, formerly business manager of *Concrete Engineering*, will become western manager. There will be no change in the present staff of *Cement Age*, Mr. Lesley continuing as editor, Frederic F. Lincoln as president of the Cement Age Co., in charge of the New York office at 30 Church St., and of the eastern advertising field, and Edward A. Trego, as associate editor.

Metal Statistics for 1911. This makes the fourth edition of Metal Statistics, published by the American Metal Market and Daily Iron and Steel Report, 81 Fulton St., New York, N. Y. This book is made up of statistics. Under the headings "Iron and Steel" there are statistics on ore, pig iron, fuel, finished products, scrap iron, and miscellaneous information relative to iron, coke, etc. Under the heading "Metals," statistics relative to copper, tin, lead, spelter, antimony, aluminum and silver, are given. This includes production at home and abroad and the total production for a number of years previous to 1911; also the consumption of metals in the United States and the exports from this country as well as imports. Under the heading "Miscellaneous" there is a large fund of

information which will be of interest to almost any one who desires statistical information in regard to metals. Messrs. Luty and French, editors of Metal Statistics, believe that the 1911 edition is in many respects the superior to any of the previous issues. The price of this book is 50 cents.

TWENTY-FIVE YEARS OF ENGINE BUILDING is the title of a most interesting and handsome publication issued by the C. & G. Cooper Co., of Mount Vernon, Ohio. It is a well-written and interesting historical sketch of the origin of the Cooper plant in 1833, followed by a description of its growth to its present size and reputation. It is illustrated by remarkably fine engravings, and the typography is of the highest standard. In fact, it was designed and laid out by an artist in typography, and no expense was spared in the mechanical work done in its production.

CLAYS AND CLAY INDUSTRY OF WASHINGTON, a report published by Solon Shedd, Ph. D., Professor of Geology at the Washington State College, Pullman, Wash. The report is neatly bound in cloth, illustrated on good paper, contains 325 pages of descriptive matter, 42 plates, and in all, is a very creditable report. Part 1 covers clay and its properties, dealing with the mode of occurrence, classification, general description, chemical and physical properties, and suggestions in regard to prospecting for clay. Part 2 covers the technology of the clay industry, and deals with the mining and preparation of clay, the preparation of clay wares, drying and burning of clay wares. Part 3 treats of clays and the clay industry of Washington, covering in order, Olympic Peninsula, Coast Mountains, Puget Sound Basin, Cascade Mountains, Okanogan Highlands, Columbia Plain, and the Blue Mountains. In treating the clays of Washington, the author classifies the various clays into (1) residual and (2) transported clays, and then gives data on the tensile strength, shrinkage, and plasticity of clays. This part is followed by a description of the various clay plants of the state. Ten counties of Eastern Washington and an equal number in Western Washington are represented in the list, as being valuable clay producers.

UNITED STATES GEOLOGICAL SURVEY PUBLICATIONS, Washington, D. C., Bulletin 436, The Fauna of the Phosphate Beds of the Park City Formation in Idaho, Wyoming, and Utah, by George H. Girty; Bulletin 442, Mineral Resources of the Southern Part of Kenai Peninsula, by U. S. Grant and D. F. Higgins; Bulletin 470A, Phosphates in Montana, by Hoyt S. Gale; The Production of Quicksilver in 1909, by H. D. McCaskey; The Production of Gold, Silver, Copper, Lead, and Zinc in the Eastern States in 1909, by H. D. McCaskey; The Production of Mica in 1909, by Douglas B. Sterrett; Lithium in 1909, by Frank L. Hess; The Production of Petroleum in 1909, by David T. Day; The Production of Antimony, Arsenic, Bismuth, and Selenium in 1909, by Frank L. Hess; The Production of Tungsten, Nickel, Cobalt, Vanadium, Titanium, Molybdenum, Uranium, Tantalum, and Tin in 1909, by Frank L. Hess.

BUREAU OF MINES, Washington, D. C., Bulletin No. 5, Washing and Coking Tests of Coal at the Fuel Testing Plant, Denver, Colo., by A. W. Belden, G. R. Delamater, J. W. Groves, and K. M. Way; Bulletin 2, North Dakota Lignite as a Fuel for Power-Plant Boilers, by D. T. Randal and Henry Kriesinger; Bulletin 3, The Coke Industry of the United States as Related to the Foundry, by Richard Moldenke.

DEPARTMENT OF COMMERCE AND LABOR, Washington, D. C., Bulletin of the Bureau of Labor, No. 90. This bulletin is of special interest at the present time. It contains fatal accidents in coal mining, recent action concerning accident compensation, foreign workmen's compensation acts and cost of industrial insurance. It may be had by addressing the Department of Commerce and Labor, Washington, D. C.

MICHIGAN GEOLOGICAL SURVEY, Publication No. 2, Geological Series No. 1, The Monroe Formation of Southern Michigan and Adjoining Regions, by A. W. Grabau and W. H. Sherzer. This is published as a part of the annual report of the Board of Geological and Biological Survey for 1909. Geologists in charge, Dr. L. L. Hubbard, Houghton, Mich., and Prof. W. H. Hobbs, Ann Arbor, Mich.

DEPARTMENT OF GEOLOGY AND NATURAL RESOURCES OF INDIANA, Thirty-Fourth Annual Report for 1909, W. S. Blatchley, State Geologist, Indianapolis, Ind.

NINETEENTH ANNUAL REPORT OF THE MINING DEPARTMENT OF TENNESSEE, Mineral Resources of Tennessee, by R. A. Shiflett, Chief Mine Inspector, Nashville, Tenn. This book can be had by addressing the Mining Department of Tennessee.

NEW ZEALAND GEOLOGICAL SURVEY, Bulletin No. 10 (New Series), The Geology of the Thames Subdivision, Hauraki Auckland, by Colin Fraser. Address J. M. Bell, Director of Department of Mines, Wellington, N. Z.

THE FARMERS' HANDBOOK OF EXPLOSIVES, by the E. I. du Pont de Nemours Powder Co., Wilmington, Del.

HOW TO OPERATE A PNEUMELECTRIC COAL PUNCHER, Pneumelectric Machine Co., Syracuse, N. Y. This information is printed in English, Slovak, Italian, Polish, and Magyar.

UNIVERSITY OF ILLINOIS BULLETIN No. 43, Freight Train Resistance, and Its Relation to Car Weight, by Edward C. Schmidt, Urbana, Ill.

CANADA DEPARTMENT OF MINES, Mines Branch, Ottawa, Canada, The Production of Cement, Lime, Clay Products, Stone, and Other Structural Materials in Canada for 1909, by John McLeish, B. A.

REPORT OF THE COMMITTEE OF THE TRANSVAAL STOPE DRILL COMPETITION, price 7s. 6d. Printed by the Transvaal Government, Transvaal, S. Africa.

THE WISCONSIN ENGINEER, which is a monthly publication of the students of the College of Engineering, University of Wisconsin, has an exceedingly fine issue for January. The contents embrace "The Strength of Oxyacetylene Welds," a thesis by Herbert L. Whitmore, B. S. '02; an article entitled "A Travel on Hudson Bay," by C. K. Leith, Professor of Geology, who is well known through his geological writings; also an article on "The Utility of the Metallographical Microscope in Engineering," by James Ashton, and still another under the caption "Telephone Service in Chicago" by Alfred U. Hoefer. There is an editorial on "The Honesty of Mining Engineers" which is well worth reading. At some future time we hope to comment along this same line.

ANTHRACITE AND BITUMINOUS OUTPUT

The first records of bituminous coal production in Pennsylvania are for the year 1840, when 464,826 short tons were mined. The total output of bituminous coal from 1840 to the close of 1909 has amounted to 2,101,215,571 short tons, from which it appears that the total production of anthracite and of bituminous coal in Pennsylvania has been nearly equally divided between the two. At the close of 1908 the total production of anthracite from the earliest time to the close of that year had exceeded the total bituminous production by about 51,000,000 tons. As, however, the production of bituminous coal in 1909 exceeded that of anthracite by more than 56,000,000 short tons, the total production of bituminous coal now exceeds that of anthracite. The Geological Survey's report on coal production in 1909 can be obtained without charge by applying to the Director of the Survey at Washington about February 20.

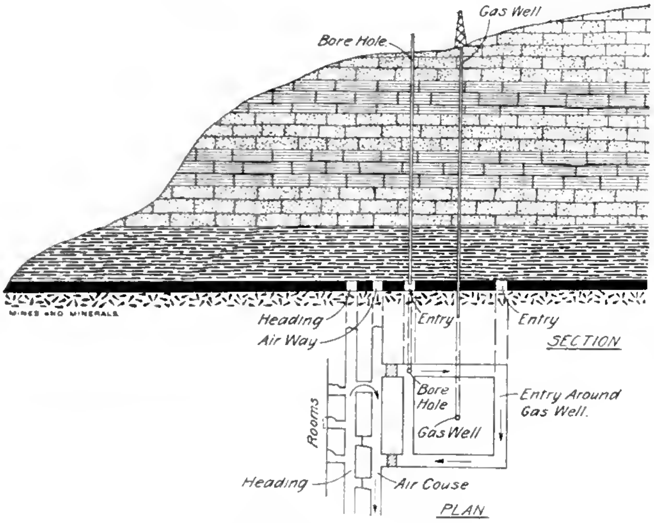
Engineer Antonio Llambias de Olivar, the representative of the boring company subsidized by the government of Uruguay, informs the Minister of Industries, Labor and Public Instruction, that in the middle of August, in the fourth boring which the company is making in the Department of Cerro Largo, a bed of coal was found at a depth of 124 meters, more than a meter thick and of quality superior to that met in the boring of November, 1908, at a depth of 140 meters.

CORRESPONDENCE

Mining Around Gas Wells

Editor Mines and Minerals:

Sir:—I am enclosing you sketch of a plan of working around a gas well which I think would be a help to some of the coal companies. The entry is to be driven around the well, and stoppings put in as shown, with a bore hole from the surface to the entry for gas to go through to the outside, or the return air could be circulated around it and remove all gas. The



Plan of Workings Around Gas Wells

arrows indicate the way the air could be circulated around the wells. I do not think the plan will stop all trouble from the wells, but I do think it will be some benefit from gas coming through the coal in any way. E. P. McOlvin

Meadowbrook, W. Va.

Editor's Note.—The subject of abandoned well holes is one of importance to the coal-mining industries in Ohio, Indiana, Illinois, Pennsylvania, and West Virginia. Mr. George Harrison has forced legislative enactments in Ohio, and other states will undoubtedly, yet the fact remains that the abandoned holes on hand must be dealt with, irrespective of the future. Mines and Minerals invites correspondence in order to bring out ideas which will cope with the undeniable danger and nuisance from these holes.

Steam in the Air-Current

Editor Mines and Minerals:

Sir:—We understand that a number of coal-mining companies in Western Pennsylvania have adopted a plan of turning exhaust steam into the fan house following a sudden drop of temperature. This is said to keep the air-currents moist and prevents the mines from becoming filled with dry dust. It is presumed that the plan adopted minimizes the danger of dust explosions.

A few years ago Mr. Galey patented a process for extracting moisture from air. This he accomplished by first passing the air through a cooling apparatus and afterwards to the hot-blast stoves where it is preheated. I have always understood that low-temperature air might contain as much moisture per cubic foot as a higher temperature air, but as soon as the cool air from outside came in contact with the warmer air inside it expanded and therefore did not contain so much moisture per cubic foot, and consequently absorbed moisture from the mines, in addition to what it would absorb before it became saturated. If the exhaust steam heats the air going into the mines and its moisture is absorbed by that air, would not the air deposit its moisture after it reached the cooler inside air of the mines, and then would not conditions be the same as if the air had not been steam heated?

Recently a patent was taken out in which a number of steam coils were used to prevent the incoming air from losing its moisture. Possibly some of your readers might throw considerable light on this subject, which is of so much importance to bituminous coal miners. X. Y. Z.

James Blick

Written for Mines and Minerals

Mr. James Blick, a Pennsylvania bituminous mine inspector for 20 years, died at Crafton, Pa., on January 4, 1911. Being so well and favorably known to the readers of Mines and Minerals, a few incidents in the life and work of this man may be of interest. Mr. Blick, 65 years ago, was born at Berkeley, Gloucestershire, England. He came to Neshannock, Mercer County, Pa., about the year 1880, and entered the coal mines there as a miner, the occupation which he had followed in England. He became a naturalized citizen at the expiration of 5 years, and was commissioned a bituminous mine inspector immediately thereafter. I became acquainted with Mr. Blick in 1880 at a mass meeting of miners held at Neshannock, and after hearing him discuss the issues then being considered, concluded that he was a miner of more than ordinary ability. Our acquaintance ripened into a lasting friendship. Recognizing Mr. Blick's ability I urged him to prepare for the mine inspectorship, and he was appointed from Mercer County to the Seventh, or Pittsburg District, which he held until obliged to give up the work owing to failing health. Mr. Blick was one of the most capable mine inspectors the state of Pennsylvania ever had in its service. No single individual did more for the bituminous miner in securing and attempting to secure efficient mining legislation.

Mr. Blick had grown into manhood before he had acquired the rudiments of education. He attended a technical night school 2 years before he came to this country. When he, however, began to realize the great handicap under which he was living and began to view life and its great responsibilities, he set his mental machinery to work to recover lost ground, and with that will power so manifest in that resolution and persistency so characteristic of the man, he mastered the fundamental principles of the sciences as they relate to mining.

Mr. Blick was thoroughly familiar with the mining literature of the present and past, and this, combined with his large and varied experience and practical knowledge made him a mining authority whose advice was much sought and accepted. He had written much on mining and humanitarian topics. His essays were written in a lucid style, never obscuring his subject by the embellishment of pretentious phrases. His aim was to furnish ideas, state facts, and establish truth in a clear and simple manner. His writings conveyed to the reader a clear idea of the man. He was a keen observer and aimed at great accuracy in all of his statements. Blick, a splendid type of the Anglo-Saxon race, stood for courage, self-control, honest and temperate habits. While he was unemotional, he was not lacking in tenderness of heart that made him feel for the unfortunates of life, and he made much personal sacrifice in their behalf. In the death of this sturdy miner, the state has lost a valuable citizen, the people of these mining communities have sustained an almost irreparable loss, and the mine inspectors with whom he was so long associated a wise counselor and a friend. Thomas K. Adams

The volume of air delivered by a mine fan varies directly as the number of revolutions. That is, a fan running 80 revolutions per minute delivers 100,000 cubic feet. How much air will this fan deliver at 160? Solving the proportion: $80 : 160 :: 100{,}000 : x$, we have $x = 200{,}000$ cubic feet.—*Jeffrey Catalog.*

THE PROPOSED COLORADO MINE LAW

By the Editorial Staff of Mines and Minerals

In our February issue we noticed in a general way, and mildly criticized, the preliminary report of the Commissioners appointed by the Governor of Colorado to suggest laws for the governance of coal mines in that state.

A Bill Introduced in the Legislature of Colorado, Which Is Unique in Its Impracticability

A bill has been introduced in the House of Representatives of Colorado, which we assume is the new law proposed by the Commission. If we are mistaken in this assumption, we most humbly apologize to the members of the Commission, and we sincerely hope, for the sake of the Commission, it is not guilty of the paternity of such a monstrosity.

Naturally, the proposed law is not all bad, but so much of it is vicious and ridiculous, and the bad so thoroughly mixed up with the good, that we hope for the sake of both the coal miners of Colorado and the mine owners of that state, that the legislature will throw out the entire proposed law and insist that a rational and practical mine law be prepared by a properly appointed Commission of men familiar enough with actual mining operations as to make it of real value.

We have not space to print the proposed law in full and will, therefore, pass over those portions which are not either vicious or impracticable without comment, and give our attention to those which are.

Section 2 of the proposed law provides for a Chief Inspector, seven Deputy Inspectors, and one Electrical Inspector. There may be some opposition to the increase of Deputy Inspectors from four to seven, but we are compelled to say that we do not agree with that opposition. If mine inspectors are to *inspect* mines, there must be a sufficient number of such officials to do it. Usually there are too few inspectors to enable more than perfunctory inspections to be made. But why an Electrical Inspector? If the men appointed to deputy inspectorships are not technically and practically capable of passing judgment on the safe use of electrical machinery and the safeguarding of electrical conductors, they are not fit for the positions.

The following extract from Section 2, providing for the appointment of a Chief Inspector, Deputy Inspector, or Electrical Inspector, will strike every reasonable man as either vicious or ridiculous.

"Whenever there exists a vacancy in the office of Chief or Deputy Inspectors or Electrical Inspector, the remaining Inspectors, excluding the Electrical Inspector, shall submit, from each of the four inspectoral districts hereinafter provided for, to the Governor, the names of two reputable coal miners of known ability and experience. From the names so submitted, the Governor shall appoint one miner from each of the four districts, and the Governor shall appoint a mining engineer of like ability and experience, and these five men, so appointed, shall constitute a Board of Examiners, whose duty it shall be to inquire into the character and qualifications of the candidates for the offices left vacant. The representatives from the inspectoral districts on successive Boards of Examiners, shall be appointed in numerical order so that all the seven districts shall share equally in the appointments of said Boards.* * *

"The Board of Examiners shall then proceed to the examination of those who may present themselves as candidates for said office or offices, and shall certify to the Governor the names of all such applicants as any four of the examiners shall find competent to fill the office under the provisions of this Act,* * *

"The Governor shall from the names certified appoint the person or persons possessing the best qualifications of the vacant office or offices of Inspector of Coal Mines."

It is hardly necessary to point out the viciousness of this section of the proposed law. It gives the mine owner no justice whatever, neither is it in the real interest of the miner It does not tend to the selection of Mine Inspectors. It tends to the appointment of political heelers, or irresponsible demagogues to positions in which they can do incalculable damage to capital invested in coal mines, and render the safety of the working miners less secure than at present.

It is ridiculous in that the Examining Boards practically appoint the Inspectors, and the Inspectors appoint the Examining Boards.

That State Mine Inspectors should be able and honorable men, fair in their dealings to both workingmen and employers, and working faithfully to carry out the provisions of all laws tending to safeguard both life and property is what the Commission evidently intended to provide for. But they will not insure the appointment of such men by the method proposed.

The ideal examining board for such purposes should consist of one qualified mining engineer of high character, one mine owner, mine manager or mine superintendent and two intelligent experienced coal miners. They should be appointed by the Governor, and in cases where both the workingmen and employers are organized, the suggestions of the mine owners as to the appointment of their representative on the board, and similar suggestions as to the appointment of the two miners, from the miners' organization, should be heeded. With such a board, the recommendation of any three of the members should suffice.

The portion of the proposed law which states that "It shall be unlawful for the Chief Inspector of Coal Mines, or any Deputy Inspector to be interested as owner, operator, engineer, stockholder, manager, director, or otherwise in any coal mine," is unjust to the inspectors. It is just and right that he shall not be interested in any way in any mine in Colorado, but why should he be prevented from investing his savings in the stock of a coal-mining company in another state, provided he does not neglect his duty as an inspector to look after his invested interests?

The proposed law also provides that the Governor *may on consultation with, and by the advice of the Chief Inspector* remove a Deputy from office. As the Chief Inspector and Deputies, under this act, will be practically appointed by the Board of Examiners, and the Inspectors will practically appoint the Boards of Examiners, there will naturally be a "community of interests" and it is hard to see what power the Governor will have in the premises. About the only thing that will tend to the removal of a Deputy will be the fact that he has made himself distasteful to his associates.

The Chief Inspector's tenure of office is 4 years, but there is no provision for his dismissal if cause for such action should arise. The Governor may be impeached, but not the Chief Mine Inspector.

Section 2 also provides that after a Deputy has inspected each mine "there shall be posted in a conspicuous place near the entrance of the mine his report of the conditions of the mine, and he shall send a copy to the Chief Inspector."

Why in the name of common sense should a copy of the report be posted at the mine entrance? That will not remedy unsafe conditions if such exist. Why not require the Deputy to notify the mine management of the nature of his findings, and if unsafe conditions exist require the management to remedy them at once? A report of his findings should naturally be sent the Chief Inspector, and also a report of his action with the mine management and results obtained thereby.

Section 4 of the proposed law, defining the Police Power of Inspectors is both incomplete and vicious. It empowers the Chief Inspector, any Deputy, any Company Inspector, and any Constable or Sheriff to arrest any person or persons who, *he has reason to believe*, is guilty of any violation of the Act. It is not necessary for any complaint to be made or warrant issued. A citizen of Colorado it matters not if he be miner, laborer, mine official, or mine owner is to be subject to arrest without warrant.

While we believe competent and qualified State Mine

Inspectors should be empowered to make arrests for violations of rational mine laws which come under their own observation, or regarding which proper sworn complaints are made; we do not believe that such power should be conferred on company officials and irresponsible constables.

Section 7, which provides for "Notice of Accidents in Mine," requires that whenever loss of life or "serious injury" shall occur, the owner of the mine must notify both the Deputy and the Chief Inspector within 24 hours. In the first place "serious injury" is not defined. What some men call serious injury others call trivial. In the second place, why should the mine owner notify *both* the Deputy and the Chief Inspector. Official notice to the Chief Inspector should go through and from his subordinate, the Deputy.

Section 8, entitled "Checking In and Out System," has considerable in it to commend it. By its use the number of men in the mine at any time would be definitely known, and by a little amplification it could be so improved that not only the number of men in the mine, but their identity would be known. The defect in this section is that it provides punishment for the mine owner who violates its provisions, but provides no penalty for the miner who may violate it by surreptitiously entering the mine through some short cut or opening other than that used regularly by the workmen.

Section 9, which provides for Company Inspectors, is one of the ridiculous features of the proposed law. It takes from the mine foreman and his assistants the responsibility which should rest on their shoulders. While company inspectors, such as some of the larger coal companies voluntarily employ, are commendable, they do not usurp the functions of the foremen. They visit and inspect the mines and report to the management, and the latter through the superintendents and foremen sees that abuses or unsafe conditions are corrected. One of the greatest elements of danger in American coal mines is, and always has been, lack of discipline, and the framers of this proposed law actually provide for still less discipline by providing officials to interfere with the one official who should have supreme authority.

Section 12, entitled "Explosives," is one that merits severe criticism. It prohibits the use of black powder under any circumstances and allows the use of "permissible" explosives only.

In the first place, black powder is better and safer for use in non-gaseous and non-dusty mines than any other explosive. It does not shatter the coal as much, and when used in shot holes near the roof is not so liable to break the top rock as is the quicker and more powerful "permissible" explosive. Experience has shown that the so-called "permissible explosives," while unquestionably the safest in gaseous and dusty mines, have a tendency to crack the roof, make it less safe, and to allow, under certain conditions, the ingress of large quantities of water. Again, the section provides that only enough powder for one day's work shall be taken into a mine at one time. Can any one tell how much this is? The skilled Welsh miner, for instance, will mine more coal in a day with half the consumption of powder than the unskilled Slav or Italian will in two days. The same skilled Welshman may, and frequently does, use twice as much powder some days as he does on others.

This section also requires the employment of shot firers in *all* mines regardless of whether they are gaseous or dusty or not, *except in mines where the longwall system of mining is used.* The fool killer has work to do when he meets the man who framed this section. We heartily commend the proper safeguarding of shot firing in gaseous or dusty mines, but we fail to see why special shot firers are necessary in non-gaseous and non-dusty mines. We also fail to see why shot firers are not necessary in longwall work if there is gas or dust present.

Section 13, entitled "Underground Workings," is very long and seems to indicate that the Commission was actuated by a desire to incorporate in the bill every conceivable idea, no matter whether it is applicable or injurious to Colorado's mines. While the items include practically all the good features of all laws of other states, they also contain all the weak, bad, and vicious features. This section should be rewritten entirely, a great deal omitted, and then arranged in some order that it may be intelligible. We note that "the maximum distance between props in room or pillar work shall be 7 feet." Here is an instance of the incorporation of a provision applicable only in special cases. There are hundreds of rooms in the mines of Colorado that do not require a single stick of timber, and there are others where the props should be almost "skin to skin." The use or non-use of timber and the distance apart of props is a distinct proposition for each and every mine in the state, and for each section of each mine as well. The provision that "every owner shall provide and maintain a water system for the purpose of conducting water to the face of each working place and throughout the entire open part of the mine in sufficient quantity to wet down the dust, and the said owner shall cause the dust to be wetted down on the *roof*, walls, timbers, and in the roadways," is superfluous and vicious, although it is provided that steam may be used and an appeal may be taken to the inspector for relief from the extreme provisions of this act. There are many mines in the state that require no watering, and water may be applied in other ways without being piped in. Likewise, the promiscuous watering of the roof is to be severely condemned. The increased fatalities due to falls from roof weakened through watering will far more than offset the possibly decreased death rate through lessened dust explosions. As in all cases the question as to whether the roof shall be watered or not must be decided for the particular mine.

"All drift and slope openings shall be lined with incombustible material for a distance of 150 feet from the mouth of the opening," should be changed to read "no outside frame building shall be constructed within 50 feet of, or be in any way connected with the timbering of any drift or slope." Mine fires do not start in the timber at the drift or slope mouth.

Section 14, relating to ventilation, is sound and should be adopted as it stands, although there are sundry items that might well be added.

Section 15 provides for the exclusive use of electric or safety lamps in Colorado mines. This is a mischievous proviso which will tend to increase accidents and lessen the earning power of contract miners. Electric mine lamps are on the market and are adapted to certain uses in the mines, but there has not yet been one produced that is entirely suitable for the working miner.

Under certain circumstances, safety lamps should be required, but when required, the type of lamp should be specified. This is not done in the proposed law. The primitive Davy lamp for instance is a "safety lamp" but it is far from being a safe working lamp and should not be allowed in any mine except when used by a competent fire boss for testing purposes only. There are a number of other types no more safe than the Davy, without its merit as a testing lamp.

Experience has shown that the best types of safety lamps yield so little light and are so constructed that a careful inspection of the roof cannot be made; consequently an increased number of accidents due to falls of roof is due to their use. Besides, no miner can accomplish as much work in the almost total darkness incident to the use of safety lamps as when he has the better light of an ordinary miner's lamp. As a matter of fact, safety lamps, like permissible powders, should only be required where absolutely necessary. Cases of absolute necessity are comparatively few where proper ventilation is provided. Even in very gaseous mines, safety lamps are not required in all portions, as the headings at least, when they are intakes, except possibly at the working face, can be kept free from gas.

Section 16, relating to abandoned workings, gob fires, safety devices, shafts, rescue apparatus and the like, contains too much under one heading. As in all sections, all the good and bad practice is included in it. It is impossible to remove *all*

fine coal or slack. The provision for double drawbars on cars where the grade is over 6 per cent. is entirely unnecessary. What is to be gained by the requirement of printing the "code" on a "board or metal plate"; why not on paper or cardboard and renewed frequently? We do not see the idea in providing that all shafts be lined with concrete or other incombustible material. If this means that no timber at all is to be used in shaft construction, what is to be done for guides and their supports? At "each mine the owner shall have trained for rescue work not less than three crews of four men each." This is a most peculiar provision. Suppose that the men refuse to be trained, where will the owner get the rescue crews? And who is going to do the training as the law has made no provision for this?

Section 17, providing for second openings, would be all right if properly constructed. If there are any mines in Colorado not provided with a second opening, the law should require such an opening, but a reasonable time in which to provide it should be allowed. This reasonable time is not provided for in the Act.

Again in Section 17 it is required in one place that every escapement shaft over 75 feet in depth shall be equipped with a "safe, adequate, and independent hoisting plant." Later on in the same Section, the proposed Act provides that "every escapement shaft over 75 feet deep and *not* equipped with a hoisting plant shall be provided with a safe and substantial stairway, etc."

This is just one instance of the carelessness with which the proposed law is drawn. There are many of them throughout the copy at hand. In this instance the proposed law requires a hoisting plant at every escapement shaft over 75 feet deep, and in the next paragraph it is implied that a hoisting plant is not necessary at escapement shafts over 75 feet deep.

Section 18 treats of mine maps and is very long. So much is required to be placed on a mine map, that it is doubtful whether anything could be clearly and distinctly shown on such a map. Again the Act requires that all maps shall be made on a scale "Not to exceed 100 feet to the inch"; this should read all maps shall be made on a scale of not more than 100 feet to the inch. This would make clear what the Commission evidently meant.

This Section also requires that copies of all maps and plans shall be kept in the office at the mine, that one copy shall be deposited in the office of the Chief Mine Inspector, in the Capitol at Denver, and that one copy shall be deposited with the Recorder of the county in which the coal mine is located. Why in the name of common sense should the Recorder of the county have a copy of the mine map and not the Deputy Inspector? There is no sense in the County Recorder having on file open to the public, including farmers, shyster lawyers, and busybodies generally, what is purely the business of the mine owner and his employes. Why a copy of the map should be filed with the Chief Inspector is another point we cannot understand; if the Deputy Inspector has his map that should be all that is necessary, and is all that is required in other states.

The proviso requiring maps mentions about everything possible to be shown on the map, including underground oil houses. This implies that the Commission framing the law did not have enough practical experience to know that the storage of oil of any character in a mine is radically wrong from the point of safety to the workingman.

It also provides that "whenever an owner is about to open a new mine, or materially extend the workings or rearrange the plan of operation or ventilation of a mine already opened, he must make a detailed surface map and a detailed plan of the proposed operation and submit it to the Chief Inspector of Coal Mines for his approval, before proceeding with such work." This is the veriest rot. No mining engineer or mine official ever laid out plans for the opening of a new coal mine or the extension of the workings of an old one, that he would be willing to stake his reputation on as being what would actually be done. Local conditions met with in a mine often materially change plans. The Commission evidently imagined that it is as easy for the mine manager or mining engineer to determine the exact plan of future workings of a coal mine as it is for them to lay out a flower garden in regular beds. Some of the provisions regarding maps, especially that one requiring maps of mines about to be closed or abandoned, are excellent.

Section 19, entitled "Fencing Piles of Slack Coal and Abandoned Pits or Shafts," was evidently framed as a sop to farmers and herdsmen. It is all right enough to require that shafts should be fenced or otherwise protected so as to prevent human beings from falling into them, but why a proviso to protect horses, cattle, and other live stock, roaming at large, should be incorporated in a mine law is beyond our comprehension.

Section 20, providing for a method of appointing an Electrical Inspector, would be commendable if an Electrical Inspector was at all necessary. It provides a method whereby an honest, capable man, free from political or other affiliation might be selected. But, as we stated before, such an official is unnecessary. In this Section there are incorporated 67 rules governing the use of electricity under ground. Whoever framed these rules knew infinitely more about the practical use of electricity in mines than the members of the Commission did about practical mining, and as a whole the rules are excellent.

Section 22, among other things, prohibits smoking in all mines. This is perfectly absurd and will work a most undeserved hardship on the majority of the miners. Why a miner working in a non-gaseous mine should be prohibited from the comfort of a pipe of tobacco after his noon-day lunch is more than we can conceive. Possibly the Commission called in Mrs. Carrie Nation to help on this Section. Smoking should not be permitted where safety lamps are necessary but there is no reason why it should not be allowed when there is absolutely no danger attending it.

Section 23 provides that "For any injury to person or property occasioned by any violations of this Act, or wilful failure to comply with its provisions by any owner of any coal mine, a right of action against the party at fault shall accrue to the party injured," but there is no provision for any right of action against the workingman who wilfully violates the provisions of the Act, and not only endangers his own life but the lives of his fellow workmen, and the safety of his employers' property.

Section 24, providing for "Scales, Inspection of Scales, Weighing of Coal, and Check Weighmen," is in line with similar legislation in other states at the instance of the miners' unions. This is a question to which two sides of an argument can be applied. We believe the matter should really be left as a matter of contract between the operator and the men. We do know of instances in some mining regions where advantage was taken of the miners by unscrupulous men engaged in the coal business. Of course, such contemptible actions were comparatively few and we never knew of a large corporation wilfully doing such things; but the writer does know as a fact and saw specially constructed screen bars, made for certain individual operators, which could be regulated surreptitiously by the use of an "S" wrench and the quantity of slack produced increased at the will of the operator, while, when the Inspector came around the openings between the bars could be speedily made of legal size. It was such occurrences as this that brought about the necessity of scales and check weighmen, and it is unfortunate that the action of a few should bring on the many the necessity of such provisions in the law.

Section 25, reading "Oils Permissible in Coal Mines," does not agree with the Section on Mine Maps, in that it limits the amount of lubricating oil which may be taken into the mine at one time as one barrel, while the Section referring to Mine Maps provides for the showing on the maps the location of inside oil houses. Does the Commission mean that only one barrel of

lubricating oil may be kept in the mine at one time and that it makes no difference if 50 barrels of lamp oil are there at the same instant, or what does it mean?

Section 26 provides for a "Coal Mine Inspection Fund" out of which all salaries, etc., to the Mine Inspectors are to be paid by imposing a tax of 1 cent per ton on the coal mined in the state. Colorado produced 12,000,000 tons of coal in 1910; 1 cent a ton tax levied on that number of tons would amount to $120,000.

Bandaged Patient in Underground Hospital

According to the salaries required to pay the proposed Inspectors, with reasonable traveling expenses and with a fund of $15,000 to cover incidentals, the Department would probably need $50,000 yearly. What is to become of the other $70,000, provided the coal production of Colorado does not exceed 12,000,000 tons? Again, why should coal-mine owners pay such a tax as this? They must necessarily pay taxes on their property the same as other property owners do, but why they should be made to pay a special tax and not the railroads, the ore mines, or any other industry, we do not understand. This is wholely repugnant to American principles, and inasmuch as in the framing of the bill the operators are given no representation in the selection of Mine Inspectors, or any other consideration, it certainly is a strong case of taxation without representation.

In framing the proposed law, the Commission failed entirely in providing for the most important requirement, if the safety of the miners and the mine is desired. That is, competent, technically informed mine foremen. There is no provision whatever for certified mine foremen; and even though the proposed law provides for "company inspectors," it does not require that they should be proven competent for such a job by any examination whatever.

In commenting on this proposed bill, we have merely taken up the worst parts of it. As a whole, it is vicious and impracticable, and if enacted in a law will tend to work an incalculable injury to the coal miners of Colorado, both in the matter of earnings and safety, and it is likewise unjust to the mine owners. It will tend to put a serious handicap on the future development of the coal mining industry of the state.

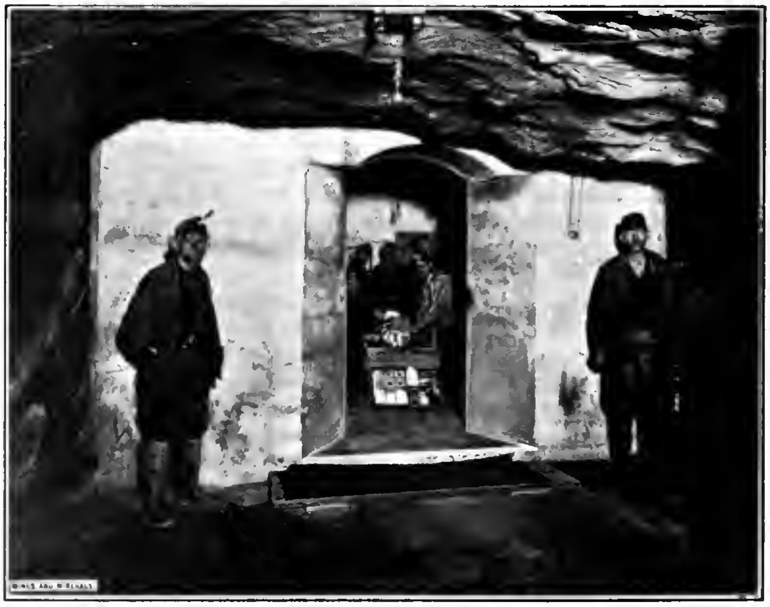
FIRST-AID WORK AT COAL MINES

*Written for Mines and Minerals, by John H. Ketner**

The very general movement on the part of mine owners, mine officials, and workingmen, in many of the coal fields of America, in the direction of "First Aid to the Injured" has been described in more or less detail in past numbers of Mines and Minerals; and the activity in this line on the part of the officials and employes of the Mineral Railroad and Mining Co., the Susquehanna Coal Co., the Lykens Valley Coal Co., the Summit Branch Coal Co., and the Lytle Coal Co. in the Schuylkill and Shamokin regions, has been previously referred to in a general way.

It is the purpose of this article, therefore, to describe in more detail the excellent means for first aid provided by the Mineral Railroad and Mining Co. and the Susquehanna Coal Co., in the Shamokin, Pa., region, and the excellent work of the officials and employes of the mines.

As in the case of the officials and employes of the Philadelphia & Reading Coal and Iron Co. and the officials and employes of the larger companies in the Lackawanna and Wyoming regions, the first-aid corps of the first-named companies held a competitive exhibition and contest on September 3, 1910, at Edgewood Park, Shamokin, Pa. The results of this field day were a revelation to the spectators and showed conclusively the value of the training and lectures given by Dr. J. M. Maurer, First-Aid Surgeon of the Mineral Railroad and Mining Co., and others engaged in the movement. At each mine in the vicinity of Shamokin and Mt. Carmel there is a first-aid corps composed of five intelligent miners and under-bosses, one of whom is appointed captain of the corps. These corps are given monthly drills in practical first-aid work, while the advisory surgeon directs the members in the proper methods of dressing and bandaging injuries. Lectures are also delivered by the surgeon and captain of each corps, the latter being chosen for his superior intelligence and talent for looking after the injured. The captains give their corps more frequent drills and impart to the members the rudiments of first-aid work as it has been



Exterior of Underground Hospital, Mineral R. R. & Mining Co.

*City editor *Shamokin Dispatch.*

given them by the surgeons. The results are that the corps at each mine is as efficient as any well-trained national guard ambulance corps, though detailed methods differ somewhat, owing to different environments and exigencies.

The mine hospitals are interesting and unique in their construction. They are located inside the mines, and when practicable they are small rooms cut into the rock. In all cases the walls and roof of the rooms are either the natural rock, whitewashed, or they are made of white concrete. They are constructed in the most durable manner, and are supplied with steam heat, and hot and cold water. Arrangements are also provided for sterilizing the water used. In the walls, closets or cabinets provided with iron doors are set. These cabinets


Interior of Underground Hospital

contain bandages, splints, sterilized gauze for wounds, picric-acid gauze for burns, roller and triangular bandages, stimulants, and appliances for treatment of shock. Each room also contains a dressing couch, a dressing chair, a stretcher, and a metallic first-aid case completely equipped with similar supplies and appliances as are in the cabinets. These cases are for use at the immediate scene of the accident in preventing hemorrhage and other deleterious effects. Each mine has an emergency hospital of this kind on every level, some having as many as four inside in addition to one on the surface. The corps connected with the surface emergency hospitals consist of outside men of various occupations, such as machinists, breaker men, blacksmiths, carpenters or any other class available for immediate service with sufficient intelligence to act properly at the proper time. The Mineral Railroad and Mining Co., and the Susquehanna Coal Co. maintain in the vicinity of Shamokin and Mt. Carmel about thirty such emergency hospitals, and they have proved invaluable in lessening suffering and aiding in the recovery of injured men.

In the furtherance of this humane work the workingmen take great interest and cooperate earnestly in the efforts of their employers to lessen the dangers and suffering due to inattention to injured men. The former inefficient and often unclean methods of handling wounds are no longer followed, and as a result, through modern methods, infection is prevented and lives saved, and permanent disability is often prevented. In addition, the example set by these emergency corps has been a great factor in educating the entire community in the necessity of up-to-date aseptic treatment of the numerous accidents and surgical contingencies of every-day life.

FIRST-AID GRADUATES, BRIER HILL, PA.

The first-aid-to-the-injured class of the Brier Hill Coke Co. graduated at Brier Hill, Pa., recently.


The American National Red Cross

Chartered by Congress

First Aid Diploma

This Certifies that David R. Millward has satisfactorily completed the course of instruction in FIRST AID, given under the auspices of THE AMERICAN NATIONAL RED CROSS, at Brier Hill, Pa.

Issued at Washington, December 27 1910

Chairman Central Committee

Chas. L. Magee
Secretary

Fig. 1. Certificate Issued to First-Aid Graduates

The class was organized by the Red Cross Society in April, 1910, and took the examination for certificates December 2, 1910. All of the 15 members passed a creditable examination, which was conducted by two physicians appointed by the Red Cross, Doctor Lang, of Pittsburg, Pa., and Doctor Lilly, of Brownsville, Pa. This class is the first to receive certificates in the soft coal or Connellsville region. The coke company gave the class all the support needed by furnishing them the necessary books and materials for practical work. Dr. Charles C. Gans, the company's physician, was the medical director, and gave the class lectures and practical work every week during the entire time. The meetings lasted from 1 hour to 1½ hours. All the men became enthusiastic over the work when once they understood the good results it could bring to themselves and fellow workmen. The doctor has found a decided difference in the men who are injured in the mine. They are bandaged or splinted as the case may call for, and instead of a large chew of tobacco being found on a cut it is nicely bandaged with the clean cloth from first-aid cabinets of Red Cross style which are furnished by the company.

Doctor Gans, who is organizing another class at the mine, expects it to be larger than the first one on account of the first class being more of an

Fig. 2. First Aid Graduates, Brier Hill, Pa.

experiment than anything else. Doctor Gans suggests that in organizing a first-aid class the bosses and superintendents should enter the first class, then later the working men. His first class was organized on this plan and included the mine foremen, assistant mine foremen, fire bosses, yard bosses, labor bosses, superintendents, and assistant superintendents, mining engineers, and then boss drivers, pit bosses, and so on, embracing the heads of different departments and fitting them for first-aid work. The plan worked admirably well, as one first-aid man can take a half dozen men and can tell them how to handle an injured man.

The members of the class shown in Fig. 2, were as follows: Thomas McCaffrey, superintendent and general manager; Robert McKay, assistant superintendent; W. M. Phelan, secretary of company; Harry Blackford, mining engineer; David Millward, mine foreman; Harry Millward, mine foreman; Thomas Rose, assistant mine foreman; Abe Gunter, fire boss; Richard McKay, fire boss; Andrew Dulik, fire boss; Ernest Elby, fire boss; Harry Hopkinson, chief machinist; John Scteffbouer, assistant machinist; John Noon, yard boss; Arthur Wilson, principal of Brier Hill Schools; Dr. Charles C. Gans, medical director.

Fig. 1 shows the certificate granted to the graduates by the Red Cross Society. The graduates of this class must feel satisfaction in knowing that they are the first in the soft-coal regions to be certificated by the Red Cross Society, but a greater satisfaction to know that they are able to relieve suffering humanity.

COMBUSTION OF COAL IN BOILERS

The quantity of smoke issuing from chimney shafts, and the number of articles upon "Smokeless Combustion," make it apparent that there is considerable difficulty in so arranging the apparatus and conditions as to obtain such perfect combustion of the fuel that no smoke shall escape from the boiler chimney, while at the same time only a slight excess of air and no unconsumed gases shall be found on analyzing the products of combustion.

Why is it difficult to obtain smokeless combustion? A little consideration of the composition and physical properties of coal, and the amount of air required for its combustion will give the reason. Take, for example, a Lancashire coal of the chemical composition, shown in Table 1.

Calculating the quantity of air required for the perfect combustion of this coal, we find that one pound of coal requires:

Coal		Air Required		Per Cent.
	Pound	Pounds	Pounds	
Fixed carbon	.6198	7.1896	7.1896	61.1
Carbon	.1802	2.0908	4.7560	38.9
Hydrogen	.0766	2.6657		
Nitrogen	.0162			
Oxygen	.0499			
Sulphur	.0064			
Ash	.0352			
Moisture	.0157			
	1.0000		11.9456	100.0

* Bulletin No. 18, Mining and Metallurgical Society of America.

Or one pound of coal requires 12 pounds of air to burn the hydrogen to water and the carbon to carbonic acid.

From these calculations it would appear that if 12 pounds of air were admitted for every pound of coal, combustion would be perfect; but as the volatile portion of the coal is liberated before the fixed carbon is consumed, and the volatile gases require a larger proportion of air than the fixed carbon, the time in which each of these constituents are consumed must be taken into account. In the case of a boiler consuming 12 pounds of coal per minute, and assuming that all the volatile gases are given off and consumed in one-quarter of a minute, the amount of air required would be as follows:

First quarter of minute for volatile matter,
$= 12 \times 4.756 = 57.072$ pounds of air,
$= 749$ cubic feet, or at the rate of 2,996 cubic feet per minute.

Three-quarters of a minute for the fixed carbon,
$= 12 \times 7.1896 = 86.275$ pounds of air, $= 1{,}133.64$ cubic feet, or at the rate of 1,511 cubic feet per minute.

The total quantity theoretically required $= 1{,}904$ cubic feet per minute.

From these figures, it will be seen that if the air supply is so regulated as to consume the volatile gases perfectly—2,996 cubic feet per minute—the quantity of air per pound of coal is greatly in excess of that theoretically required by the coal.

This excess of air carries off heat, and leaves it to be determined whether it is more economical to reduce the supply of air, thus allowing some combustible gases to escape, or have smokeless combustion with excess of air. If the volatile matter and fixed carbon burned away at a uniform rate, of course the air supply could be easily regulated, but this is not the case when coal is burned directly with air. It may be said that admitting the supply of air for the volatile matter above the fire-grate and that for the fixed carbon below and through the fire would solve the difficulty, but again the unequal rate of burning renders this impossible.

TABLE 1

Proximate Analysis

Fixed carbon	61.98
Volatile matter	32.29
Sulphur	.64
Ash	3.52
Moisture	1.57
	100.00

Ultimate Analysis

Carbon	80.00
Hydrogen	7.66
Nitrogen	1.62
Oxygen, by difference	4.99
Sulphur	.64
Ash	3.52
Moisture	1.57
	100.00

Stated in another way:

Fixed carbon		61.98
Carbon	Volatile matter 32.29 per cent.	18.02
Hydrogen		7.66
Nitrogen		1.62
Oxygen		4.99
Sulphur		.64
Ash		3.57
Moisture		1.57
		100.00

In the case of firing by hand, where, say, 100 pounds of coal is put on the fire at a time, it is doubtful whether it would be possible to admit the large quantity of air necessary for the volatile gases in time to consume them before they escaped from the chimney.

It is the usual practice to allow 24 pounds of air per pound of coal, instead of the 12 pounds theoretically necessary, and this is in agreement with the foregoing calculations; but even this amount of air, as a constant supply, will not give smokeless combustion unless the coal is put on the fire in small portions at a time, or is fed continuously by mechanical stokers. The extra 12 pounds of air carries away heat equal to about 9 per cent. of the heating power of the coal; in this example

Weight of air		Temperature		Specific heat		Heat Units
12 lb.	×	500° F.	×	.238	=	1,428

The quantity of air necessary for the perfect combustion of any coal, therefore, depends upon the proportion of volatile matter in the coal, the rate at which this volatile matter is given off in the hot furnace, and the rate at which the fuel is put upon the fire; the perfect admixture of the air with gases, and the heating of this air also tend to minimize the quantity of air necessary to prevent the escape of smoke from the chimney.—*The Engineer, London.*

In early November, 16-year-old Miss Jones lifted 52,800 tons of rock with 400 pounds of explosive. Miss Jones then fried an egg with the same kind of powder scattered on the ground to demonstrate its safety.—*Cleveland Press.*

ELECTRIC SHOCK IN MINES

*Written for Mines and Minerals, by Sidney F. Walker**

An English View of the Conditions Affecting the Use of Electricity Underground

Not long ago the Miners Union in the County of Durham, England, passed a resolution that no electrical apparatus should be used in the mine, and the manager of one of the largest coal mines in the county will not use electrical apparatus anywhere except in the immediate vicinity of the pit bottom. In 1909 the Association of Mining Electrical Engineers was formed in Great Britain, with branches in various places, which at the time of writing this article totaled more than 500 in membership.

The object of the association is to bring about an exchange of opinions between men engaged in the practical work of electricity in mines, as it is the very small matters that give the trouble and not the theory. It is the little bit of dirt where it ought not to be, the loose wire on the terminal, the loose cable end in the screw, the loosely made joint, the badly covered joint, the carelessly insulated cable, and so on, which lead to trouble. It is these and many more similar things that have been the cause of the greater number of deaths.

It has been stated that 80 per cent. of the members of the Association of Mining and Electrical Engineers are engaged in looking after apparatus in mines, the remainder being colliery managers, consulting engineers engaged in mining work, and a certain number of representatives of manufacturers. The use of electricity in mines is a three-sided problem at least.

There is the electrical side, there is the mechanical side, and there is the mining side; and the mining side is at least as important as either of the others. Electrical men cannot possibly know much about mining. They will acquire a knowledge of mining sufficient for their needs as they go along if they are wise and keep their eyes and ears open. The president of the association gravitated into mining work from electrical. He commenced life as an articled pupil to a firm that was fitting up electrical apparatus in mines, and was left in charge of an electrical plant at a colliery in the Midlands. While there he made use of the opportunity to study mining and to qualify as a mining engineer, and from that he gravitated to a subordinate position on the mining staff, and thence to that of the manager of the large concern he now superintends. Every electrical engineer engaged in mining would do well to follow his example, so far as studying mining conditions goes. It is a very old rule, originally applied to barristers, that a man should know everything of his own profession and as much as he can of every other. In the case of electrical engineers engaged in mining work, they should know everything about electricity, as far as time will allow, and as much as they possibly can learn about mining. In the carrying out of electrical installations in mines it often happens that the requirements of the electrical engineer run counter to the conditions that the colliery manager considers safe. The electrical engineer cannot easily understand the limitations that are imposed upon him, and it is only by a friendly exchange of views, a friendly exchange of thought, such as took place during the meetings of the branches of the association during last session, that the two views of the two branches of engineering can be reconciled.

There is another very serious point in connection with the use of electricity in mines; viz., the want of familiarity of the great bulk of electrical engineers with mining conditions. The conditions under which miners have to work are totally different to those under which men work on the surface, and miners may often be found carrying out operations under conditions that to the ordinary surface workman appear just as marvelous as those of the sailor on a topsail yard. In the navy in the old sailing-ship days they had a saying, "One hand for the Queen, and one for yourself," but in a great many instances both hands were given for the Queen. In mining work it is somewhat similar, often one hand for the colliery owner, and one for yourself, and again it very often happens that both hands have to be used in the service of the mine owner.

It is necessary to have electrical engineers to keep electrical apparatus in order, but they are not able to do as much work and to work as comfortably, or with as little danger, on apparatus in mines as they would work on the surface, until they have been working in the mines for a considerable time. When the writer was investigating the working of coal-cutting machines in British collieries some years back he found that the working miner who had been given a certain amount of instruction in the handling of a coal-cutting machine—sufficient to enable him to turn the power on and off, to guide the machine, etc.—produced far better work, cut far more coal, and that his machine gave far less trouble than the mechanic who was brought to the machine from the surface. The mechanic who was accustomed to working on the surface felt just as little at home at the face of the coal as a landsman would feel "outside the lift" on a topsail yard. This, it will easily be understood, leads to many difficulties in the way of keeping apparatus going. It leads to the requirement of a considerably larger number of men to look after electrical apparatus than would otherwise be necessary. In a great many instances a man executing a repair to a cable, to a switch or distribution box, to a motor or its controller, is working under very awkward conditions, and is only too anxious to get it done and get away. If this side of the question be considered, the writer thinks that the toll of life, and the number of accidents that have taken place in British mines, will be seen not to have been so large as from their actual number they would appear.

The writer has summed the matter up on several occasions in two words; viz., insulation and care. If the insulation of the whole of the apparatus is maintained at a certain standard, the chances of shock are reduced to a minimum, and the writer believes would be very few indeed. If to the maintenance of good insulation everywhere, care were added on the part of every one concerned, he believes that accidents would be practically eliminated. One of the most striking features which came out in the discussions which took place at the branches of the Association of Mining Electrical Engineers was the fact that a very large number of the accidents in mines were due to want of care. In some cases the carelessness actually approached foolhardiness. Men who knew, or should have known, the risks which they took in doing certain things, deliberately did them, and were killed. One very striking instance will probably illustrate the matter. At a certain colliery there was an extra high-tension switchboard working at somewhere about 3,000 volts or over. Some alterations were required to the connections at the back of the board, and that portion of the apparatus was disconnected from the service and carefully rendered harmless. The other portion of the space at the back of the board, where the live conductors were fixed, was carefully divided off from the portion where the alterations were being carried out, so that unless a man wilfully and knowingly removed the division it was practically impossible for him to do himself any harm. The man who was carrying out the alterations was a highly skilled electrical engineer, in receipt of a fairly substantial salary. Apparently he was in a hurry to get away to a football match, or something of that kind, and after carrying out the alterations to the portion of the board required, he deliberately removed the partition mentioned above, either to get something behind the other portion of the switchboard or to get at some connection there, and accidentally touching a live conductor, was killed. In another case, which is also illustrative of the want of care, a comparative youngster, who was not so fully imbibed with respect for the killing properties of electrical conductors as he ought to be have been, made connection for testing purposes between two conductors at the back of the board by means of a piece of stiff bent copper wire held in his

* Bloomfield Crescent, Bath, England.

hand. By accident he also made connection with another live conductor, between which and one of the conductors he was connecting considerable pressure existed. Fortunately for himself, the connection was only for an instant. The result was a powerful flash, which blinded him for some weeks, and which it is to be hoped was a lesson to him. In this case also the youngster had a certain amount of knowledge, which ought to have kept him from playing monkey tricks such as that described. A more serious monkey trick that a youngster played at a central generating station, and which also resulted in temporary blindness to the operator, was the throwing of a piece of fuse wire up in the air, in the neighborhood of a switchboard; the fuse wire fell upon a pair of conductors between which a considerable pressure existed, and being immediately melted, a blinding flash followed, with the result mentioned.

Shocks occur when a man either touches one conductor between which and the ground upon which he stands a considerable difference of pressure exists, or accidentally touches two conductors between which a considerable difference of pressure exists, by his two hands, or two other parts of his body. Cases have occurred in which men have accidentally made contact with their heads against a conductor, between which and the ground on which they stood a considerable difference of pressure has existed. In other cases men while holding one conductor with the hand have accidentally made contact with another conductor at a considerable difference of pressure, with either hand, head, or some other part of the body. More than one accident has occurred from a man's back coming in contact with a conductor. In the upper part of the back, it will be remembered, the delicate organs of the chest are exposed, and a conductor touching the skin there a current passes through the lungs, which may easily cause such serious damage as to destroy life.

How the Conditions Occur Under Which Shocks May Be Taken.—The conditions under which shocks have been received by men working in mines, arise in two ways, both being really due to want of insulation. In one class of cases the insulation of some portion of the apparatus has either perished or been damaged in some way, and a man has caught hold of what he has supposed to be an insulating envelope, or a metal or some other body that was insulated from the service, and has actually brought his hand into more or less intimate connection with the service itself. One fruitful cause of accidents of this kind is the imperfect covering of joints. In mines generally, and in certain British coal mines in particular, falls of roof are somewhat frequent. Cables furnishing current for lamps and motors have been suspended from insulators secured to props by the side of the road. Falls of rock have sometimes merely damaged the insulation of the cable, and sometimes have parted the cables. In either case, if the damaged insulation is not made quite good, if the covering of the joint, or of the damaged portion of the cable, is not made of as high an electrical resistance as that of the cable before the accident took place, a man touching the presumably covered joint may receive a shock that will be fatal. Unfortunately, numerous cases have occurred of this kind. In the West of Scotland, for instance, it is the practice for what are called "brushers" to go into the mine after the coal has been filled into the trams to clear out the coal dust. In one instance at any rate, and probably in more, a "brusher" has found a cable in his way and has caught hold of it, believing it to be perfectly harmless, as it was apparently insulated, and has attempted to move it out of the way, with the result that he has received a shock and been killed. A striking instance, that was mentioned in the discussion upon shock at the writer's branch, of the danger of imperfectly covering joints was the following: A haulage motor had stopped working and an electrician had been sent for to put it right. He found a wire broken in the motor, repaired it, and, as he thought, covered it, leaving the machine running all right. He proceeded to go out of the mine, but when he had walked a little way, as men often do, feeling anxious about the work, he returned to see that it was still working all right. Unfortunately, the motor being stopped between journeys, he touched the joint that he had made and received a shock by which he was killed. In this instance it was apparently want of knowledge that was at the bottom of the trouble. The man thought he had covered the joint sufficiently. According to the information that was given during the discussion, the joint was covered with wet tape and of course the insulation resistance was very low.

In other cases, what ought to have been perfectly harmless metallic bodies have been rendered alive, and have been the cause of fatal shocks to men touching them, through defective insulation of some parts of the electrical service. Cases such as the following have been again unfortunately only too frequent. Men walking to the face, or returning from it, have caught hold of the galvanized iron electric signal wires that form part of the equipment of nearly every engine plane in British collieries, and have received shocks from which they have died. In other cases, a man has caught hold of the iron edge of a mine wagon and been killed. Again, a somewhat fruitful source of fatal shock has been the haulage rope of coal-cutting machines. In British collieries the longwall coal-cutting machine is very largely employed. It will be remembered that in the working of this machine it is usual to have a working face ranging from 100 yards upwards. The machine, whether it is one of the disk, the bar, or the chain type, is drawn along the face, cutting its groove as it goes along. For the purpose of hauling the machine along the face a small galvanized-steel haulage rope is carried out in front of the machine, and is wound up upon a small haulage drum attached to the machine, as it moves along. The length of the rope being limited, it has to be moved forward from time to time, during each cutting shift, as the machine comes up to the prop to which the rope is attached. It is the work of an unskilled man who goes with the machine, usually to move the rails where rails are employed, and to carry forward the haulage rope, making a fresh attachment in front of the machine, and seeing that the haulage gear is working properly. In several instances the carcase of the machine has become connected to the electrical service through the defect of the insulation of a cable, or some other part of the apparatus, and the whole machine, and all the cables that are connected with it, including the haulage rope, have been made alive. The result has been that a man taking hold of the haulage rope, standing upon the ground, has received a shock that has killed him.

EXPORT OF COAL

Coal is becoming one of the most important and useful articles of exportation in the foreign trade of the United States. The important items of exportation are: cotton, approximately $530,000,000; meat and dairy products, $130,000,000; copper and mineral oil about $94,000,000 each; lumber about $40,000,000; corn about $28,000,000; wheat, approximately $23,000,000; leather, $38,000,000; and tobacco, $36,000,000. While, of course, cotton, meats, copper, and mineral oil are of much larger value in their respective totals, coal now exceeds in value of its exportations corn, wheat, tobacco, lumber, leather, and numerous other articles which have been looked upon as important factors in our export trade, and the growth in the exportation of coal has been more rapid than that in many other leading articles of exportation. About one-third of the $45,000,000 worth of coal exported, or $15,000,000 worth, is anthracite; about $27,000,000 worth bituminous coal, and about $3,000,000 worth coke. The growth in the exportation of bituminous has been of late more rapid than that of anthracite, the increase in bituminous in 1910 over 1908 being more than $3,000,000, and in anthracite an increase of about $1,000,000 in the stated value as shown by the export figures published by the Bureau of Statistics of the Department of Commerce and Labor.

METHODS OF REMOVING COAL PILLARS

*By F. W. Cunningham**

Different Methods in Use in Pittsburg Coal Seam—Their Advantages and Disadvantages

The subject of this paper embodies one of the most important features in coal mining. It carries with it a combination of perplexing and difficult problems, among which are the following: How is it possible to afford the greatest degree of safety to the workmen, and recover the largest percentage of coal with the least possible danger to the property from squeezes, creeps, etc., and at the same time to arrange the system of mining so as to produce the largest percentage of lump coal and the smallest percentage of fine coal? If the operator is to continue in the business of shipping coal, make a reasonable profit on his investment with the present prices and profitably recover all the pillar coal, he has indeed a serious problem to solve.

It can readily be seen that this is a broad subject, because each coal seam has its own local difficulties to contend with, therefore, not having the time to prepare an article that may cover all conditions, the Pittsburg coal seam is used as an example; however, where conditions are similar in other localities the remarks will apply.

The first thing to be considered is the size of the pillar and the room. These matters will depend on the thickness of the cover, which determines the roof pressure, strength of roof, and whether the floor is hard or soft. A soft floor or hard roof requires wide pillars, while with a hard floor and a soft roof the width of pillars may be less. The inclination of the seam should be considered, as a wider pillar should be left where dip is steep than where the inclination is less. The thick seam requires a greater width of pillar than one that has less thickness.

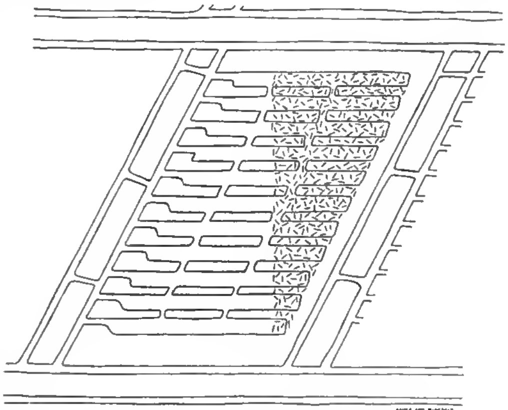


FIG. 1. SHOWING FACES OF PILLARS PERPENDICULAR TO THE SIDE

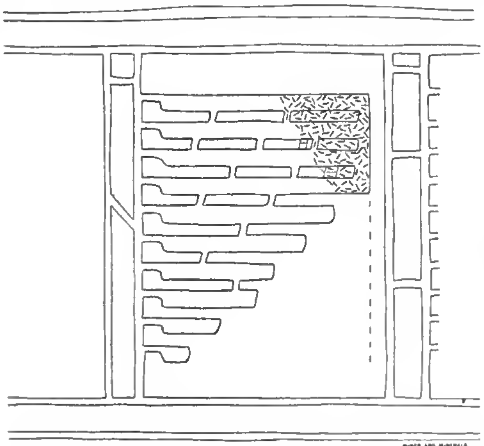


FIG. 2. PILLAR DRAWING, FACES AT ANGLE TO SIDE

The physical structure of the coal has a bearing on the thickness of pillars to be left; for instance, a soft or slippy coal requires wider pillars than a hard compact coal. The width of the rooms will depend largely on the width of the pillars, and the strength of the roof rock.

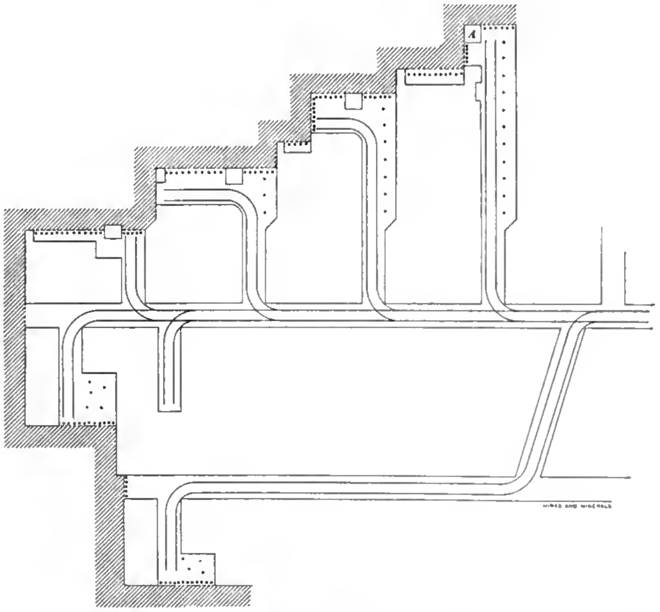


FIG. 3. RIB WORK WHERE CURVE IS USED, SHOWING LOCATION OF TRACKS AND POSTS

It is impossible to give exact rules for determining the proper size of pillars, as each case in practice requires special consideration.

The roof pressure per square foot exerted on pillars is proportional to the thickness of the cover and the total width of the room and pillar, and inversely proportional to the width of the pillar.

Let depth of cover $= d$; width of room opening $= O$; width of pillar $= w$.

The roof pressure per square foot exerted on the pillars is proportional to the expression, $d\left(\frac{w+O}{w}\right)$.

There are a number of ways of drawing pillars, or at least a number are being tried, but when all are classified they may be divided into drawing pillars at the end and drawing pillars in sectional divisions. In drawing pillars on the end the first attack is made by cross-cutting the pillar after the room or entry is driven its distance. This furnishes a retreating face that permits the pillar to be worked back, sometimes square and sometimes by offsets, depending on the width of the pillar, the strength and thickness of the coal, and the nature of the overlying strata. In the second method of drawing, the pillars are cross-cut into sections along their length so they may be drawn by end or side slices, depending on the nature of the coal, inclination of the seam, the thickness of the overlying cover, and the kind of roof.

The greatest percentage of pillar coal is obtained under the system that does not allow the pillars to be squeezed.

The two general systems of drawing are: (1) All the pillars in a panel are taken out together with the end lifts in such a way as to keep the faces of all the pillars stepped in line, or echelon, and perpendicular to the sides of the pillars, Fig. 1; (2) the pillars are drawn by cross-cutting through the fast ends of the innermost pillars in a panel; then two or more pillars, according to the circumstances, are drawn and the centers of the faces of the pillars are made to lie in a straight line that makes an angle of 35 to 50 degrees with the sides of the pillars, Fig. 2. It is very important that the pillars should be kept in the line of roof fracture so that each shall support its proportional share

* Read before Coal Mining Institute of America, December, 1910.

of the overlying strata, then the roof pressure will not bear too heavily upon any one of them and crush the coal. The timbers supporting the roof should be drawn to within a short distance behind the end of the pillars so as to induce a roof fall up to them.

Before pillar drawing is commenced the following system should be planned and carried out unwaveringly: Draw the

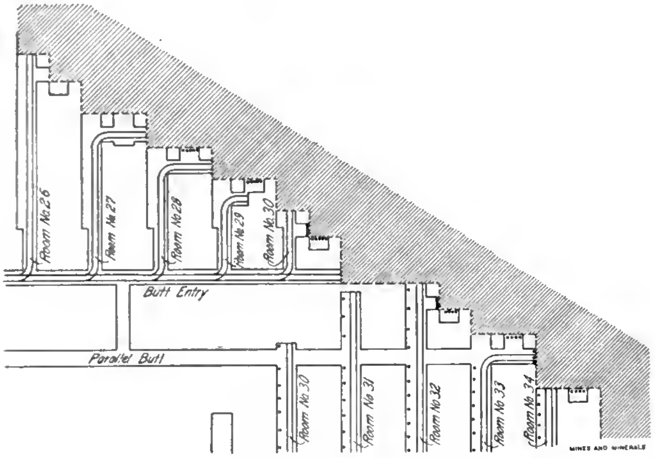


Fig. 4. Rib Work Showing Method of Working Ends of Pillars

room pillars back regularly so soon as the rooms reach their limit. This will avoid the possibility of starting a creep or squeeze by holding open too large an excavated area. Before room pillars are drawn, however, the effect their withdrawal will have on adjoining workings and at the surface should receive attention.

Before the entry pillars are drawn, it should be ascertained whether all the coal possible has been mined through these entries, and whether or not the entries will be required for later development of the mine.

Among other causes, squeezes in the mines are sometimes due to the improper drawing of pillars and to leaving pillars too small and in the wrong places.

The topography of the surface relative to hills and dales should be considered when starting to draw pillars, and relative to this subject a question may be asked, which is an important one; viz., How many coal properties have contour maps of the surface? While it is true that some coal companies require these data, a number of operators consider them an expensive luxury as they did the mine survey in years passed, and if their mines are examined it will be seen that very few have escaped squeezes. Suppose, for example, the rocks at the surface rise abruptly each side of a narrow valley to say 200 or 300 feet. Would it be proper to commence the pillar drawing under this valley? And if so, what effect would it have on the adjoining pillars, considering that no provision has been made in the plan of the mine for the surface conditions, and the mine is worked as are those usually in the Pittsburg district?

In such cases the cover, especially that directly above the coal, is a very important factor in determining the width of rooms and pillars. With strong roofs, wide rooms may be worked, but with increased width comes increased weight upon the room pillars. With weak roof and thick cover the conditions necessitate narrower rooms and larger pillars. Therefore, in view of the fact that there is no exact rule to predetermine the size of pillars, width of rooms for the different conditions met with in the mines, the nature of the overlying strata, only as experience has taught locally, it is suggested that the size of pillars and width of rooms be such as to effectively support the heavier and weaker strata that overly the mine, irrespective of the stronger and lesser thickness of the overlying cover in other parts of the mine.

In the removal of pillars, and regardless of the method adopted in their extraction, systematic timbering should be carried on by competent persons employed for that purpose in order that the timbering be done in accordance with the system adopted.

In the Connellsville coke region systematic timbering and pillar drawing is more carefully followed than in mines outside of this region. It has been found that by strictly enforcing the rules relative to timbering and pillar work it is possible to maintain perfect fracture lines in the roof, some of which are over a mile in length. The overlying weight is therefore evenly divided on the end of the pillars throughout the whole line. By making use of the fracture lines and proper timbering, it is possible to recover practically all of the coal.

In the Connellsville coke region the large pillar and narrow room system is adopted; very little coal is cut by machines, and the smaller and finer the coal, the better it is adapted to coking purposes.

Pillar drawing in the Pittsburg coal seam, especially in those mines outside of the coke region, that ship coal to market, has been so irregular that a vast amount of coal left in the pillars will never be recovered. This is due to the fact that most of the coal mined from the pillars is small and the market is not active for this sized product. The operator to meet the demands of the coal market frequently issues orders to stop the pillar work, and especially when the trade is slow this is a common order. The results are that an excessive number of rooms are driven their distances and pillars stand a number of years before an attempt is made to pull them. When the pillar work is again started, the rooms are required to be retimbered and old falls cleaned up, making an expensive operation that could have been avoided had the pillars been mined at the proper time. Frequently, after a suspension of pillar drawing, when a panel of pillars is again attacked a squeeze makes its appearance and extends over a large area before it is stopped.

The remedy for this would naturally be (and is suggested by the practice in the coke region) to leave large pillars and narrower rooms, a system of mining which would probably overcome the difficulty so far as squeezes are concerned. But the product of the mine is not used for coking purposes, nor is it put on the market as slack or fine coal.

The operator in the Pittsburg coal field, with the price of coal where it is today, must get the largest percentage of lump

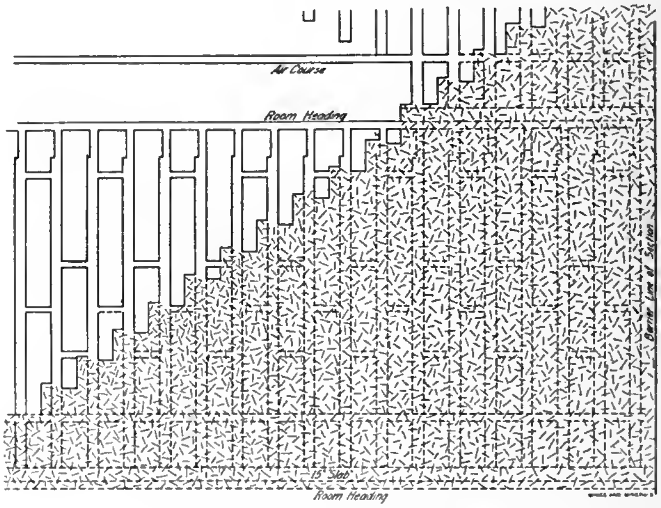


Fig. 5. Pillar Drawing by Steps

with the least amount of fine coal, and this by machine mining, in order that he may compete with coal operators in other fields. To accomplish this the rooms are driven as wide as is possible in order to obtain the greatest percentages of lump coal, and the room pillars are left as narrow as possible, for from them comes the greatest percentage of crushed coal. Under such conditions it is readily seen that the coke region system of wide

pillars and narrow rooms does not appeal to the operator whose product goes on the market for other than coking purposes.

A system that would appeal to the operator in this strait, is the one in which the greatest amount of lump coal can be recovered by the use of mining machines, and the least amount of pick work is needed.

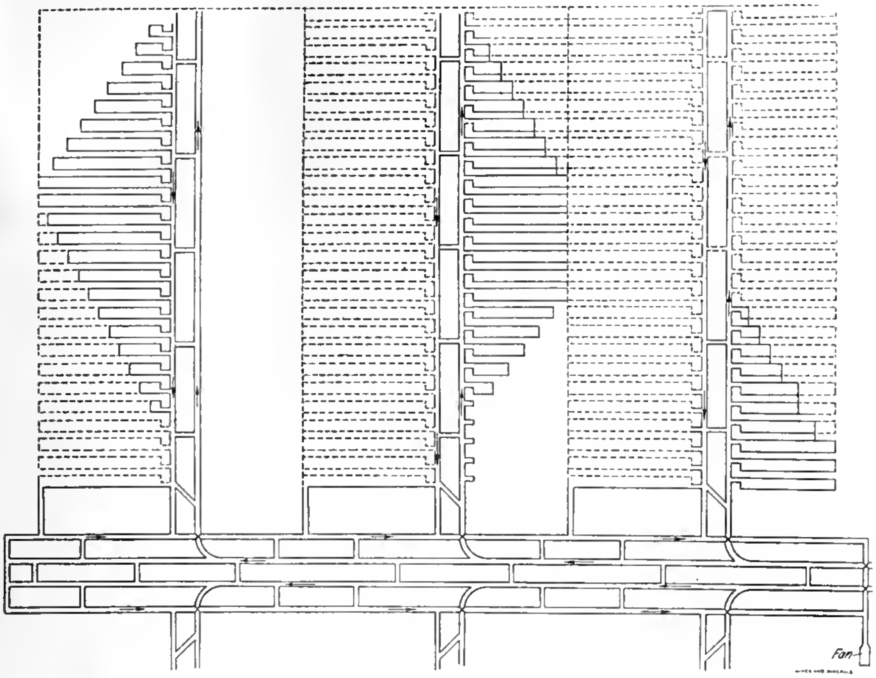


FIG. 6. PLAN OF PILLAR DRAWING USED IN PITTSBURG DISTRICT

The sizes of pillars are regulated by the depth of the overlying cover, physical structure of the coal, kind of roof and floor, and the inclination of the coal seam.

The width of the room is regulated according to the width of the pillar, excepting in the Connellsville coke region where the narrow room and wide pillar is adopted.

Pillar drawing, either on the end or in sectional divisions, requires that the ends of the pillars be kept in perfect alinement, whether or not the line of roof fracture is perpendicular or at an angle of 50 degrees or less with the room pillar.

Systematic timbering is necessary in the extraction of pillars and should be carried a short distance behind the pillar. A competent person or persons should look after the pillar work, to see that the timbering and the proper mining of the coal are strictly in compliance with the rules laid down for such work.

Pillar work should be so arranged that when a room is driven its distance the pillar could be started back immediately, with the end of this pillar stepped in line with the other pillars that are being drawn back in the section. This mining system avoids too large an open area being supported by room pillars longer than is necessary to remove the pillars. If the mining is not carried on with regularity, and there is a large open area supported by room pillars, especially if they are small and irregularly worked, squeezes are likely to cause trouble and the workmen encounter a greater degree of danger.

To show some of the methods adopted in pillar drawing the following plans are presented for consideration: Fig. 3 represents rib work showing where curves are used, and the location of the tracks and posts. This method is principally followed in the coke region; it is being replaced by later methods which have proved to be more successful.

Fig. 4 is a plan showing rib work, the location of tracks and posts, the method of working the ends of the pillars. The difference between this and the method shown in Fig. 3 is that the chain pillars are worked from the rooms off the next butt entry. It has an advantage over the method shown in Fig. 3, in maintaining a more uniform fracture line on the chain pillars.

Fig. 5 is a plan showing rib work without curve, a slab taken off the right-hand side of the room, also chain pillars taken out from rooms on the next lower butt. This system is adapted to the coke region or where narrow rooms are worked and they do not have to contend with the storing of refuse along the side of the rooms; however, it could be worked from wide rooms with the refuse in the middle of the room and a track along each rib. This system has been tried at some mines in the Pittsburg district, with a track along each rib; it did not prove a success.

Fig. 6 is a plan of rib work used in some of the mines in the Pittsburg district with a wide room and a narrow pillar and the track along one side of the room; the refuse on the other side fills the rooms in some mines to within 2 or 3 feet from the roof. The rooms are worked from 21 to 24 feet wide and the pillar 15 feet thick.

Fig. 7 is another plan of rib work used in some of the mines in the Pittsburg district, a wide room with a narrow pillar. The pillar work starting on the first end of No. 1 butt, then being worked to the top, then from the top of No. 2 butt to the bottom of this butt. One advantage in working the pillars as shown by this plan is that the pillar work is always in the return end of the air-current, then if open lights are used in the rooms, the air-current passing the goaf would not be carried by the rooms where open lights may be permitted. The methods in Figs. 6 and 7, if carried on regularly will give as good a result as any that have been tried in this district, that is, considering lump coal and it cut by mining machines.

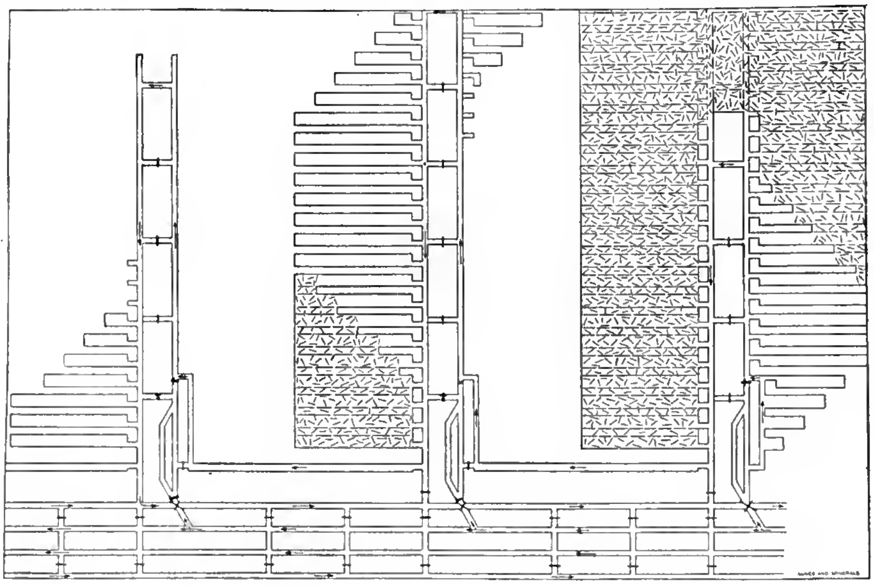


FIG. 7. PILLAR DRAWING WITH WIDE ROOM AND NARROW PILLAR

Fig. 8 is a plan showing the amount of coal left in the room pillars where the system is the road in the middle of the room

and the room pillars not being drawn. These room pillars are left as narrow as possible after the room has advanced 100 feet, or to the first cut-through, then as a general rule, about the time the room is up its distance, the room caves and the coal in the pillars is lost; however, the coal in the first end of the room pillars is very often all recovered by working the pillar along the side of the fall up to a point where it is considered dangerous to advance farther. There is one company in my district that mines over 2,500,000 tons of coal annually by this system, and their claim is that they recover 90 per cent. of marketable coal. There are features connected with this system that I do not approve of, and do not think it a proper system to be used in this district.

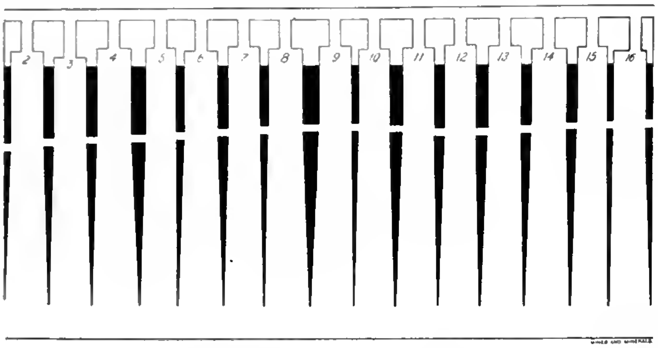


Fig. 8. Method With Narrow Pillars

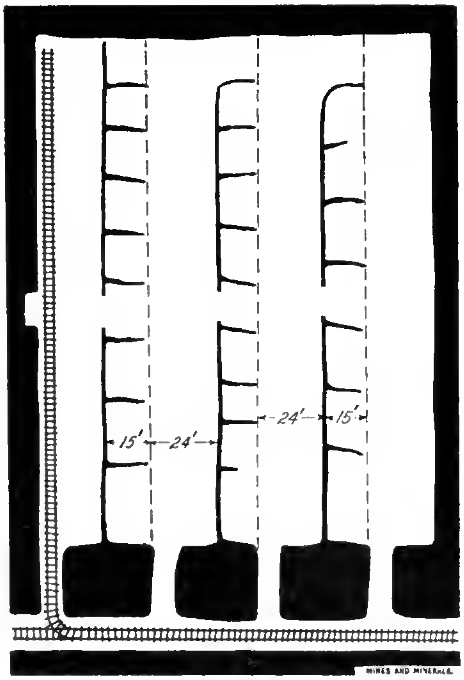


Fig. 9. Showing Coal Left in Room Pillars

Fig. 9 is a plan showing the amount of coal left in the room pillars on rib work where the system of mining is the road on the side and draw the pillar; the room is 24 feet wide and the pillar about 15 feet wide, where it is impossible to keep the goaf from sliding down where the miner is doing his work, as is usually the case in many mines in the Pittsburg district. The coal left in the pillar, as shown on the plan, varies from 12 to 18 inches thick, and this coal is left next the goaf on each cut the miner makes across the rib. There is about as much if not more coal left in the pillar where this system is carried out as in the system represented on Fig. 8, and very often blocks of coal in addition to what is represented on the plan are lost. I do not want to be understood that it is necessary to leave this sheet of coal next the goaf throughout the whole Pittsburg district, but there are sections in this district where it is impossible to keep the rib falls from taking the face and knocking out any posts set to prevent it. Where this condition exists it is necessary to make a new cut across the rib each time a fall is made.

Figs. 10, 11, and 12 are sketches showing location of tracks, posts, and the size of falls to be made. This is one of the latest systems and is claimed to be the best yet adopted in the coke region.

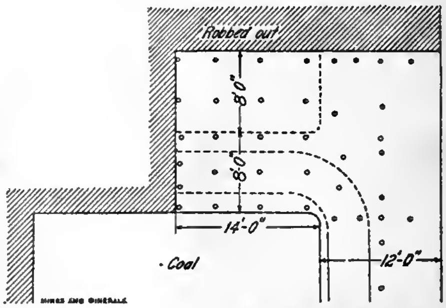


Fig. 10. Coal Extracted, Place Ready to Make Last Fall in 42-Foot Center Rooms

Fig. 13 is a plan showing where 70 per cent. of the pillars were recovered by mining machines; the machine would make a cut 21 feet wide through the rib, then leave a stump of 8 or 9 feet to be taken out by pick mining, as shown by the different cross hatching. In this particular mine they have been very successful in the removal of the pillars by this system and up to this writing, covering a period of over one year, there was no coal lost nor is there any appearance of a squeeze or creep. The surface is from 150 to 250 feet above the coal.

Fig. 14 is a sketch of longwall retreating in a modified form. This plan is suggested in order to eliminate the cost of narrow work in turning rooms and making cut-throughs to recover a larger percentage of the coal, and to reduce the chances for a squeeze or creep to a minimum. It is realized by the writer that this system cannot be adopted at most mines, but there are some mines in the western part of this state where he feels confident that the system, if carried out properly, could be worked to a good advantage.

The system requires the butt entries to be driven up their distance, then cut across the block of coal toward the next butt entry one-half of the distance, usually 150 feet; then a cut is made from the next butt entry toward this one until the two cuts meet, the one being kept about 25 feet in advance of the

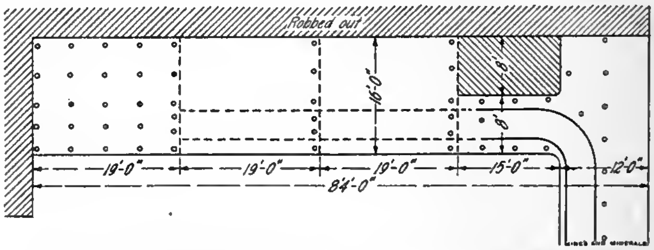


Fig. 11. Posting and Size of Falls in 84-Foot Center Rooms

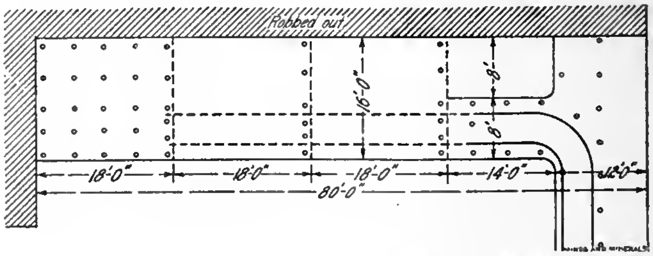


Fig. 12. Posting and Size of Falls in 80-Foot Center Rooms

other; the track is kept close to the face, the timbering will require three or four rows of posts in order that the falls will not take the face; the coal could be cut by mining machines and quite a large number of loaders could be worked on each lift; the ventilation could be very easily controlled and the workmen receive the benefit of the air-current while doing their work. If it is found that the system works to a better advantage

on the butts, the system could be changed by driving two rooms through the block, then using the same method in the middle of the block that is shown at the end in the sketch.

PARAFFINE OIL IN OHIO MINES

Section 974 of the Mining Laws of Ohio, provides: No person, firm or corporation shall compound, sell or offer for sale,

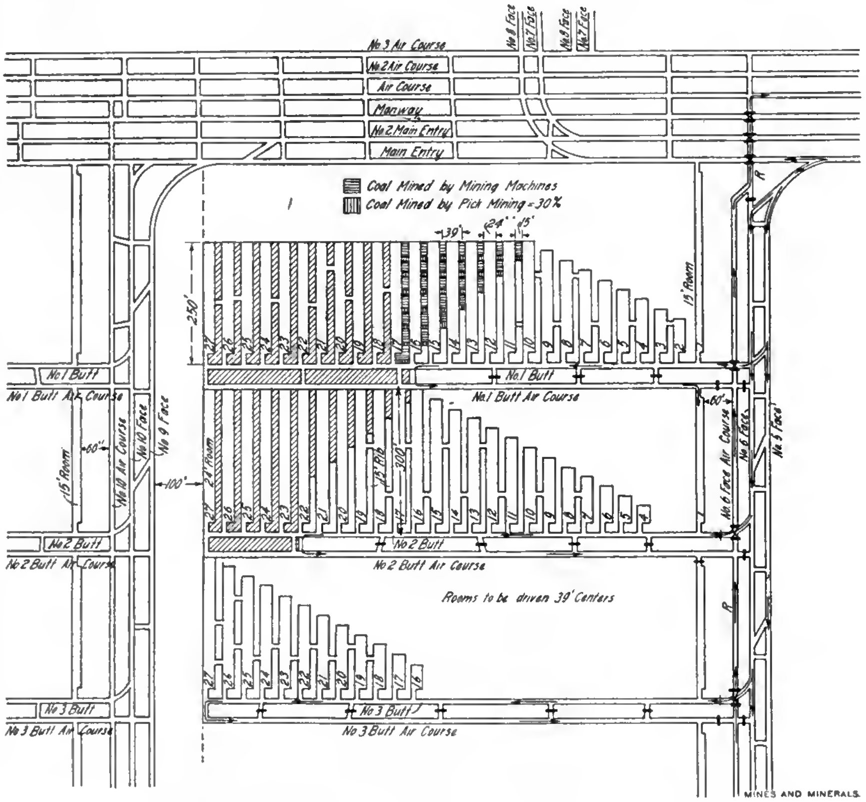


Fig. 13. Method Where Pillars Are Recovered by Mining Machines

for illuminating purposes in any coal mine, any oil other than oil composed of not less than 84 per cent. of pure animal or vegetable oil or both, and not more than 16 per cent. pure mineral oil. The gravity of such animal or vegetable oil shall not be less than 21½°, and not more than 22½° Baume scale, measured by Tagliabue or other standard hydrometer, at a temperature of 60° F.; the gravity of such mineral oil shall not be less than 34°, and not more than 36° Baume scale, measured by Tagliabue or other standard hydrometer, at a temperature of 60° F., and the gravity of the mixture shall not exceed 24° Baume scale, measured by Tagliabue or other standard hydrometer at a temperature of 60° F. Paraffine wax reduced to a liquid state is a mineral oil, and contains no animal or vegetable oil; this fact alone, under the provisions of Section 974, just quoted, would make it unlawful for any person, firm or corporation to sell or offer it for sale for illuminating purposes in any coal mine in the state of Ohio; and the use of the same by any person or persons for illuminating purposes in mines is prohibited by Section 975 of the mining laws. Another reason is that the law provides that the specific gravity of the mixture shall not exceed 24° Baume scale, measured by Tagliabue or other standard hydrometer at a temperature of 60° F. As reported to me, the gravity of paraffine oil is 34° Baume scale; therefore, the use of paraffine wax reduced to a liquid would be prohibited for that reason also.

I am of the opinion that paraffine wax can not be sold or used as an illuminant in mines for open lamps, because it does not come up to the requirements of Section 974 of the Mining Laws of Ohio. Timothy S. Hogan, Attorney General

GRAVITY DETERMINATION OF COAL

*By Abraham G. Blakeley and Edwin M. Chance **

During the routine work of this laboratory, it is necessary to determine specific gravity on a large number of samples of anthracite coal. As the methods available gave but little satisfaction commensurate with the time and labor entailed, we had recourse, after considerable experimentation, to the herein described procedure.

The apparatus, first of all, is simple, and may be found in almost any laboratory. The first requirement is a stout flask of about 250 cubic centimeters capacity, with a long and rather slender neck, terminating in a flaring mouth, such as the "copper determination" flask supplied by the Denver Fire Clay Co. This flask is to be marked at the base of the neck, either by a file scratch or by etching, and its volume up to this mark determined by means of a 100-cubic-centimeter automatic pipette and a 100-cubic-centimeter burette, graduated in .2 cubic centimeter.

The burette should have a three-way stop-cock, and be connected with a reservoir either below or above the level of its top. In the former case suction is used to fill the burette, while in the latter case gravity is the means. It is better to use the elevated reservoir and gravity feed for the pipette. Of course it is understood that a plain burette and pipette can be used, the form described merely having the advantage of rapidity.

The flask is calibrated as follows: The flask is rinsed with water, inverted, and allowed to drain for 1 minute; 100 cubic centimeters of water are then added from the pipette; the flask is then filled to the mark from the burette. In calibrating by this method, inaccuracies in either pipette or burette do not militate against the accuracy of the specific gravity result, as the same method is pursued in the final determination.

The actual determination is carried out as follows: 100 grams of coal previously crushed to about 4 mesh, and carefully sampled, are weighed to .1 gram, and placed in the

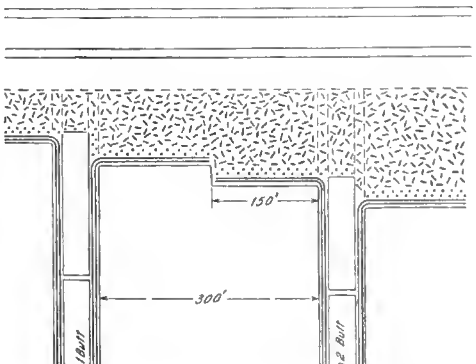


Fig. 14. Modified Longwall Retreating

flask; 100 cubic centimeters of water are run in; a one-hole stopper fitted with bent glass tube and rubber connection is inserted in the mouth of the flask; suction is applied, and the flask is shaken, care being taken that no water is splashed

* Chemists for Philadelphia & Reading Coal and Iron Co.

into the glass tube. When air bubbles cease to be disengaged, the suction is interrupted and the stopper withdrawn. Water is now run in from the burette till the flask is filled to the mark, care being taken to work down any coal which may have stuck to the neck.

The results may be calculated as follows:

$$\text{Specific gravity} = \frac{100}{V_0 - V_1}$$

wherein V_0 = volume of the flask; V_1 = number of cubic centimeters of water added to coal.

No calculation at all need be made if a table of reciprocals be available, as specific gravity = 100 × reciprocal of $V_0 - V_1$. As an example, 100 grams of coal were taken; volume of flask, $V_0 = 262$ cubic centimeters; volume of water added to coal, $V_1 = 197$ cubic centimeters;

$$\text{Specific gravity} = \frac{100}{262 - 197} = 100 \times \frac{1}{65} = 1.54$$

After making a determination the flask is emptied and rinsed, then allowed to drain for 1 minute, when another determination may be made.

This method is very rapid and so simple that an inexperienced operator can without difficulty make over 80 determinations per day, accurate to within .02 unit of specific gravity. It is, of course, self-evident that this method has a wide range of applicability; the gravities of such materials as rocks, drillings, etc. may be determined with ease.

While the authors claim nothing new or radical in this process, still they submit it as a procedure by which results of moderate accuracy can be obtained with great rapidity, while the outlay for special apparatus is practically nil.

Weight of 1 Cubic Foot at Different Specific Gravities

Specific Gravity	Weight Per Cubic Foot	Specific Gravity	Weight Per Cubic Foot
1.40	87.3	1.63	101.6
1.41	87.9	1.64	102.2
1.42	88.5	1.65	102.8
1.43	89.2	1.66	103.5
1.44	89.8	1.67	104.1
1.45	90.4	1.68	104.7
1.46	91.0	1.69	105.3
1.47	91.6	1.70	105.9
1.48	92.3	1.71	106.6
1.49	92.9	1.72	107.3
1.50	93.6	1.73	107.9
1.51	94.1	1.74	108.5
1.52	94.8	1.75	109.1
1.53	95.4	1.76	109.7
1.54	96.0	1.77	110.4
1.55	96.6	1.78	111.0
1.56	97.2	1.79	111.6
1.57	97.8	1.80	112.2
1.58	98.4	1.81	112.8
1.59	99.1	1.82	113.4
1.60	99.8	1.83	114.0
1.61	100.4	1.84	114.6
1.62	101.0	1.85	115.1

Calculated on basis of 1 cubic foot of water weighing 62.35 pounds at ordinary temperature.

Table of Reciprocals for Determining Specific Gravity of Coal

Number	100/Number	Number	100/Number	Number	100/Number
53.0	1.89	62.0	1.61	71.0	1.41
53.5	1.87	62.5	1.60	71.5	1.40
54.0	1.85	63.0	1.59	72.0	1.39
54.5	1.83	63.5	1.57	72.5	1.38
55.0	1.82	64.0	1.56	73.0	1.37
55.5	1.80	64.5	1.55	73.5	1.36
56.0	1.79	65.0	1.54	74.0	1.35
56.5	1.77	65.5	1.53	74.5	1.34
57.0	1.75	66.0	1.52	75.0	1.33
57.5	1.74	66.5	1.50	75.5	1.32
58.0	1.72	67.0	1.49	76.0	1.32
58.5	1.71	67.5	1.48	76.5	1.31
59.0	1.69	68.0	1.47	77.0	1.30
59.5	1.68	68.5	1.46	77.5	1.29
60.0	1.67	69.0	1.45	78.0	1.28
60.5	1.65	69.5	1.44	78.5	1.27
61.0	1.64	70.0	1.43		
61.5	1.63	70.5	1.42		

TO PREVENT DUSTING OF CONCRETE FLOORS

By Albert Moyer, Asso. Am. Soc. C. E.

Cement floors, particularly in office buildings or warehouses, which do not have the advantage of obtaining the necessary moisture from the atmosphere such as outside floors and sidewalks on which the dew falls at night, if not properly protected and kept damp, become prematurely dry and are therefore more or less porous and weak, causing easy abrasion under foot traffic, or what is commonly known as dusting.

Care should be exercised in keeping such floors damp by covering with wet sand, wet hay, or straw, for a week or more until the floor has properly hardened. If this has not been done and the floors are found to dust under foot traffic, the following remedy will be found very easy to accomplish, economical, and effective.

Wash the floor thoroughly with clean water, scrubbing with a stiff broom or scrubbing brush, removing all dirt and loose particles. Allow the surface to dry; as soon as dry apply a solution of one part water glass (sodium silicate) of 40° Baume, and 3 to 4 parts of water, the proportion of water depending upon the porosity of the concrete. The denser the concrete the weaker the solution required. Stir well, and apply this mixture with a brush (a large whitewash brush with a long handle will be found the most economical). Do not mix a greater quantity than you can use in an hour.

If this solution is sufficiently thin, it will penetrate the pores of the concrete. Allow the concrete surface thus treated to dry. As soon as dry, wash off with clean water using a mop. Again allow surface to dry and apply the solution as before. Allow to dry and again wash off with clean water, using a mop. As soon as the surface is again dry, apply the solution as before. If the third coat does not flush to the surface apply another coat as above.

The sodium silicate which remains on the surface, not having come in contact with the other alkalies in the concrete, is readily soluble in water and can therefore be easily washed off, thus evening up the color and texture of the floor. That which has penetrated into the pores, having come in contact with the other alkalies in the concrete, has formed into an insoluble and very hard material, hardening the surface, preventing dusting and adding materially to the wearing value of the floor.

DISCOVERY OF COAL IN BRAZIL

Quite recently, within the state of Pernambuco, was discovered what promises to be a most valuable coal field. The area embraces about 22 square leagues, about 300 meters (1 meter = 3.28 feet) above the level of the sea and, according to the opinion of experts, has every appearance of true Carboniferous strata.

The first vein was encountered at a depth of 20 meters underlying a formation of clay mixed with sand and coal fragments impregnated with sulphides. While the analysis shows the first vein exposed to be of only fair quality, its richness increases as the shaft deepens. The following published analysis is from the expert's report:

Moisture	1.900
Volatile matter	18.815
Ash	20.520
Carbon	58.733
Loss	.032
	100.000

Notwithstanding that this coal area is quite small in comparison with many in the United States and other countries, nevertheless this discovery will act as a strong incentive toward further exploration in Brazil. As this particular field is located so that the coal can be advantageously mined and transported, it will prove of great value in furnishing cheaper fuel to the existing factories and manufacturing concerns, as well as to those contemplated within the states of Pernambuco, Alagoas, Sergipe, and Bahia.—*United States Consular Report.*

LIABILITY FOR INDUSTRIAL ACCIDENTS

*By Sion B. Smith, Esq., Pittsburg, Pa.**

Different Methods by Which the Losses From Personal Accidents May Be Equitably Distributed

The whole country stands aghast when a disaster in a coal mine wipes out 200 lives, yet it is a lamentable fact that the Juggernaut of industry takes toll of 150 lives every day and merely brings forth sociological discussions in literary magazines. Has it occurred to any one that we have a very considerable body of natural resources in the shape of 150 human beings a day who are not only subtracted from the productive efficiency of society, but whose going leaves added burden on society. A vast army of over half a million every year, which to the unbiased onlooker seems sorely in need of conservation as a natural resource of great value to society, is not easily placed. No amount of legislation or inspection, or safety devices will prevent there being accidents, and consequently uncompensated loss to society. The loss must be paid for in some way. In the very nature of things it must fall on the employer, the employe, the families of the employe, or the general public.

The law of the land makes division of this loss as between the employer and employe; the law of necessity performs the same function as between the family and the public. All of the load that cannot be put on the employer falls on the employe and his family. If they have no means, the public must carry the load through our poor houses and charitable organizations. Each one in the line from the public back to the employer tries to shift the burden to the next fellow, but so far the employer seems to have been the most successful in avoiding responsibility. The poor workman and his family gets it both going and coming; the employer will pay him no damages that he can avoid and the public will not take care of him until he begins to starve.

But you may ask, has the employe no rights at all? He has the best right in the world against the fellow servant whose negligence causes him injury. If that fellow servant happens to be a "Hunky," with no property, that is his misfortune; but it does not impair the injured man's right in the least. He has a beautiful right that is just as good as new, not worn a bit, though absolutely worthless.

There are many fire-escapes to save the employer from being burned in a damage suit, as most of you know from observation at least.

It is a matter of common knowledge that recoveries are had in court but for a very small per cent. of the accidents that occur. The injury or death of a workman is as certainly a loss to society as the burning of a machine is to its owner. Who stands the loss? If the workman can be made to carry the loss, it is on him. If he has no accumulations with which to carry it, then the proverbial impossibility of getting wool off a toad forces the load on to the next pair of shoulders, the public, otherwise known as society. The poor fellow must live and be supported, or die and be buried. Both cost money, and he has a family that cannot afford either.

You and I do not realize that the injury or death of a workman is any loss to us, until we see him and his family as wards of the county or subjects of charity. Then we see the money go and set up a howl.

That howl is resulting in more or less general enactment of two classes of laws, designed to throw the load back on the employer; first, safety appliances and inspection laws; and second, laws imposing liability either directly or indirectly upon the employer for injuries to and resulting death of employes. Laws of the first class either contain a specified penalty or liability for their violation, or the mere fact of the violation is construed to be negligence which renders the employer liable. With this you are familiar.

* Read before the Coal Mining Institute of America at Uniontown, Pa., June, 1910.

The second class of laws gives a right of action against the employer to the survivor or dependents of a workman whose death results from an injury received in his employment, a right which did not exist at common law; and in some countries, though not as yet in the United States, a right to compensation from the mere happening of the accident, irrespective of the question of negligence.

This is not based on any principle of natural right or justice, for there is no natural justice in compelling one man to pay for the misdeed of another. But it is founded purely on necessity, the supreme necessity of the state to protect itself against having the maimed and crippled and their dependents become public charges, and to place that burden upon the one who can best sustain it. It is the same reason exactly, before the law, for marriage, i. e., to fix responsibility for the support of offspring.

All attempts in this country to make the employer directly liable for injuries to his employe, irrespective of the question of negligence, have failed, generally, through being held unconstitutional by the courts. The Act of Congress of 1900 is a shining example. One of the high officials of this Institute encountered the same stumbling block some years ago before the Pennsylvania legislature.

Under a government like ours it seems impossible to get over—and pretty hard to get around—the Constitution in matters like this. But in monarchial governments they can and have gone to the root of the matter, and in a score or more foreign countries laws are in force providing compensation for industrial accidents.

The earliest form of this compensation was by mutual insurance. Insurance against industrial accidents can be traced back to the old trade guilds. It existed before 1600 in the Hartz Mountains in Germany. The first attempt to administer it through governmental agency was in Germany in 1861. Ten years later an employers' liability law was enacted, and in 1883 compulsory sick and accident insurace was added.

The system has been adopted in some form in 22 foreign countries. In some instances the compensation is paid by the government; in others by the government in part, and in part by the employers; in others by the employers alone; and in still others by the employers and employes in varying proportions. The administration of the indemnity funds is by insurance departments of the government, by mutual companies under governmental supervision, and sometimes by a board in which the employes have representation. The funds are raised by mutual contribution, by special taxes on the employers collected as other taxes, or in some instances from general taxation.

In a majority of the cases the benefits of the compulsory insurance are limited to workmen whose earnings do not exceed a certain amount per year, ranging from 720 marks ($138.96), in Finland, to £300 ($1,459.95), in Great Britain, and £400 ($1,946.60), in Queensland, the average being about $500.

The amount of indemnity paid is usually based on the amount of the workman's earnings for the preceding 1, 2, or 3 years. For partial disability they are usually paid hospital and surgical expenses and a percentage of their weekly wage for a limited length of time. For total disability the indemnity ranges from 50 to 100 per cent. of the actual or estimated wage of the preceding 1, 2, 3, or 6 years. In case the injury results in death the family or dependents receive from 40 to 60 per cent. of the earnings, with provision that the amounts paid shall in no instance be less than a certain minimum nor more than a certain maximum. In Sweden and New South Wales the indemnity is placed at a fixed sum for all, regardless of the previous earning capacity. And in substantially all countries a fixed funeral expense is allowed.

There are, of course, certain healthy limitations imposed upon both parties. An employe who wilfully injures himself cannot recover compensation, and where the injury is the result of his drunkenness the compensation is reduced one-half.

Where the injury results from the wilful act of gross negligence of the employer he loses the benefit of these acts, and where it results from violation of protective measures of the law the liability is increased.

In some countries the employer is exempt from all civil liability outside of the fixed legal compensation, except where the injury occurred through his intentional act; in others the common law liability remains, and the employe has the choice of which right he will pursue, the common-law right subject to the uncertainties of a law suit, or the statutory right, which is sure but limited.

Any agreement by the workman, as for example an agreement to release the employer from responsibility in case of accident, is absolutely void. And any attempt by the employer to load the cost of the insurance on the laborer, either directly or indirectly, is in some instances made a criminal offense.

In one instance the shyster lawyer got a black eye when the law fixed the fees that might be charged and invalidated any agreement to pay more. Another clog on crookedness is a provision that in case the employer goes into insolvency the indemnities that he may be bound to pay, for example, in the form of weekly payments, are capitalized and made a lien on the bankrupt estate.

Usually, and properly, the employer who has to pay an indemnity is subrogated to the rights of the injured employe as against third parties, and thereby he may sometimes be able to put the load on the one really to blame. In some instances where the workman gets a benefit from a society in which the employer pays a part or all of the dues, the indemnity to be paid by the employer is reduced proportionally.

That this system works, is proved by the extent to which it has been adopted. And their statistics show that the number of accidents has actually been decreased by the adoption of the system of universal compensation for injury, and also that the actual amount paid out is not as great in proportion as the cost of accidents in the United States.

In the United States there has been but little accomplished in this direction until within the last 5 years, and mostly within 2 years. The "fellow servant" defense has been modified or abolished in 35 states, counting the District of Columbia, the Philippines, and Porto Rico as states. This modification extends only to railroads and common carriers in 23 of those states. And the doctrine of "assumed risk" has been modified in nine states along the line of saving the right of a workman to recover who reports a defect and goes back to work before it is remedied.

A peculiar development of the last 2 years has been the introduction of a doctrine of comparative negligence in 14 states, whereby the amount recoverable is proportioned to the degree of culpability of the respective parties.

State supervision seems to have reached the limit in Illinois, where a law was passed last year requiring even the common miner to pass an examination before a state board and obtain a certificate before he will be allowed to work in a mine.

In Pennsylvania both legislative enactment and judicial interpretation have stood steadfast against any lightening of the perils of labor or increasing of the burdens of production. But a few of the states have at various times attempted to obtain compensation for all workmen who are injured in the course of their employment. A number of years ago Maryland passed a compulsory industrial accident insurance law, but before it drew its first long breath it was declared unconstitutional by the courts. Early in the present year a somewhat similar bill was introduced, which provides for indemnity for all industrial accidents from a fund provided by mutual compulsory contributions from employers and employes.

New Jersey attempted to reach the question indirectly by authorizing insurance companies to make special rates for associations of employes of one employer.

Two states, one in the East and one in the West, have been notable in the past year for legislation along this line, Massachusetts and Montana. Under the Massachusetts act either the employer or 10 per cent. or more of his employes may submit to the Board of Conciliation and Arbitration a plan of compensation for injury received in the course of employment, without reference to legal liability, based upon a percentage of average earnings, and, when approved, the employer and employe may contract that the benefits provided by this plan shall be in lieu of all legal right of action. This is purely voluntary.

In Montana, the act requires all workmen, laborers, and employes employed in and around any coal mines, etc., to be insured in accordance with the provisions of this act, against accident occurring in the course of their occupation. To pay this a fund is raised, under the control of the auditor of the state, from a tax of 1 cent a ton on all coal mined, and 1 per cent. of the gross wages of the employes, to be deducted by the employer and paid over. The amount of compensation provided is $3,000 to the family or dependents in case of death, and not to exceed $1 per working day for permanent disability. Acceptance of benefits under this act bars action for damages, and the commencement of an action forfeits all claim to benefits.

A particularly encouraging development of the past few months is a voluntary indemnity plan adopted by the United States Steel Corporation and the International Harvester Co., which disregards all question of legal liability, does away with "contributory negligence," "assumed risk," and the "fellow servant" doctrine, and compensates every employe injured while at work.

Under the corporation plan the relief is greater for married men than for single, and increases according to the number of children and length of service. During temporary disablement single men receive 35 per cent. of their wages, and married men 50 per cent., with an additional 5 per cent. for each child under 16, and 2 per cent. for each year of service above 5. For permanent injury a lump payment is made, based upon the extent to which the injury interferes with employment, and also upon the annual earnings. In case of death the widow and children receive 1½ years wages, with an additional 10 per cent. for each child under 16 and 3 per cent. for each year of service above 5.

The Harvester company's plan calls for a payment equal to 3 years average wages, in no event less than $1,500 or more than $4,000. For loss of a hand or foot, 1½ years wages, not less than $500 or more than $3,000. For loss of both hands or feet, or one hand and one foot, 4 years wages, but in no event less than $2,000. For temporary disability, one-fourth wages for 30 days, one-half wages beyond that for 2 years, and a pension if the disability continues total.

If a few individual corporations do this through a dawning consciousness of the duty resting on the "brother's keeper," if one state can pass a law to accomplish the same result—which pray God the courts may sustain—if employers' accident insurance, the cost of which is based upon the pay roll, can make a profit out of this traffic in the blood of the workman, surely it is but a short step to a universal indemnity law. Every up-to-date employer carries employers' accident insurance. What possible difference can there be practically whether he does this voluntarily, at a profit to the insurance companies, or under the compulsion of the law and saves the profit?

To be sure, the insurance company may fight the poor devil who is injured to the last ditch of litigation, and pay as few losses as possible; but statistics in foreign countries show that even that process costs more in this country than their system of universal compensation at a fixed moderate rate.

It would seem to be more than a hair-brained phantasy that we can subserve the ends of justice and equity toward the injured, the protection of society from possible charges upon it, and do this with less derangement of existing con-

ditions and more effectively, by a system of universal indemnity at a definite rate. Whether this be accomplished by government indemnity, by compulsory employers' liability, by mutual insurance, or in any other way, is a purely incidental matter. The main point is the certainty of compensation and the consequent security and protection not only to the injured, but to society. It may be ideal; it certainly is not an unapproachable

Fig. 1. Corner of Mine Room, Jeffrey Exhibit

ideal; and it seems to be an ideal of which social necessity demands the speedy realization.

In order that you may know just how the law distributes this particular portion of the cost of production represented by the lives and limbs of the employes, it may be well to state in general terms and in a very untechnical way the legal liability of employers for injuries to their employes.

The English common law lies at the foundation of our doctrine of employer's liability, but its principles have been so differently interpreted by the courts and modified by the legislatures that there is nothing approaching unity in the law as it stands today.

In the earlier days the whole risk of the employment was deemed to be assumed by the employe. You say at once, that is an unfair imposition. Is it? Last week the lightning struck your chimney and knocked the top off. Before you went to work you hired two men to top it out again. While you were in the mine the mason carelessly let a brick slip and it hit the hod carrier and broke his leg. Are you to be obliged to pay the hod carrier for that carelessness of the mason? But you quit work at night and started up the shaft, and the engineer carelessly lost control of the hoist and allowed the cage to drop, and you got your leg broken, and you were mighty glad it was not your head. Is there any more reason why the company should pay you for your broken leg? To the average intelligence it would seem just that the one whose negligence caused the injury should be held responsible for it. Therefore the earlier interpretations of the law imposed all the risks of the trade upon the employe.

But gradually there developed an idea that the employer owed certain duties of protection to the employe, and the original law has been modified to the extent of requiring the employer to afford such protection to the employe as a working place, tools, appliances, materials, etc., that are reasonably safe for the intended use and reasonably well adapted to perform the work in contemplation.

And he must use the same degree of care in the maintenance and repair of those appliances, and therefore he must make inspection adequate to discover the need of repairs or the lack of safety conditions.

In the same way he must use that care in the selection of his employes that he bestows on the selection and maintenance of the inanimate and irresponsible instrumentalities of the business; and the employment of workmen of improper age, habits, morals, or practices may charge him with liability for accident resulting through them.

It is also his duty to provide and enforce proper and adequate rules for the conduct of the work, and to give inexperienced workmen instruction and warning as to the hazard of their employment.

When a man enters into a contract of employment, the law imports into that agreement the assumption by the employe of the risks ordinarily incident to the employment, and such other risks as are known to and appreciated by him. This is on the theory that if he is competent to do the work, he must understand the conditions under which it is carried on and therefore takes the chances and is bound by the result.

MINING INDUSTRIAL EXPOSITIONS

Written for Mines and Minerals

Permanent and portable industrial expositions are so common they receive little attention except in the towns where they are held. Within the past year there have been several expositions in which the displays of mining machinery and apparatus attracted widespread interest. Meyers & Whaley introduced their coal-loading machine for the first time at the Appalachian Exposition, and the Jeffrey Mfg. Co. their Armorplate locomotive at the Western Pennsylvania Mining Exposition. It would appear as if manufacturers and others were inclined to use these expositions as "coming-out" functions in mining-machine society. The brain work and money expended in arranging the reception rooms in which the functions are held appeals undoubtedly to the decorator, and the display is a matter of pride for the exhibitor, who expends money lavishly that his product may be well furbished.

In the Western Pennsylvania Mining Exposition the Jeffrey Mfg. Co. placed their mining machinery in an artificial coal mine which had a built-up coal face 45 feet long and 6 feet high, and altogether their exhibit occupied a floor space 100×30. The coal face shown in Fig. 1 is masonized to represent as close as it was possible an artificial rib constructed from the Hazel Mine coal of the Pittsburg-Buffalo Coal Co. In this the shortwall electric coal cutter, in charge of a demonstrator, made an undercut. At the left of this machine was a self-propelling steel truck equipped with an automatic cable reel that winds up the electric cable as the truck moves out of the room. This coal cutter was unloaded from the truck by its own power, made the sumping cut at the right-hand rib and a cut across the face until the left-hand rib was reached. A steel feed-cable,

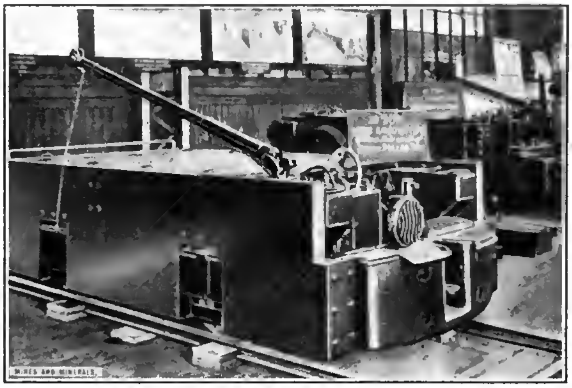


Fig. 2. Armor Plate Locomotive, Jeffrey Exhibit

wound on a power-driven drum at the front end, but not shown, pulled the machine across the face of the coal at a speed regulated by the hardness of the coal in which the cut was made. Another cable which had no connection with the power served as a guide to hold the machine to its work at a proper angle for its greatest cutting efficiency.

When the machine finished the cut it was loaded on to the truck with its own power. The narrow cutter head and short distance from the machine to the coal face appealed mostly to those who are obliged to undercut coal pillars that have a tendency to come down when undercut by hand. In order to show the regular electric chain mining machine, that also was set up as shown in Fig. 3. The flame and gas-proof motor and starting box on this machine was new to some and interested those whose mines generated small quantities of gas. To the left of this coal cutter was an electric auger drill, and to the right an air-driven auger drill attached to a small motor-driven air compressor by a hose. The coal cutter and electric auger were explained and operated by the demonstrator.

Another interesting exhibit in this section was the 13-ton electric mine locomotive, with its frame made of armor-plate

Fig. 3. Chain Mining Machine at Work, Jeffrey Exhibit

steel, and its sides and ends made of rolled steel, the object being to strengthen the machine so as to withstand the heaviest kind of service.

Two other Jeffrey locomotives shown at the exhibit were built with structural-steel frames and equipped with motor capacity sufficient to slip the wheels and to stand continuous mine service. Each of these locomotives had mounted on the front end an independent motor-driven crab, having a drum to hold 300 feet of $\frac{3}{8}$-inch flexible steel rope. Daily demonstrations were made with this crab device which hauled pressed steel cars from one end of the room to the other end, to show how loaded mine cars are pulled from the face of the mine room while the locomotive remains in the entry.

On the wall back and above the exhibit were a number of photographic reproductions showing the Jeffrey machines and locomotives in various situations in large mines all over the country.

UNITED MINE WORKERS' EXPENSE ACCOUNT

The cost of strikes met by the international organization of the United Mine Workers in the past 11 years is stated as follows:

Year 1900	$ 144,462.50
Year 1901	202,202.71
Year 1902	1,834,506.53
Year 1903	301,922.44
Year 1904	1,065,435.47
Year 1905	753,626.02
Year 1906	805,599.92
Year 1907	105,045.57
Year 1908	744,897.19
Year 1909	600,267.39
Year 1910	1,532,020.42
Total	$8,089,986.16

A statement by districts shows that the enormous sum of $1,286,031.32 has been spent in the unsuccessful effort to organize in Alabama, $968,775 in Nova Scotia, $707,676.45 in Colorado, and $669,938.81 in West Virginia.

"To the $8,000,000 spent by the international organization," adds Mr. Lewis, "must be added the hundreds of thousands of dollars contributed by our members to the districts, and spent for the same purpose. If you will examine these two statements, you will discover that where we have spent the most money, in support of strikes, is where we have the least membership."

BAVARIAN GRAPHITE INDUSTRY

By George Nicolas Ifft, Nuremberg

The incorporation at Munich of an association for the development of Bavarian graphite deposits calls attention to an industry which for several centuries has been carried on in a remote corner of Germany, and according to methods as primitive today as they were 300 years ago.

In the Bavarian forest northeast of Passau, are found deposits of graphite, which is especially adapted to the manufacture of crucibles. In this respect, only the Ceylon graphite and certain graphite found in the United States, it is claimed, is equal to the Bavarian product.

That graphite deposits, presumably so valuable, have heretofore played so unimportant a part in the industrial life of the country is largely due to the fact that the mineral is the property of the owners of the land upon which the deposits are found. The graphite deposits are thus owned by innumerable small farmers, who mine it according to individual ideas and in the crudest manner, through shallow surface diggings, and mostly only in the winter months as a side issue to farming, forestry, or laboring. The mining so far has been little more than grubbing about on the surface, the ore being found in pockets, when a cellar or a shallow excavation is made by the farmer who happens to own the land. No one has ever gone to any depth for the ore or tried to mine it systematically. The ore is found in an out-of-the-way and primitive district without railway communication. A branch railway from Passau into this section, however, is now in course of construction, and already a number of the land owners at Pfaffenreuth have pooled issues and are preparing to inaugurate graphite mining in a more systematic manner, if still on small scale. How trifling has been the work in the past is illustrated by the total production of graphite ores from this district during the past years; as follows:

Year	Tons	Value	Year	Tons	Value
1905	4,921	$49,703	1908	4,844	$58,907
1906	4,055	46,092	1909	6,774	63,272
1907	4,033	47,921			

This seems to show a steady increase in production, but the average is not above that of the preceding five years. It represents work on 50 to 70 farms and employment for part of each year of 200 to 300 men. The ores run 10 to 25 per cent. pure graphite, averaging about 15 per cent. It is expected that modern methods of refining and handling will increase this average materially.

The new company which has just been organized will make the first attempt at systematic mining in this district. For the beginning it has secured control of about 750 acres of graphite land in the so-called Rampers, Haunersdorf, and Taubinger districts. It has also options running until 1917 upon about 1,500 acres more in the same and neighboring districts. The company estimates a minimum production for the first year of 13,500 tons of graphite ores. It will begin work with a force of about 450 men.

During the year 1909 Germany imported 29,191 tons of refined graphite, 9,080 tons of which were from Ceylon and 14,512 tons, a much inferior product, from Austria-Hungary. The new company hopes to be able to supply in a large measure the German demand for graphite for crucible purposes and at the same time develop a considerable industry and a new source of wealth in a remote and sparsely settled section of the Bavarian forests.

DATA OF PETRODYNAMICS

By R. Dawson Norris Hall, Consulting Engineer, Du Bois, Pa.

Of all the overlying coverings of mineral bodies, the weakest are those consisting of sand, loam, or clay, which often have practically no cohesion.

Flow of Loose Measures. Contraction of Volume of Clay After Settling in Water and Drying

Sand has its greatest strength when moderately wet; when dry, it runs freely if it be piled at an angle exceeding 30 degrees; when very wet, it is a "quicksand," the word well expressing its liveliness and fluidity.

Loam and clay on the other hand, become cemented on drying and stand at a steeper angle, depending on the relative quantities of their constituents and the strength of the binding cements.

While either open-cut or forepoling methods must be used in mining the minerals under such incoherent materials when they are found alone, it usually happens that a thin layer of rock will be found as the main covering of the mineral; but if this rock is badly creviced or becomes sheared in any way, the loose material, which rested upon it, will flow in and almost completely fill the mine workings, especially if there be water present, either in the loose material or supplied by some run or by an underground water channel.

These bodies of quicksand and bog are of recent formation, the remains of lakes, glaciers, and buried rivers. The water drains into such deposits, and where the geologic formation is a basin does not drain away from the loose material. Often no cements have made their appearance to create rocks out of loose shifting beds, or in the presence of so much water these fail to react or harden in such manner as to bind the measures they impregnate.

Such deposits may be very thick. The glacial drift in the states of Ohio and Michigan reaches in places a thickness of 300 feet. The Wyoming buried valley has deposits of incoherent sand 200 feet deep and the weight of this body of loose sand was the probable cause of the Pittston disaster. The flow of ice in the glacial period dammed back many of the streams, which till that time flowed northward, and later reversed their direction. Before this reversal, naturally, large bodies of water were impounded, in which were deposited thick beds of sand and slime, which are often not well cemented and compacted to this day. If such deposits are self-draining, they are well compacted; if not, they are still in a somewhat fluid condition.

The flow of quicksand, owing to the strength of the immediate covering, may not take place till after the final caving of the workings takes place. An instance has been reported to the writer, where a mine was worked up to and below the surface of a town lot. The pillars were withdrawn. No damage was done to the adjoining lot, which had not been undermined, but strange to say, a third lot, beyond the second, which also was not undermined, was seriously affected. A loose sand, on this lot, washed down into the cave under the property which had been undermined, passing beneath the stable intermediate lot without disturbing its surface.

How readily a quaking bog can move is well illustrated by the moving peat bogs of Ireland, which occasionally break loose from their moorings, engulfing houses and property. Wherever there are quicksands or bogs, there is a risk that the mining on one property will cause a sinkage on another neighboring to it, and the damage may be quite widespread. These quicksands and bogs are often covered by ground, which, being well drained, affords a fair foundation and good farming land. Moreover the sands and bogs may not be very "quick" until a cave permits the water to wash them down into the cracks and cavings beneath. It would appear that shafts sunk through quicksands or quaking bogs should have the protection of larger pillars than are provided for shafts in solid material. For whenever mining is completed around the shaft pillar and caving takes place, the flow of sand or bog toward that caving may threaten the stability of shafts, head-frame, and mine buildings. The same security might be insured by only mining out the mineral in part near a small shaft pillar as would be obtained by leaving a very much larger shaft pillar, so that a removal of silt around the shaft could not occur. The shaft workings will ultimately drain the water from the overlying silts of the region, and after that is thoroughly done the danger to the mine property will be reduced, as the dried silts are less likely to be engulfed.

But even if not one cubic foot of sand, clay, or loam does escape from its due place in the measures, yet nevertheless there may be such a loss of water as to result in a surface settlement. It is said that the cause of the weakness of the foundations of St. Paul's Cathedral in London, England, is to be found in the shrinkage of the London clay as a result of the building of the underground railroads, a work which has abstracted much water from that measure. The London clay, laid down immediately after the Cretaceous period, is in London only 64 feet thick, though in the vicinity it reaches 500 feet. It is a well recognized fact, that the lowering of ground water makes old foundations insecure, as the supports shrink from under them when the water is removed. Similarly when sand is drained, it furnishes a better foundation, than when wet, for any structure put on it after draining.

It might be well to examine the investigations of Seelheim, King, and Slichter, in this connection.

Seelheim[5], in an article on the methods of determining the permeability of bodies, that is, their power of permitting the transference of liquids and gases through their pores, has shown that when a mixture of very fine clay and water is allowed to stand quietly for some time, under conditions where no jarring can take place, the clay settles in layers, but contains a large amount of water. He found that, where no jarring took place, the upper layers contained more water than the lower ones; the proportion being one volume of clay to 3.84 volumes of water in the upper strata and one volume of clay to 1.78 volumes of water in the lower strata. That is to say in the loosest settling 79.34 per cent. of the volume of sediment was water and in the closer packing, there was still 64.03 per cent. of pore space.

Where the settling was caused to take place under frequent jarrings, Seelheim obtained a uniform texture throughout and a greater compactness but there was still a pore space of 54.54 per cent. He further showed that there was no sensible reduction of pore space when the sedimentation was caused to take place under the pressure of 102 feet of water instead of a few feet.

Prof. F. H. King[7], on behalf of the United States Geological Survey, has investigated the proportion of pores to clay in bodies of fine dry clay dropped into place in small lots, tamped and jarred, an operation which results, he has found, in greater density of clay than is obtained from simply pouring in mass, shaking and striking off the surface.

Prof. Chas. S. Slichter[8], also of the Geological Survey, has calculated the pore space resulting from placing rigid and true spheres of uniform size in contact with each other at various angles. Where true and equal spheres are placed immediately over one another in similar manner to the spots on a six-spot playing card but in close order, then the pore space is 47.64 per cent. of the whole and it will be noted that Professor King has found this to be the average space left in dry clay dust, tamped and jarred into place as above described. But it is not likely that we have here the limit of clay density from compaction. Sandstones very readily shake down to a porosity of about 36 per cent. and some even lower, even below the porosity given by Professor Slichter as the result of placing equal uniform spheres, set like the spots on an eight-spot playing card, only in close formation and with 12 points of contact between spheres. The lowest percentage of porosity found by Slichter, obtained as above set forth, was 25.95. Professor King obtained practical

porosities of 25.55 and 25.43 per cent. Here doubtless smaller spheres partly filled the pores between larger spheres.

It is not unlikely that under pressure clay similarly receives a further settling, and, moreover, a degree of distortion may well take place fitting the spheroidal or cubical units to those around them, which distortion after enduring for a long time may become permanent. For a tombstone, resting for years on a head and a foot-stone will bend and become set in its acquired shape so that the curvature is only slowly removed when the strain on the stone is reversed by turning it over. In final analysis, we find that clay only absorbs 8 to 15 per cent. of its bulk of water and passes little or none.

Clearly such a pore is not open or it would allow water to pass, and the clay also is far denser than any theory of elementary units of the nature of spheres and in mere point contact would indicate. True, part of the loss of pore space is due to coatings of these elementary units, the action of cementation and filling tending to continue till it has progressed so far that further penetration of water through the clay becomes impossible. It is therefore necessary to warn the reader of these remarks that the last line in the accompanying Table 1 would only furnish a valuable guide to the possible shrinkage from drainage and pressure alone, if the whole of the loss of porosity in clay could be ascribed to those causes.

Table 1. Showing Volumetric Shrinkages of Clay Slimes

Nature of Material	Clay : Water or Clay Pores	Water Space or Pore Space Per Cent.	Comparative Volume	Authority
Upper layer of subsided clay in water........	1 : 3.840	79.34	100.00	Seelheim
Lower layer of subsided clay in water........	1 : 1.780	64.03	57.43	Seelheim
Same after frequent jarrings............	1 : 1.200	54.54	45.45	Seelheim
Fine dry clays.......	1 : 11.000 to 1 : .818	50.00 to 45.00	41.32 to 37.56	King King
Perfect spheres of equal diameter closest arrangement...... ...	1 : .354	25.95	27.97	Slichter
Stiff clay..........	1 : .177 to 1 : .087	15.00 to 8.00	23.76 to 22.31	Fanning Fanning

It should be noted that the decrease in volume in a slime, especially when under pressure of overlying measures or other slimes, affects only the depth, unless the basin is small and has shelving sides. For the area of the stratum remains constant, only the depth varies for the weight will spread the slimes as fast as they shrink, so that the whole stratum remains continuous. The vertical shrinkage is equal to the shrinkage in gross volume.

A clay slime, 200 feet thick, will reduce to 50 feet and less, as a result of drainage, and though such a result is rare, for most bogs are more concentrated than Seelheim's upper slime, yet the figures above suggest what an action drainage has in shrinking roof coverings of mines and how even clays of great age may lose bulk by mining operations and let down the rock or surface with its buildings above them.

German and English investigations[10] have been made of the shrinkage in air of flint clays. A flint clay drying in air will shrink in all directions 5 per cent., so that it will measure linearly only 95 per cent. as much as before shrinkage. Its volume will be only as the cube of 95 or 85.74 per cent. The loss in drying is therefore 14.26 per cent., and this, if the clay were plastic, so as to give laterally with freedom, would reduce the thickness of the bed 14 feet 3 inches in every 100 feet of depth of measure.

A measurement of the iron ore deposits in what was once the channel of a running stream in a coal mine or of the sun cracks in a bog or road puddle will immediately convince us that the linear contraction of the deposit, bog, or puddle, from a condition, not of fluidity, but of incipient cohesion, to a state of partial dryness is not less than $\frac{1}{15}$ or 6.5 per cent. The volumetric contraction will therefore be not less than 18.25 per cent. Such a reduction in roof or floor of a mine would powerfully modify conditions.

AUTHORITIES

[1]Edmond Orton, State Geologist, Ohio, Encyc. Brit., Vol. XVII, page 734.
[2]C. K. Adams, Encyc. Brit., Vol. XVI, page 238.
[3]H. B. Wheatley, Encyc. Brit., Vol. XVI, page 279.
Horace B. Woodward, Geology England and Wales, page 435.
[4]Ira O. Baker, C. E., Treatise on Masonry Construction, pages 195-196.
[5]Methoden zur Bestimmung der Durchlassigkeit des Bodens, Zeitschrift fur analytische Chemie, Vol. XIX, page 387.
[6]F. H. King, U. S. G. S., 19th Annual Report, 1897-1898, Part II, page 78.
[7]Ibid, page 708 *et seq.*
[8]Chas. S. Slichter, U. S. G. S., 19th Annual Report, 1897-1898, Part II, page 305, *et seq.*
[9]J. T. Fanning, Treatise on Hydraulic and Water Supply Engineering, page 103.
[10]Heinrich Ries, U. S. G. S., 19th Annual Report, 1897-1898, Part VI, continued, pages 404, 422, 434, 437, 441, and 446. Twelve samples of Kaolin with average air shrinkage of 5.5 per cent. and extremes 2.67 per cent. and 10.58 per cent.

CEMENT-LINED PUMPS AND COLUMN PIPES

Written for Mines and Mineral

Next to drilling and blasting the most important item of cost in mine operations is that of pumping water. The cost is increased by the acids and acid salts which are frequently found in the water in metal and coal mines. If sulphate of copper is in the water it will attack the pump and the pump-column pipes, replacing the iron of those pipes with what is termed cement copper, according to the reaction $CuSO_4 + Fe = FeSO_4 + Cu$. If sulphuric acid is in the water it will attack the iron of the column pipes and corrode them badly, according to the reaction $H_2SO_4 + Fe = FeSO_4 + 2H$.

In those mines where mineral salts are in solution it may be necessary to use column pipes of bronze which are not attacked by acid water so readily as iron pipes. The cost of these pipes, however, is high, and generally wherever possible wooden pipes or wooden linings are substituted. There are occasions when the cost of pumping water alone will be the most expensive item of mine operation, but this is one of the difficulties imposed by nature and can only be overcome by the use of drainage tunnels, which are not only expensive but would be impossible in many instances.

Where several coal mines are owned by the same company the cost of pumping may be reduced by having a central pumping station into which all the water is drained, and at these pumping stations large pumping plants are installed.* The evolution of mine pumps extends over a long period and has included almost every known means of raising water, from baling by hand and carrying the water in leather bags up a ladder, to more recent cement-lined electric-power pumps. Mine pumps, like every other class of machinery, should be made as simple as possible, owing to the fact that they are liable to be injured and do not have the same care that pumps on the surface would receive. They must also be constructed

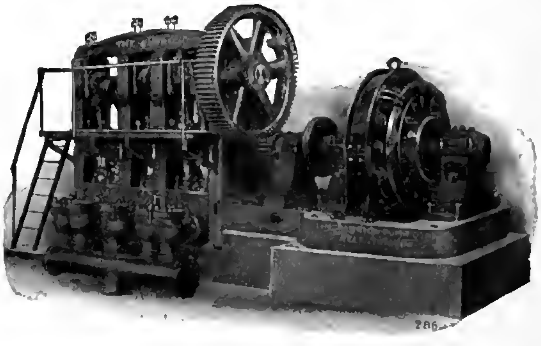


Fig. 1. Quintuplex Electric Pump

* Mines and Minerals, Vol. XXVI, page 205; *Ibid*, Vol. XXX, page 457

with a water end that will not corrode or be cut out by small particles of grit which are sure to pass through the valves in course of time. To prevent this latter trouble as much as possible there should be sand traps in the ditches leading to the sump which should be made large; and all chips and dirt, as far as possible, should be prevented from going into the sump; otherwise a snorer should be placed on the tail-pipe.

One of the latest power pumps for use in mines is shown

Fig. 2

in Fig. 1. This is a power pump driven by an electric motor. It has a number of features which add to its convenience and usefulness in mines, since its adoption eliminates steam-pipe lines in shafts or slopes leading to the pump which greatly affect the atmospherical conditions as well as the timbers and roof. It also eliminates the exhaust steam and the apparatus which is usually needed to condense the exhaust and which increases the temperature of the sump water. If the water is acidulous, an increase in temperature usually makes it more corrosive on pumps and column pipes. By the use of electricity and the electric pump with two or three wires the cost of pumping and the pump repairs has been reduced.

The Aldrich Pump Co., which specializes on electric mine pumps, has discovered a process of lining the water cylinder of its power pump so that water carrying as much as 75 grains of free sulphuric acid to the gallon seems to have little effect on the working barrels, where previous to its introduction a working barrel only lasted about 6 weeks.

The pump shown in Fig. 2 is the one under consideration and is working against a vertical height of 150 feet, pumping 1,000 gallons of water per minute. The pump shown in Fig. 1 lifts 1,000 gallons of water per minute 600 feet, high. It is installed in the coal mines of the Kingston Coal Co., Kingston, Pa. The water cylinders are cement lined and the plungers operated by a 200-horsepower motor. The Aldrich Pump Co. claims that its cement-lined pipes will give a service equal to that of bronze water pipes, and at the same time are very much cheaper to install.

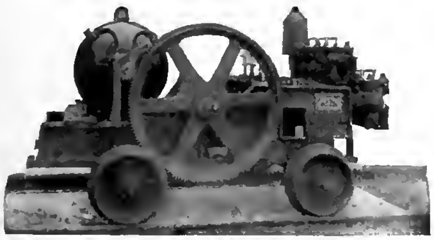


Fig. 3

In the anthracite region one company at least has had its triple-expansion wood-lined pump, that was manufactured by another company, relined with this cement. The cement lining can be applied to any type of water end without making special castings, and it can be applied to any surface whether spherical or not.

Fig. 3 shows a portable power pump whose water end is lined with cement. This pump will deliver 300 gallons per minute against 350 feet head, and is at work in the mines of the Consolidated Coal Co., Fairmont, W. Va.

Cement lining may be applied to column pipes and protect them from corrosion the same as the water cylinder, and it is claimed that it will give better service than wood lining in the same positions.

ELECTRIC CIRCUIT PROBLEMS

The application of electricity to mining operations is so varied and extensive that electric circuits form an important part of the equipment of collieries and some metal mines.

In mines, especially where electric haulage is used, the installation of a circuit costs more than the rest of the plant, and in addition is as expensive to keep up unless carefully watched. Fortunately the capacity and resistance to the circuit can be readily calculated; unfortunately the insulation can not, and becomes therefore a serious problem that can be solved only by the electrical worker on the spot. This is the more to be regretted, for the efficiency of the plant,

Fig. 1. Trolley-Wire Hanger in Roof

and in some cases the safety of miners, depends on the insulation. In Circular No. 23 of the United States Bureau of Standards and in the Report of the American Mining Congress Commission on the Standardization of Electrical Equipment in Mining, no mention whatever is made of trolley-wire hangers, which are about as important as any other part of an electrical haulage system. Wherever possible, the trolley wire must be placed to one side of the track, and the hangers may be fastened directly to the roof, to brackets, to overhead timbers, or in some other suitable way. In Fig. 1 is shown a Jeffrey electric locomotive pulling a coal train. In the illustration, the trolley wire is suspended from the roof by a combination of expansion bolt, hanger, and trolley clamp. One of the two methods of fastening the trolley wire to the roof is by means of a hole drilled in the rock in which is driven a wooden plug, and then by means of a screw the hanger is attached to the plug. While this may be slightly more economical, it is hardly so sure a support as the expansion bolt, and since safety and efficiency are more to be considered than slight economy in first cost, the latter is to be preferred. Both the screw and the expansion bolt terminate in a stud bolt to which the hanger shown in Fig. 2 is screwed. Apparently the hanger is a simple appliance and anything in that line would answer the purpose, when as a matter of fact it is the most important device entering into the trolley system of underground haulage. In the first place it is an insulator and in the second place it must support the trolley wire which has some weight

and considerable vibration. If these were the only considerations it would not be necessary to give the hanger such a complicated shape as that shown in Fig. 2. The Ohio Brass Co. has been making hangers a number of years and this is the outcome of their experience and not a mere guess. The skirt is constructed so that water will not trickle on to the wire to cause short circuiting; or if any does run over, it will on account of the double groove in the petticoat, fail to make connection. In some mines headroom is of great importance and this is minimized by the hanger being but 1½ inches in height. In high haulways this length adds strength to the connections. Inside the hanger the insulating material known as "Dirigo" cements the bolt in place and from the peculiar construction of the hanger cannot be worked out. Another feature is the hexagonal arrangement on the skirt which permits of quick attachment to the roof bolt and the use of a wrench. If this hanger is to be attached to a roof timber the mine hanger screw only is needed. In Fig. 3 is shown the hanger attached to the collar of a timber set. In such cases either the hanger shown in Fig. 2 or that in Fig. 4 may be used; in either case the fastening requires less careful attention and workmanship than when attached to the roof rock. Both lugs in Fig. 4 project far enough so that the skirt does not interfere with the 2½-inch lag screws by which it is fastened to the mine timbers. To the stud bolt pendant from these hangers the trolley clamp is attached. The galvanized malleable iron clamp, shown in Fig. 5, known as the Modoc trolley wire clamp, consists of a suspension casting,

Fig. 2. Screw Trolley-Wire Hanger

Fig. 3. Hanger Attached to Timber

a conical sleeve provided with a hexagonal nut for the application of the wrench when installing, and two interlocking jaws. All parts are held together by two lugs on one of the jaws which engage projections on the other. These jaws are intended to hold the parts together before installation and are not under strain when the clamp is in service. The height of the clamp is 2 inches, making it desirable for use in low-bed mines, while the length of the jaws being 3½ inches, a firm grip is had on the trolley wire. This clamp is also the product of the Ohio Brass Co., of Canton, Ohio, and has been designed by experienced men with a view to overcoming obstacles met with in practice.

The jaw castings are attached to the suspension casting by a flange at its lowest point and by revolving the conical operating sleeve, which is mounted upon it, the suspension casting is screwed upon the hanger stud bolt and draws the jaw castings over the conical portion of the operating sleeve which, in turn, causes a spreading action on the upper faces of the jaws and exerts a very powerful leverage or gripping upon the trolley wire.

It is possible to attach the clamp to the trolley wire just tight enough to hold the wire in position until it has been stretched.

The clamps can then be tightened permanently upon the wire and in perfect alinement with it, thus eliminating backing off the clamps to line them up, which is often necessary with other styles of clamps.

Fig. 4. Lugged Mine-Wire Hanger

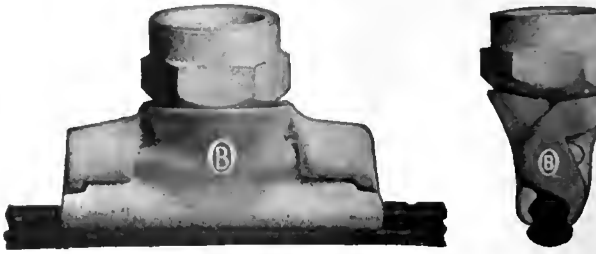


Fig. 5. Modoc Trolley-Wire Clamp

The clamp jaws are so narrow there is ample clearance for the flanges of the trolley wheel.

TRADE NOTICES

The Westfalia Engineering Co., of 42 Broadway, New York City, will keep in stock what is known as the Westfalia Rescue Apparatus. This agency is the only one authorized to give quotations, fill orders and supply duplicate parts.

Catalog No. 26 of the Jeffrey Mfg. Co., Columbus, Ohio, is on fans for ventilation of mines. It is worth owning.

The Sullivan Machinery Co., of Chicago, Ill., have opened a branch office at Spokane, Wash. Stocks of Sullivan machinery will be carried as in Seattle, Wash., and Nelson, B. C.

The Sanford-Day Iron Works, of Knoxville, Tenn., have manufactured a large double sheave drum that is said to be the largest in Tennessee and can be used on an incline 5,000 feet long. The main shaft is 7 inches in diameter, with 21-inch bearing on each side.

The New River Collieries Co., a subsidiary of the American Smelting and Refining Co., has placed orders for three coal tipples for their No. 1, No. 5, and No. 6 mines, at Eccles, W. Va. Each equipment is complete, consisting of "Link-Belt" dumping hoppers, shaking screens, picking bands, chutes, etc., and are excellent types of modern West Virginia tipple practice. The machinery—electrically driven—will be designed, constructed, and erected by the Link-Belt Co.

ANSWERS TO EXAMINATION QUESTIONS

Written for Mines and Minerals, by J. T. Beard

NOTE.—By special request of a number of subscribers the remaining questions of the last examination in the bituminous regions of Pennsylvania are answered at this time. The mine-foremen, second-grade questions of this examination were answered in MINES AND MINERALS in July, 1910, but owing to lack of space the first-grade questions and the fire bosses' examination were omitted.

Pennsylvania Bituminous Mine Foremen's and Fire Bosses' Examination Held April 5-8, 1910

MINE FOREMEN'S EXAMINATION

(*Second-Grade Questions.—Continued*)

QUES. 22.—What method would you adopt in drawing pillars in order to insure the greatest safety to the men and the most complete extraction of the coal?

ANS.—Much will depend on the kind of roof, floor, and coal; the thickness, depth, and inclination of the seam; and other conditions affecting the work. Pillar work should be done by experienced miners; the work should be kept in line, and places not allowed to stand idle; the roof should be carefully examined for fault lines and slips each day, and necessary timber be used to make the place safe. All posts should be drawn regularly as the work progresses and no timber be left standing in the waste.

QUES. 23.—Describe the kind of explosives best adapted to blasting different kinds of coal and rock, keeping in view the safety of the workmen.

ANS.—In any case, the size of the grain, which determines largely the quickness with which the powder acts, should be suited to the hardness of the material to be blasted. The smaller the grain the quicker the powder acts. Rock and hard coal require a quicker powder than a loose shale or soft coal. A deflagrating powder, as black powder, is slower and better adapted to blasting coal, while the detonating powders of the nitroglycerine class, as dynamite, are better adapted to rock. A quick powder breaks the coal too small and produces much fine coal and dust. Where the coal or adjacent strata contain gas, or the coal is friable and produces much fine dust, flameless powders should be used.

QUES. 24.—State in detail what should be done to remove or reduce to a minimum the dangers due to coal dust.

ANS.—Adopt a powder that is suited to the coal and allow no other powder to be used in blasting coal. Take proper precautions to avoid any excessive use of powder. Have all fine coal and slack loaded out with the coal, and permit no accumulations of fine coal and dust at the face. Use tight cars in the mine and do not overload them. Have the roadways cleaned regularly, and, if necessary, sprinkle them with water often enough to lay the dust. If the mine is dry and dusty, especially if some gas is present and the coal friable or highly inflammable, each place should be tested for gas and sprinkled before firing a shot therein. If necessary appoint competent men and authorize them to examine all shots and condemn any that, in their opinion, are unsafe. If the conditions further require, employ shot firers to examine and fire shots after the men have left the mine.

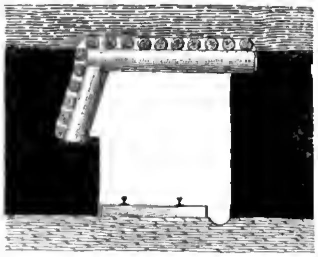


FIG. 1

QUES. 25.—What precautions would you use in timbering haulageways in a mine to prevent the timbers from being knocked out should cars leave the track?

ANS.—If the coal is firm and strong, short legs may be used to support the collars, the foot of the leg being set in a hitch cut in the coal, as shown in Fig. 1, or above the coal where roof slate is taken down to make headway for the cars. In some cases, no legs are used, but the ends of the collar are set in hitches cut in the coal, as shown in Fig. 2. When this is done one of the hitches must be cut so as to allow the stick to be ended into place and wedged. Where the coal is too brittle to support the timbers, long legs must be used standing on the floor, each leg being recessed into the rib beyond the reach of a derailed car.

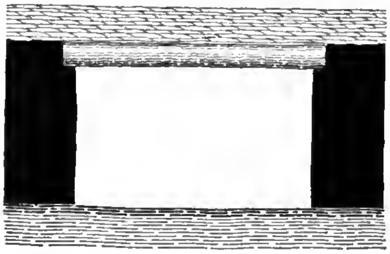


FIG. 2

QUES. 26.—If a water gauge of 1.2 inches produces a velocity of 500 feet per minute in an airway 6 ft. × 8 ft. and 1 mile long, what water gauge will be required to produce a velocity of 750 feet per minute in an airway of the same section and 2 miles long?

ANS.—The two airways being of the same section will have equal perimeters and areas, and, in that case, the pressure as shown by the water gauge will be proportional to the length of the airway and the square of the velocity of the current; or, in other words, the water-gauge ratio will be equal to the product of the length ratio and the square of the velocity ratio. Thus, calling the required water gauge x,

$$\frac{x}{1.2}=\frac{2}{1}\times\left(\frac{750}{500}\right)^2=2\left(\frac{3}{2}\right)^2=2\times\frac{9}{4}=4.5$$

and $$x=1.2\times 4.5=5.4 \text{ in.}$$

NOTE.—When a question is stated absolutely, as above, care should be taken that the statement is correct. As a fact, by the usual method of calculation and the average results of mining practice, it would require a water gauge of practically 3 inches to produce a current velocity 500 feet per minute in an airway 6 ft. × 8 ft. 1 mile long. Some candidates would calculate at once the required water gauge to produce a velocity of 750 feet per minute in the same airway, 2 miles in length, as follows:

$$\text{water gauge}=\frac{.00000002\times 10,560\times 28\times 750^2}{48}=13.3 \text{ in.}$$

which is the case practically, but is not the correct answer to the question asked. The question is unintentionally a catch question.—ED.

QUES. 27.—What method would you adopt for timbering a room having a tender roof when chain and puncher machines are to be used, to insure the safety of the workmen?

ANS.—Posts should be set in regular rows a specified distance apart, and the distance of the first row of posts from the face of the coal should not exceed this amount unless temporary lagging is used to support the roof above the machine. In order to allow the machine to move along the face, each post in the first row should be taken out in turn and reset as the machine passes the place. Sprags should be set or wedges be driven under the coal as the machine advances.

QUES. 28.—If an airway is 6 ft. × 8 ft. and 2,000 feet long and the velocity of the current 500 feet per minute, what is the power producing the ventilation?

ANS.—

$$H=\frac{.00000002\times 2,000\times 28\times 500^3}{33,000}=4.24 \text{ H. P.}$$

QUES. 29.—What general conditions would guide you in determining the width of rooms and the dimensions of room pillars?

ANS.—First, the maximum width of opening or width of rooms in a mine is determined by the depth of cover or roof pressure, the kind of the roof, floor, and coal, and the length of time the rooms must be kept open. The proper width of pillar is then decided, and depends on the width of the rooms, depth

of cover, kind of roof and floor, strength of coal, thickness and inclination of the seam. The length of room pillars will depend chiefly on the inclination of the seam and the requirements of ventilation.

Ques. 30.—Which method do you prefer, the timbering of the working places by the miners employed therein, or by practical timbermen employed for that purpose? Give reasons.

Ans.—Every miner should learn how to set a post where it is most needed, and be required to timber his own place; as this would make him more watchful and careful of his own safety. When a post is needed it should be set at once, and any delay is dangerous. The miner should do the work, because, if compelled to wait for a regular timberman, he would, in practically every case, continue to work in the danger and risk his life.

FIRST-GRADE QUESTIONS

Ques. 32.—In a mine giving off 2,500 cubic feet of marsh gas per minute the volume of air entering the intake opening is 4,500,000 cubic feet per hour; what is the percentage of gas in the return current? Would you consider this percentage of gas dangerous?

Ans.—Assuming that the temperature of the return air is the same as that of the intake, the volume of the return current of air and gas is $4,500,000 \div 60 + 2,500 = 77,500$ cubic feet per minute. The percentage of gas in this current is $\frac{2,500 \times 100}{77,500} = 3.22$ per cent. This would be a dangerous percentage of gas for general working conditions in bituminous mines.

Ques. 33.—Why is afterdamp a variable mixture?

Ans.—Because it is the product of the explosion of variable mixtures of gas and air, often involving quantities of fine coal dust and other gases than marsh gas. The products of the combustion (afterdamp) will depend on the kind of gases burned and the relative proportions of air and dust present. Hence, afterdamp is always a variable mixture, depending on conditions that are never wholly alike in any two cases.

Ques. 34.—What method of timbering and drawing timber would you adopt in pillar work to insure the best results in respect to both safety and economy; and when, where, and how would you instruct the workmen?

Ans.—In pillar work posts must be set as experience shows they are needed. No rule can be laid down, as the roof and pillar conditions vary widely. The condition of the pillar coal and the movement of the roof must be closely watched by the effect on the posts stood behind and about the workmen for their protection. Sufficient timber must be always on hand. Posts must be drawn regularly as the work proceeds so as to induce falls and relieve the roof pressure on the pillars. To draw posts the chain and dog should be used instead of trying to knock out the post with a sledge. The men should be instructed, before attempting the work, by those who have had experience, and this should be done at the time and place.

Ques. 35.—What method would you adopt in drilling, charging, and firing shots to insure the greatest safety to the workmen and the mine?

Ans.—The miners should drill their own holes according to their best judgment and experience. Each hole should be inspected and passed or condemned by a duly authorized and competent inspector before the same is charged. The inspector should determine the weight of charge to be placed in each hole at the same time, and make a full record of each examination. In most cases, no shots should be fired till a fixed time arrives, and then only in regular order of rotation beginning on the end of the air and proceeding toward the intake. Violation of this rule should be punished. Where the conditions require, all shots should be fired by shot firers after the men have left the mine.

Ques. 36.—Which gas diffuses the more rapidly, CH_4 or CO_2? Give the ratio of diffusion of these two gases.

Ans.—Marsh gas (CH_4) diffuses more rapidly than carbon dioxide (CO_2). The relative rates of diffusion of the two gases are expressed by the ratio

$$\frac{1}{\sqrt{8}} : \frac{1}{\sqrt{22}}; \text{ or } 1 : .603$$

in which 8 and 22 are the densities, respectively, of marsh gas and carbon dioxide.

Ques. 37.—What gases are commonly found in the bituminous mines of Pennsylvania?

Ans.—Marsh gas (CH_4), carbon monoxide (CO), carbon dioxide (CO_2), olefiant gas (C_2H_4), and hydrogen sulphide.

Ques. 38.—What instructions would you give shot firers before permitting them to enter upon their duties?

Ans.—Allow a sufficient time after the men have left the mine for the return air to become clear and free from dust. Ascertain that the air-current is traveling with its usual velocity. Begin work on the last of the air in each section or air-split of the mine. Examine each shot carefully, and do not fire a shot that appears to be doubtful or in any way unsafe. Never fire more than one shot at a time in a close heading or room, and always give time for the air to clear before firing a second shot in a close room. Where gas is generated it is necessary to test each place for gas before firing a shot therein.

Ques. 39.—Give the specific gravities of the gases common to the bituminous mines of Pennsylvania and state where each gas is to be mostly found.

Ans.—Marsh gas, specific gravity .559; found mostly where it issues from the coal or strata, or where it has accumulated in rise workings, or at the roof of rooms or entries. Carbon monoxide, specific gravity .967; found mostly in abandoned rooms and other badly ventilated places, or in the vicinity of gob fires, or where shots have been fired in a close room or heading. Carbon dioxide, specific gravity 1.529; found in low places, dip workings, and at the floor of rooms or entries that are poorly ventilated. Hydrogen sulphide, specific gravity 1.1912; rarely found in troublesome quantity, except in some damp low places where sulphide of iron (pyrites) occurs in the coal and the ventilation is poor.

Ques. 40.—When and where should regulators be used in conducting air-currents in mines?

Ans.—Regulators are required whenever the air is to be divided in other proportions than the natural division of the current as determined by the potential of each split. A regulator should be placed, if practicable, at the mouth of the split where the air is divided, but not on a haulage road.

Ques. 41.—What precautions would you take when driving entries through gaseous and faulted territory?

Ans.—Use only safety lamps and make a test before beginning each shift to ascertain the first signs of an increase or decrease of gas at the head of each entry. Any change in the percentage of gas for the same quantity of air passing would indicate a corresponding increase or decrease in the outflow of gas from the strata; and, in most cases, if this continued, it would indicate the near approach to a fault. To make such a test a sight-indicator should be used in a lamp burning a non-volatile oil. Volatile oils, as gasoline or naptha, heat in gas and do not, on this account, give reliable readings. It is well, also, to keep a drill hole, say from 3 to 5 yards in advance of the face of each heading, and watch closely for any increase in amount of water draining from the strata.

Ques. 42.—Describe in detail what examination you would make before firing shots.

Ans.—Always examine the place for gas, both at the roof and in any places not reached directly by the air-current. Observe also if the usual quantity of air is passing, and that the same is fairly clear or free from dust. See that all tools and powder are removed to a safe place and that there is no accumulation of fine coal and dust in the vicinity of the shot.

FIRE BOSSES' EXAMINATION

Ques. 1.—When and where would you expect a sudden inflow or outburst of explosive gas, and how would you guard against danger therefrom.

Ans.—When driving headings or rooms to a fault line or toward abandoned workings, or when drawing back pillars, or during a heavy squeeze or creep, or accompanying a heavy fall of roof. To guard against this danger safety lamps should be used exclusively when driving rooms or entries in faulted strata, or drawing back pillars, or when approaching abandoned workings, or during the progress of a mine squeeze, if there is any reason to suspect gas being in the strata. The danger arising from heavy roof falls is best avoided by reducing the amount of standing area by drawing timber and allowing worked-out places to close.

Ques. 2.—What precautions would you take with your safety lamp before entering the mine, and how would you proceed with its use in the mine to insure the greatest safety?

Ans.—The lamp should be thoroughly cleaned, filled, and trimmed, the gauzes examined, and the parts put together carefully. If possible, before being taken into the mine, the lighted lamp should be tested by exposure to an explosive atmosphere in a box or test chamber so arranged that the lamp can be observed during the test. When using the lamp in the mine, it should be held erect, should not be swung or allowed to fall. The lamp should be observed constantly, and the flame should not be carried too high, or the lamp allowed to smoke. It should not be exposed to gas longer than to determine the kind and amount of gas present; it should be carefully shielded from a strong air-current or blast. In case the lamp flames, it should quietly but promptly be lowered and removed from the gas. No quick motion should be made.

Ques. 3.—What qualifications, other than those required by law, are necessary to make a good, efficient fire boss?

Ans.—The fire boss must be intelligent, faithful, and conscientious in his work; must have a good theoretical and practical knowledge of mine gases and their behavior in mines, the atmosphere, and the effect produced in mines by changes in atmospheric pressure, temperature, and moisture; the principle, construction, and use of the safety lamp, barometer, anemometer, water gauge thermometer, and hygrometer, in mining; the principles of mine ventilation, and the means and methods of generating and conducting air-currents in mines.

Ques. 4.—When does a safety lamp become unsafe?

Ans.—When the lamp is not clean, or not properly put together, or has been injured in some part, or is improperly or carelessly handled by an incompetent person; when exposed to a strong air-current, or allowed to fall, or is swung, or carried too rapidly against the air, or exposed for too long a time to gas; and lastly, when the lamp is improperly designed.

Ques. 5.—What protection is secured in a mine worked exclusively with locked safety lamps that cannot be expected in a mine worked exclusively with open lights?

Ans.—The mine using safeties exclusively is protected against sudden outbursts of gas due to a heavy fall of roof, or the sudden tapping of a large feeder, or a sudden fall of barometric pressure where large abandoned areas exist containing gas. Such a mine is also protected against the sudden disablement of the fan or other means of ventilation.

Ques. 6.—Name the gases usually found in bituminous mines; state where they are found, and give briefly their injurious effect on the workmen.

Ans.—This question is partly answered in reply to Ques. 37 and Ques. 39 of the mine foremen's examination. Marsh gas has no injurious effect when mixed with air and breathed, but is injurious by reason of its inflammability and because it forms an explosive mixture with air. Carbon monoxide and hydrogen sulphide are each extremely poisonous when breathed, even in small quantities; they are also, like marsh gas, inflammable and explosive in air. Carbon dioxide is not poisonous but suffocates when breathed in sufficient quantity.

Ques. 7.—When, where, and how would you instruct the men as to the use of danger signals to get the best results?

Ans.—All men should be informed and instructed as to the use of danger signals before they are allowed to enter the mine. This should be done as far as possible at the mine before they enter; but in order to make these danger signs and signals better understood by all, they should be further explained and their operation shown underground and before beginning work.

Ques. 8.—Could an explosion occur from a sudden outburst of marsh gas under a high pressure; and, if so, under what circumstances?

Ans.—A sudden outburst of marsh gas in any part of a mine often fills the adjoining workings with an explosive atmosphere before the men can be notified to put out their open lights and withdraw. Where open lights are used it is possible that an explosion may occur under these circumstances.

Ques. 9.—If an explosion occurred in a safety lamp, or if the lamp should suddenly fill with flame, what would you do to insure your safety?

Ans.—Lower the lamp promptly but slowly to the floor; and, if possible, if the flaming ceases, withdraw quietly carrying the lamp low and making as little commotion as possible. If the lamp continues to flame after it is on the floor, it may be possible to smother the flame by wrapping the lamp in a coat; but this must be done carefully to avoid forcing the flame through the gauze by the wind caused in folding the coat about the lamp.

Ques. 10.—As a fire boss, what instruction would you give the men in your section in regard to the use of safety lamps, assuming they are not familiar with the same, and when and where should such instructions be given to secure the best results?

Ans.—A safety lamp should never be given to a man till he has been made fully acquainted with all the points mentioned in reply to Ques. 4, which instruction should be illustrated practically with a lamp. This should be done at the mine before entering, and special attention should be given these men underground by the fire boss till they understand fully how to handle and use a lamp.

Ques. 11.—What duties should be assumed by a fire boss, acting as assistant mine foreman, while on his second visit through the mine or any part thereof?

Ans.—On his second visit he should carefully inspect each working place to see that the same is properly timbered, a sufficient amount of posts and caps on hand, the coal properly mined and spragged, the lamps used properly, and the place properly ventilated, etc.

Ques. 12.—Are you in favor of mixed lights in a mine? Give reasons for your answer.

Ans.—No. Because the conditions requiring the use of safeties often arise suddenly and without warning; and because an accumulated body of gas may be suddenly disturbed by a heavy fall of roof or by a windy shot, or other cause; and render the mine air in that vicinity explosive. Moreover, the general miner is too willing to run risks with open lights that expose himself and others to grave danger. If the conditions require the use of safeties in any section of a mine, they should be used exclusively in that section.

Ques. 13.—In testing for explosive gas, how would you satisfy yourself that there was not sufficient gas to indicate its presence on the flame of a lamp?

Ans.—The more sensitive test is to lower the flame to the smallest possible size and elevate the lamp slowly to the place where the gas is suspected or may be found. Some prefer testing with the normal flame. Good eyesight is required and the eyes must be screened with the hand to detect the first indication of gas. Many fail to detect gas when present, because they proceed hastily and disturb the gas too much before the

test can be made, and then pass on, leaving the gas undetected, while more deliberation would have revealed its presence. The best and surest test is with the sight indicator, which shows clearly the exact condition without guessing.

Ques. 14.—Do you think it advisable to report gas on the fire bosses' record book when you would have to hold your safety lamp against the face of the coal to detect it, and could not get a showing in the lamp by holding it up to the roof?

Ans.—Under the present system, to report "gas" in a working place would mean a dangerous quantity of gas, and would debar men from working there till the danger was removed. Therefore, the report should only show the finding of sufficient gas to make the place unsafe for work. Where extra caution is required, owing to a free outflow of gas from coal or strata, or from lack of efficient circulation of air at the face, or other cause, the report should show such condition. Every fire boss should be equipped to show the exact gaseous condition in every place examined. On his second round he should give this matter closer attention than he is able to do when making his first early morning examination, because it is when the mine is working that the greatest danger exists. The fire boss should keep his own private record showing the exact percentage of gas each day at certain points in the mine

Catalogs Received

In writing for catalogs, please mention Mines and Minerals

The Jeffrey Mfg. Co., Columbus, Ohio, Catalog No. 26A, Jeffrey Mine Fans, 36 pages.

Allis-Chalmers Co., Milwaukee, Wis., Bulletin No. 1432, Stamp Mills and accessory machinery for free-milling gold ores, 60 pages

The Bristol Co., Waterbury, Conn., Bulletin No. 152, Bristol's Patent Steel Belt Lacing, 12 pages.

The Cutler-Hammer Mfg. Co., Milwaukee, Wis.. Elevator Controllers, Schureman Types, 63 pages.

De Laval Steam Turbine Co., Trenton, N. J., Catalog A, De Laval Steam Turbines, Single-Stage Type, 120 pages.

S. Flory Mfg. Co., Bangor, Pa., Flory Hoisting Engines, Steam and Electric, 120 pages.

General Electric Co., Schenectady, N. Y., Bulletin 4685, Belt-Driven Alternators, 12 pages; Bulletin 4804, Direct-Connected Generating Sets, 12 pages; Bulletin 4810, Portable and Stationary Air Compressor Sets, 8 pages.

The Hill Clutch Co., Cleveland, Ohio, Catalog No. 9, is quite a book on Power Transmission Machinery, and contains 194 pages.

Ingersoll-Rand Co., 11 Broadway, New York, N. Y., The "Heart Beat" Oiler, 4 pages; Bulletin 3007, Class "PB" Duplex Power-Driven Air Compressors, 24 pages; Bulletin 4109, Temple-Ingersoll "Electric-Air" Rock Drills, 24 pages; Bulletin 9003, Rock Drill Mountings, Steels, Hose, and Accessories, 44 pages.

J. Geo. Leyner Engineering Works Co., Littleton, Colo., Bulletin 1027, Reprints of Advertisements, 12 pages; Bulletin 1026, Tunneling on Los Angeles Aqueduct, 16 pages.

Link-Belt Co., Philadelphia, Pa., Booklet No. 95, Conveying Machinery for Coal Mines, 42 pages.

National Electric Lamp Association, Cleveland, Ohio, Bulletin 5C, Tantalum Multiple Lamps, 12 pages; Bulletin 8B, "Mazda" Miniature and Low-Voltage Lamps, 12 pages; Bulletin No. 15, Electric Sign Lighting, 16 pages.

Pneumelectric Machine Co., Syracuse, N. Y., An Electric Coal Puncher, 11 pages; Pneumelectric, 39 pages.

Stephens-Adamson Mfg. Co., Aurora, Ill., The Labor Saver, Efficiency, Economy, 24 pages.

J. B. Lippincott Co., Philadelphia, Pa., The Value of Certain Paint Oils, 18 pages.

Stromberg-Carlson Telephone Mfg. Co., Rochester, N. Y., Booklet No. 253, Stromberg-Carlson Inter-Comm-Phone Systems, 20 pages.

Eagle Foundry and Machine Co., Fort Scott, Kansas. Pamphlet, 8 pages, descriptive of Cyclone Propeller Fans with special reference to underground installations.

New Inventions

Complete specifications and drawings for any of the following patents can be obtained from the Commissioner of Patents, Washington, D. C., at the following rates

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
At entire set of patents	1 cent each

Patents Pertaining to Mining Issued January 3 to January 31, 1911, Inclusive

No. 980,776. Drill coupling, Martin Hardsocg, Ottumwa, Iowa.

No. 980,196. System of mining coal, Charles Arnold Cabell, Carbon, W. Va.

No. 980,244. Ore classifier, John P. Ginivin, Seattle, Wash.

No. 980,202. Ore concentrator and sliming table, Christopher A. Christensen, Portland, Ore.

Nos. 980,890 and 980,891. Process for dry ore separation, William W. Bonson, Dubuque, Iowa.

No. 981,344. Coke and gas oven, Louis Wilputte, Joliet, Ill.

No. 981,170. Method of preventing mine explosions, John W. Coleman, Maybeury, W. Va.

No. 981,226. Mine roof support, James Gardner Sanderson and Clarence B. Sturges, Scranton, Pa.

No. 981,243. Mining bit, William W. Bittenbender and Elias G. Bittenbender, Nanticoke, Pa.

No. 981,201. Method of mining coal, Harry A. Kuhn, Pittsburg, Pa.

No. 981,306. Mining tool, James Pollock, New York, N. Y.

No. 981,230. Apparatus for classifying and concentrating ores, Rene Emile Trottier, Hussein-Dey, France.

No. 981,172. Sand trap for pumps, Rudolph Conrader, Erie, Pa.

No. 981,073. Chemical shot firer, John Dowd, Chicopee, Kans.

No. 981,732. Leveling ram for coke ovens, Louis Wilputte, Joliet, Ill.

No. 981,908. Safety mechanism for mine cages and elevators, Thomas G. Baird, Ladd, Ill.

No. 982,095. Safety appliance for mines, Henry William Scheiler, Lawndale, Ill.

No. 981,681. Centrifugal ore-separating apparatus, Philip F. Peck, Tacoma, Wash.

Nos. 981,679, 981,680 and 981,682. Centrifugal ore-separator, Philip F. Peck, Tacoma, Wash.

No. 981,880. Process for roasting sulphide ores, Charles W. Renwick, Isabella, Tenn.

No. 982,765. Apparatus for determining water in coke, etc., James A. P. Crisfield, Philadelphia, Pa.

No. 982,625. Spraying mechanism for coke ovens, Thomas J. Mitchell, Uniontown, and James A. McCreary, Connellsville, Pa.

No. 982,590. Apparatus for quenching, screening, and loading coke, Albert Goodall, Spennymoor, England.

No. 982,612. Miner's lamp, George Laws, Philipsburg, Pa.

No. 982,583. Hydraulic classifier for ores, James N. Flood, Denver, Colo.

No. 982,243. Desulphurizing apparatus for refractory ores, Charles Anderson Case, New York, N. Y.

No. 982,245. Process for reducing and desulphurizing refractory ores, Charles Anderson Case, New York, N. Y.

No. 982,983. Gas burner for coke ovens, Robert Muller, Essen-on-the-Ruhr, Germany.

No. 982,899. Crushing machine, Josiah E. Symons, Chicago, Ill.

No. 982,785. Ore concentrator, Samuel K. Behrend, Denver, Colo.

No. 983,067. Ore pulverizer, John J. Knight, Alameda, Cal.

Nos. 983,068 and 983,069. Tube mill, Povl T. Lindhard, New York, N. Y.

Mines and Minerals

Vol. XXXI—No. 9. SCRANTON, PA.—APRIL, 1911—DENVER, COLO. Price, 25 Cents

THE PECK'S SHAFT BREAKER

Written for Mines and Minerals, by E. B. Wilson

Process of Preparing Anthracite by a Combination of Wet and Dry Methods

It has been the custom in the Lackawanna anthracite field of Pennsylvania to prepare coal by the dry process, while in the Middle and Schuylkill anthracite fields it has been the custom to use the wet process. One of the most up-to-date breakers in the Lackawanna valley is that at the Peck's shaft of the Mt. Jessup Coal Co. at Peckville, which combines many of the approved old and new features in breaker construction, and has adopted both the dry and the wet preparation of coal.

In addition to this improvement, the old method of hoisting the coal to the top of the breaker in cars, then dumping, has been discarded, and the new method of dumping at the surface and using an inclined conveyer substituted. Another of the many features worthy of particular notice is the absence of revolving screens—there not being one in the building—that style having been discarded for the shaking variety. The toothed rolls are made of manganese steel and in segments, not individual teeth. At first the advantage of this move is not so apparent until it is understood that the manganese teeth are less liable to breakage and continue to remain sufficiently sharp to crack the coal long after the replaceable steel tooth has dulled and commenced to pulverize the coal. The economy of throwing away an entire segment of manganese steel when a tooth becomes broken, where in the older style roll it is only necessary to knock out a tooth and replace it with another at first seems doubtful. The economy is figured out as follows: The teeth of a manganese roll do not break, but wear uniformly; the labor involved in removing and repairing a broken tooth frequently costs more than a segment, which can be placed in a short time; finally, the wear is greatly in favor of manganese, not to mention the better cracking ability of the solid tooth.

All coal previous to going to the boy slate pickers passes through one of the two kinds of spiral pickers. The wet coal goes to a picker designed for the purpose, while the dry coal goes to another kind. It can be readily understood that wet coal will slide on an incline differently from dry coal, and this difference is sufficient to require special construction for wet-coal spiral pickers. At this breaker the jigs are run on the bony coal that comes from the slate pickers.

The advantages to be derived from the wet preparation of coal are as follows: Dust is kept down; the mechanical separation of coal and impurities is readily performed through the medium of water; and finer material can be treated than by the dry process, since by the abundant use of water the screens do not clog.

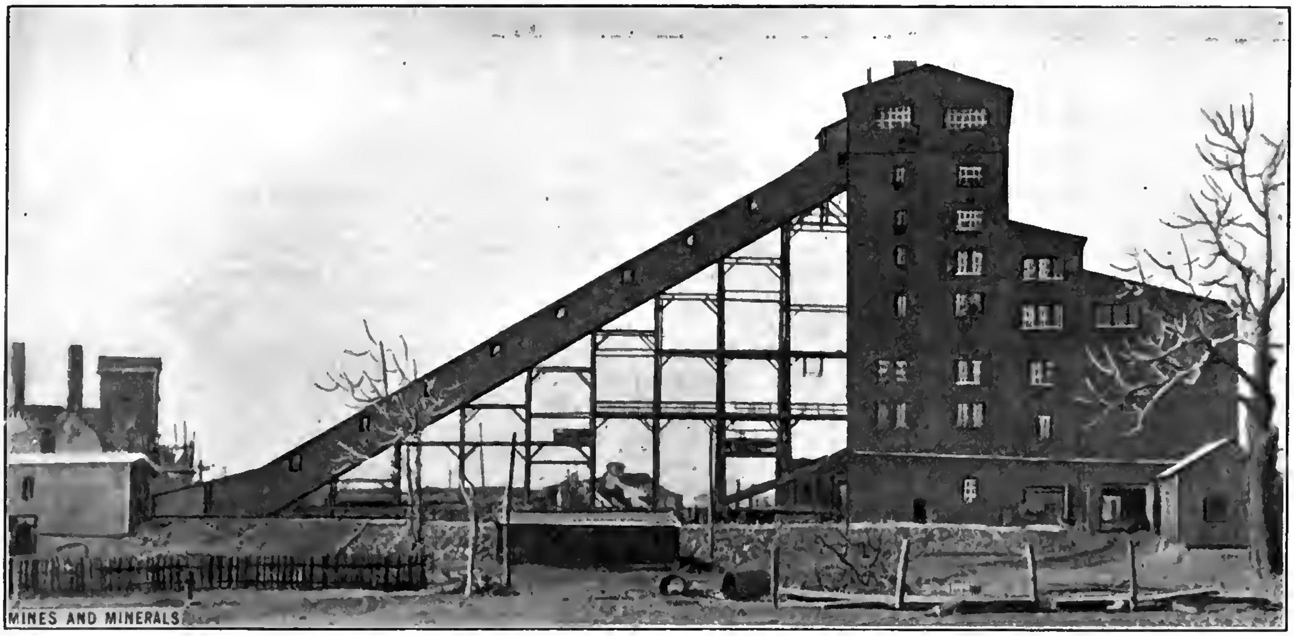


Fig. 1. Peck's Shaft Breaker, Mt. Jessup Coal Co.

The disadvantages are the expense of pumping, the sloppy condition of the breaker, and the disposal of the refuse. From the advantages and disadvantages mentioned it does not require more than average discernment to note that a compromise between the two systems of preparation is better under present-day markets than either separately when small coal was unsalable. To obtain a general idea of a modern compromise breaker, a description of the Peck's shaft breaker shown in Fig. 1, is ideal.

The long incline is for the scraper line which lifts the coal from the tipple house at the bottom to the very top of the breaker. To the rear of the tipple house is the Sterling boiler plant.

While the breaker is constructed of wood it is equipped with sprinkling and fire-extinguishing apparatus from the top to the bottom, so that any or all floors can be deluged with water if necessary. The sprinklers can be turned on in several places at the bottom of the breaker, while the water in the fire

plugs is always under pressure and cannot be turned off anywhere except at the plugs on the various floors. This arrangement makes the system positive, with no possible chance of any one tampering with the valves; consequently it is always ready for immediate use.

The coal comes from the Dunmore and from the Clark beds. It is hoisted in cars up an underground incline about 2,400 feet long, and then the cars are run by gravity a short distance to the foot of the shaft in the Clark bed. From the Clark bed, which is about 430 feet under cover, all coal mined is hoisted in cars through a vertical shaft to the surface. From the shaft collar the coal cars run by gravity in a semicircle to the dump house, where the tipple shown in Fig. 2 is installed. As a loaded car approaches this dumping device, it is assumed for description that there is an empty car standing on the movable and pivoted rails, held there by two tipple horns. When the loaded car reaches the lever, shown between the rails to the rear, its weight presses it down, thereby spreading the horns which held the empty cars stationary. As the loaded car moves forward it strikes the rear of the empty car, setting that in motion, so that it runs down an incline around to the car haul, shown indistinctly in Fig. 1 between the middle bent under the incline, and then back to the shaft collar, completing the circle.

Fig. 2. Car-Dumping Arrangement

So soon as the loaded car passes over the lever the horns of the tipple assume an upright position, and catching the forward wheels, bring the approaching car to a stop Underneath the floor of the tipple there is a steam cylinder whose piston is attached to the rear of the tipple rails by a yoke. When steam is admitted to the cylinder the piston lifts the pivoted rails and the rear of the car so that the car is tilted at the proper angle for discharging its load, after which the steam is turned off and the tipple and car return to their normal position.

There is a lever on the post shown to the left of the tipple which works the throttle valve leading to the steam cylinder, thus allowing the dumpsman full control. As the car commences to tip, a weight is disengaged which spreads the rails in front of the tipple so that no coal strikes them, and as soon as the car commences to go back to its normal position the rails return to their original place for the empty car to cross over them.

The coal is dumped in a hopper from which it is fed automatically by a revolving wheel to the scraper line, which is 228 feet long between centers and pitched at an angle of 30 degrees. The flights in this line are rigidly fastened to flat-linked Wilmot chains and do not come in contact with the trough. The chain moves on hard-wood rails without a squeak, and the rails being properly lubricated, there is little friction or wear.

The conveyer delivers the coal directly to the chute at the top of the breaker leading to the lump-coal shakers shown in Fig. 3. Large wooden fingers placed at the bottom of the chute prevent the large pieces rushing on the screen with momentum sufficient to injure it. This arrangement does away with a gate, and as there is no intermittent rush of coal the screen is never flooded. From the lump shakers the coal goes to the traveling picking belt, where the rock and coal frozen to rock are separated by hand as shown. The coal remaining on the belt is comparatively clean and is delivered by the belt to the crushing rolls. The rolls crush dry, but they, with the natural ventilation, create sufficient air-current to carry the little dust made out of the chimney shown in the top of the breaker, Fig. 1. The rolls in this breaker are run at about 125 revolutions per minute, or at about one-half the speed most rolls are driven. It has been found that by driving large rolls slowly better crushing results are obtained on anthracite than by running small rolls fast. This system would not apply to crushing bituminous coal fine. From the rolls the coal goes by gravity to the dry-coal shaking screens where it is sized into stove, egg, and nut, the undersizes going to the small-coal screens. From the dry-coal screens the stove, egg, and nut go to the spiral slate pickers shown in Fig. 4, there being a picker for each size. Attention is called to the fact that in this breaker all moving parts of machinery are fenced, and in order to get to the machinery the person must go out of his way. All the screens in this breaker are of the shaking variety, even to the finest material, but are so arranged that there is less vibration than in breakers using rotary screens. It will be noticed also in Fig. 4 that the timber frame is bracketed and tie-braced, a system of framing that is followed throughout the structure. Where this system of framing is used the timber legs must be perpendicular; and the cross-beams be made flush with the posts and tied so that there will be no spaces between timbers. In this particular breaker the machinery is arranged so that it fails to get in step, but this advantage is due as much to the delivery of the coal by means of the scraper line as to the bent construction.

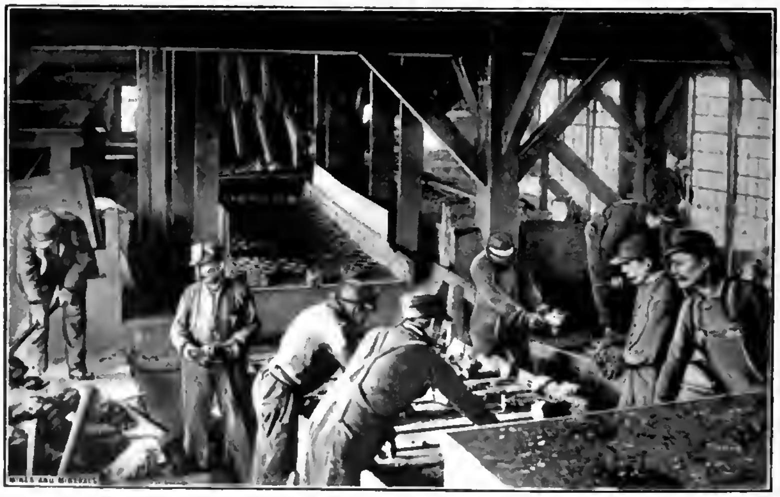


Fig. 3. Lump-Coal Shakers and Picking Table

From the spiral slate pickers the clean coal goes to the slate picker boys shown in Fig. 5. Breaker boys are artists in their line, being able to spear slate or bony coal with either hand, and without lifting their eyes place either in its proper chute. The bony coal picked out by the boys and that from the spiral pickers is delivered to Hazleton jigs, where a further separation of bony coal and slate occurs.

Part of the coal that passes through the lump-coal screens goes to 27″×36″ rolls, the remainder to the mud coal screens where egg, stove, chestnut, and pea are separated, also flat pieces of slate and bony coal. These sizes are sent direct to the spiral pickers and the undersizes to the small-coal shaking screens. From the spirals the coal goes to the boys' picking chutes, while the bony coal goes to the jigs. These rolls can also be employed if it is desired to break down coal to smaller sizes. At this breaker the lip screens are arranged so that they deliver to a trough pitched from each side to a pocket situated about in the center of the breaker, but below the broad-gauged track. Whenever there is a rejected car of coal it is dumped into this pocket and carried by a scraper line to the elevators, hence to the dirty-coal screens at the top of the breaker and so back through the breaker.

Fig. 5. Boy Slate Pickers

When the writer was actively engaged in anthracite mining it was the custom to construct a flow sheet and build the breaker to conform; at the present time the custom is reversed, with the result that innumerable changes and adjustments are made before the plant works satisfactorily. One operator remarked that the flow sheet was the last thing considered. As he was a mining engineer—well, those who know say the writer is not altogether politic.

The Peck's shaft breaker was constructed by Kingsley & Wescott, of Scranton, after ideas deduced from the experience of K. M. Smith, vice-president of the company, for which reason there will be few changes made, if any, in the flow sheet of this breaker.

A detailed account of the coal preparation is subtended.

The flow sheet of this breaker is shown in Fig. 6. The incline conveyer delivers the coal to the lump-coal screen *a*; all that goes over is carried away by picking belt *b* and delivered to 32″×36″ crushing rolls *c*.

Fig. 4. Spiral Pickers

The coal that passes through the rolls is elevated by *d* to the dry-coal shakers *e*, where it is sized into egg, stove, and nut, the undersize going to shaker *f*, where it is sized into pea and Nos. 1, 2, and 3 buckwheat.

The coal which passes through *a* and rests on grate-coal shakers *g* goes to a pair of 27″×36″ rolls *h* and is elevated and prepared over dry shakers *e* into egg, stove, nut, and pea. From *e* all coal goes to the dry-coal spiral pickers *i*, and from them to the boys' picking chutes *j*, and to the bins.

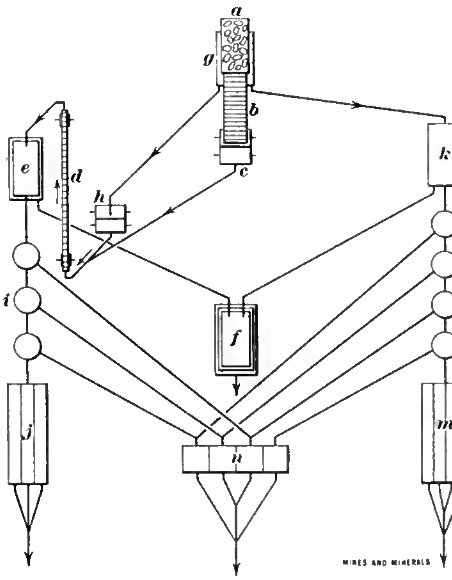


Fig. 6. Flow Sheet

That coal which goes through grate-coal shakers *g*, goes to the mud screens *k*, which are sprayed with water, and is sized into egg, stove, nut, and pea and sent to the wet spiral pickers *l*. The undersize from the mud screens goes to table *f* and is sized wet to pea and Nos. 1, 2, and 3 buckwheat. From the spiral pickers the coal goes to picking tables *m*. The bone from the dry pickers *i* goes to egg, stove, and nut coal jigs *n*, that from the wet spiral pickers, to egg, stove, nut, and pea coal jigs. The product of the jigs goes to bins.

The writer is indebted to P. A. Murphy, outside foreman, for the information relative to the details of this breaker that are contained in this article.

STEEL MINE SHAFT CONSTRUCTION

*By R. B. Woodworth, C. E.**

The circular shaft lined with brick or cast iron is extremely common in England and on the Continent. Meyer in discussing English practice remarks that the rectangular shaft is rather common in Cumberland, indicating that elsewhere its use is rather uncommon, and it is noteworthy that of the 14 shafts described in his book on "Mining Methods in Europe," all are circular with the exception of one, and that one is elliptical. It is also worthy of note that of 150 shafts listed by J. Riemer in his book entitled "Shaft Sinking Under Difficult Conditions," all but five are circular. Meyer also notes that there is a circular shaft in a Grundy County coal mine near Coal City, 60 miles west of Chicago. Outside of this the writer recalls but one other instance of a circular shaft which has come under his own observation, and certainly the common form in the United States is the rectangular.

Conditions Influencing the Material for Shaft Lining. Forms of Steel Linings and Supports

The advantages of the circular shaft as compared with the rectangular are stated by R. A. S. Redmayne in Vol. 2 of his work on "Modern Practice in Mining," to be:

1. The decreased cost of sinking due to the elimination of corners.

2. The smaller cost in lining, a rectangular shaft lined with timber costing £2 5s. per foot, whereas a circular shaft lined with brick at 20 shillings per 1,000 would cost £1 4s. 2d. per foot.

3. The greater difficulty of shutting off water in the square shaft.

4. The greater difficulty of placing a rectangular shaft in the most suitable situation with regard to surface and underground arrangements, position of railway sidings, etc., it being desirable that the greatest length of a rectangular shaft should be across the cleat of the stone so that the long side may be in the position most easily supported.

5. The increased danger and increased expense in a rectangular shaft when it passes through a fault.

6. The greater durability of cast iron and brick usually employed in lining a circular shaft as compared with the timber with which the rectangular shaft is lined.

These considerations in the ultimate analysis are due largely to differences in mining conditions, and the practice in building shafts in England and on the Continent is different from that which obtains in America largely by reason of the different material through which the shafts are sunk and the materials which are used in their construction. In certain parts of England, for example, and in certain mines on the Continent, the upper strata are of recent geological origin; the shafts are sunk in the valley while the seams of coal or ore mined outcrop on the hills or at the surface, sometimes many miles away. The result of this configuration of the soil is to permit enormous masses of water to percolate into and form reservoirs in the strata lying above the coal or ore. As the shafts are sunk, these strata of course have to be penetrated, causing difficulties in the way of pumping and keeping out the water. By reason of this difference also the pressures on the shaft lining increase approximately in the ratio of the hydrostatic head. In the United States, however, the coal mines are either not of any great depth, or where they are, they almost always pass through strata of Carboniferous time which are approximately horizontal, very hard, free from faults, and relatively free from water. In the iron-mining regions the ore occurs in troughs of slight extent with the strata deeply fissured, folded, and faulted; consequently the shaft is sunk through material which may contain considerable quantities of water, with sand, gravel, shales, etc., but after bed rock is reached its extremely hard nature eliminates difficulties of the kind met with in England. In England, and on the Continent also, suitable brick for lining shafts is not very much more expensive than wood, and what little difference there is in expense of first cost is overweighed by the considerations of durability and maintenance, which are paramount in English and Continental construction. For these reasons the English and Continental coal deposits are usually worked through circular shafts which offer a high degree of resistance through arch action to the pressure from the surrounding material. These shafts when shallow and through firm strata are usually constructed of brick, and Fig. 1 shows what may be considered common practice in English shaft and heading construction and indicates the solid and fireproof construction of the shaft itself with the strength and stability of the gangway supports. The mines owned by the Hulton Colliery Co., in the Lancashire coal field have a total output of about 500,000 tons per annum, the bulk of which is raised from two shafts known as Atherton No. 3 and Atherton No. 4, which were sunk in 1900 and are each 24 feet in diameter in the clear. The shafts are lined with brick and the gangway approaches to shafts for the upper seams and at the bottom of both shafts are formed on a method now much approved in this coal field. Strong steel beams are set upon side walls of brick; two beams 12 inches wide by 7 inches deep are placed immediately under the last bricking ring of the shaft and are repeated at intervals of 12 inches, the single beams at more frequent intervals. These beams are covered to form lagging with 2-inch pine plank or ⅜-inch boiler plate. Side walls are good brickwork 3 to 4 feet wide. Where the floor is bad, the thickness at the bottom may increase to 5 feet and taper to 2 feet or 2 feet 6 inches at the top.

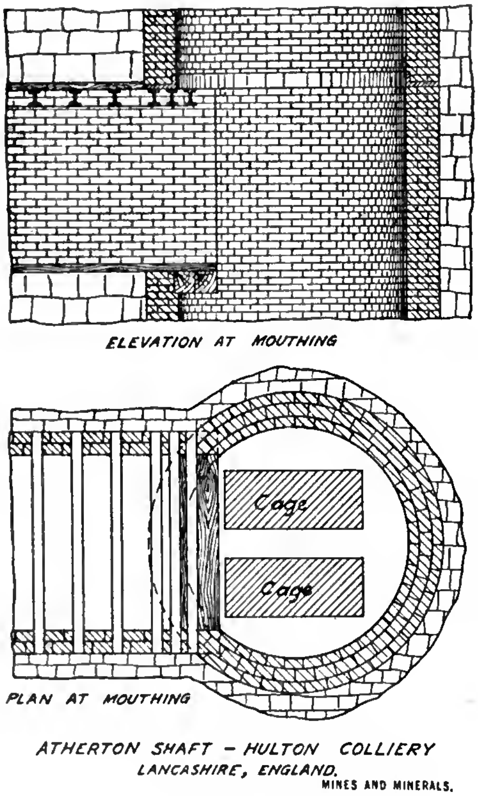


Fig. 1. Brick-Lined Shaft, Lancashire, England

Where the shafts pass through strata such as the magnesian limestone in the north of England, the Permian sandstone of the central countries and the chalk and green sand in the north of France and Westphalia, which carry large quantities of water, the main dependence is on cast-iron tubbing, which is built up in segmental framing rings piled upon each other throughout the entire depth of the water-bearing strata, this kind of lining being also peculiarly adapted to the circular form. The thickness of this lining is usually figured to resist hydrostatic pressure, the minimum thickness being about ¾ inch, and running to 4¾ inches, which is generally recognized as the maximum thickness which can safely be manufactured and readily handled. A new method of lining has been introduced in France which uses an iron ring ¾ inch thick lined inside with reinforced concrete of a thickness and strength to carry the pressure, the iron ring serving merely to protect

*Abstract from paper read before the Lake Superior Mining Institute, August, 1910.

the concrete from the water. In cases where iron rings 4¾ inches thick were formerly in use the saving in metal by the new system was about 20 tons per meter of shaft, the latter being 6 meters in diameter, the iron tubbing being the same thickness for all pressures. In the method of sinking shafts through water-bearing strata adopted by Kind & Chaudron, in Belgium and Germany, shafts have likewise been lined with wrought-iron plate tubes, and as rolled material has come into

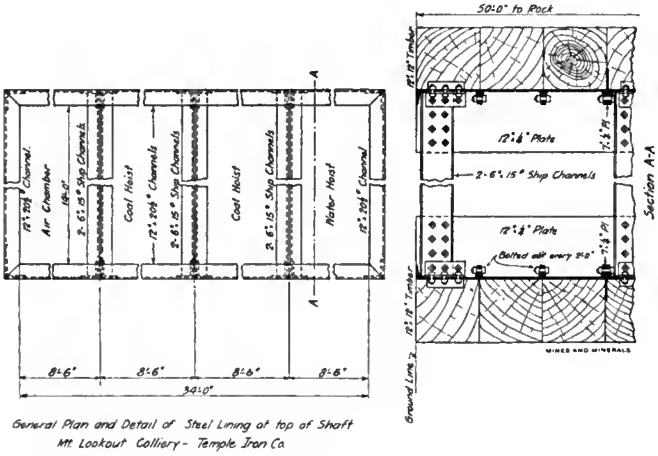


FIG. 2. STEEL-LINED SHAFT

larger use, the cast-iron tubbing has been more and more replaced by steel plates and shapes. So far back as 1894, for example, at the Grimsby iron ore mine in the Siegen district, Prussia, the shaft from the sixth to the seventh level was built of channels. At the Bockswiese mine, in the Harz Mountains, iron lining of I beams was experimentally adopted in place of masonry. The rings were 2 feet 6 inches apart, lagged with tees, the intermediate spaces being filled with stone, at a cost of 6 per cent. less than the ordinary masonry methods of construction.

If, however, the circular shaft is built large enough to carry the rectangular cages which are in almost universal use, there will always be an excess useless area requiring an excess of excavation as compared with the rectangular form, which, in turn, is best adapted for framing in wood or steel and which offers the largest useful area for a minimum amount of excavation. On the other hand, for the same area the perimeter of a circular shaft is less than that of a rectangular shaft. The useless area in a circular shaft may run from 25 per cent. to 45 per cent. and means that much additional expense in excavation and maintenance, which must be offset by cheapness of lining material in order to make it economical as compared with rectangular construction. The rectangular form of shaft conforms to the cages and skips which are used in hoisting men and materials, and represents, therefore, the minimum of excavation, and if properly divided by compartment separators of sufficient stiffness, offers no special difficulties from a technical standpoint. It is the form best adapted for framing in wood, and if framed in wood, steel may be substituted therefor with perfection, these materials being interchangeable and the variations in details insignificant. The cheapness of timber as a lining material has, therefore, made it more satisfactory from the standpoint of American mining practice and the chief advantages of the circular shaft in England and on the Continent do not have very much force under American conditions. The rectangular shaft may, therefore, be considered as the typical and best form of construction. The circular shaft has an advantage over a rectangular one in that it offers less rubbing surface and hence less friction to the air-currents and provides better ventilation. The mining laws, however, almost always require separate air-shafts and this consideration also is of little force.

The elliptical shaft is an endeavor to combine some of the advantages of the circular shaft and the rectangular; namely, strength and economy of space, but has not met with favor in England owing to the difficulty of keeping a deep shaft of that shape plumb and the obstacles presented to effective walling or tubbing. In Redmayne's discussion of this subject, however, the use of concrete seems to have been overlooked. The concrete-lined shaft is almost always elliptical in form and concrete seems to be the material best adapted in the construction of elliptical shafts, though it is possible to use brick. The difficulty in the use of cast-iron segmental lining is due to the fact that special sections would be required, whereas in the circular shaft the sections may be made all alike and interchangeable. In the elliptical shaft there is some elimination of useless space. It is safe to say that the elliptical shaft comes between the circular and the rectangular so far as availability of space is concerned. The elliptical shaft also takes advantage of arch action in the material itself so that the concrete may be made of less thickness than if rectangular. The concrete-lined shafts built in the United States have usually been sunk by the caisson method, the lining being built upon a steel cutting edge from the ground upwards and sunk into position by its own weight, whereas with the cast iron or wooden-lined shaft the extension of a shaft may proceed by building from the lower end downwards or by extensions downward in short sections successively built up on wooden or steel bearers. The caisson method of sinking is limited to about 120 feet. It is satisfactory through soft materials, but not through hard, and below the ground rock level, friction on the sides, the difficulties of excavation, etc., increase enormously.

The various uses of steel in connection with mine shafts may be illustrated by a few representative examples, which

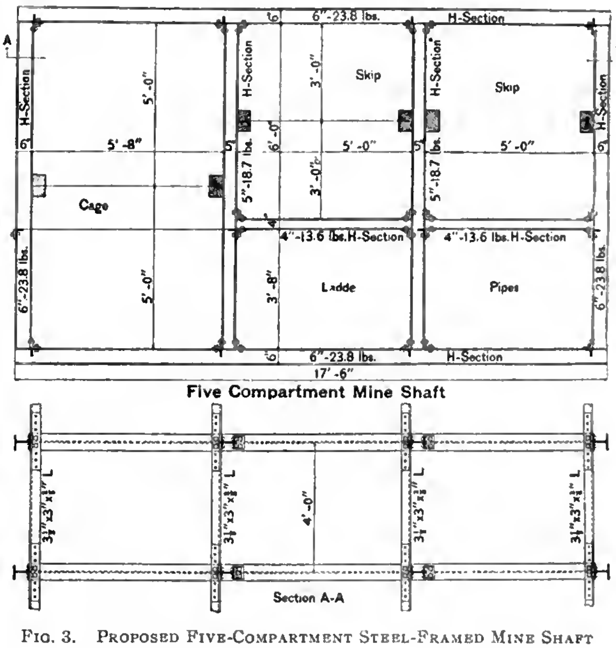


FIG. 3. PROPOSED FIVE-COMPARTMENT STEEL-FRAMED MINE SHAFT

will serve to bring out also its advantage in this form of construction. The writer will not say that there are no cast-iron lined mine shafts in the United States, but only that none such have come under his observation. If there are any, he will appreciate any information as to their whereabouts, dimensions, depths, etc.

A method of lining based on the same idea in steel is to be found at the Mt. Lookout colliery of the Temple Iron Co. Here the original timbered shaft was carried down to rock through quicksand, and when it had decayed it became neces-

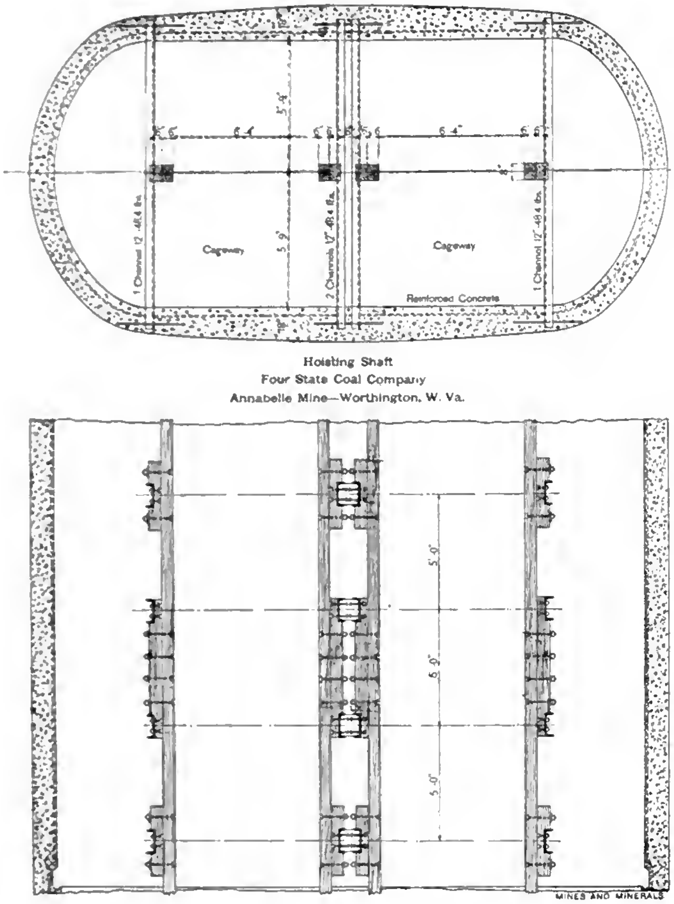


Fig. 4. Elliptical Concrete Hoisting Shaft, With Steel Ship-Building Channel Buntons

sary to put in some material which could be placed without disturbing the original structure and without cutting down the working space in the compartments for coal hoists, water hoists, etc. Fig. 2 shows the details of this construction in which 12-inch channels were used back to back, separated by anchor plates to catch in the wood of the original lining, and double 6-inch channels to take the place of the 10″ × 12″ buntons originally separating the compartments. The construction was very simple and could be made more so by the use of **H**-sections, which are better adapted to resist compressive stresses developed by the earth pressure than the channels, and of less weight as well.

The use of this form of construction in sinking new shafts would avoid the excavation of from 3 to 4 feet additional space around the shaft opening and very materially lessen the original cost of the construction. The lining, after the same fashion, could be constructed for a rectangular shaft by the use of steel beams spaced about 4 feet centers laid with webs horizontal, suspended one set from the other with heavy rods or angles and filled in between with concrete, in which case it would be necessary only to provide forms for the inner faces of the shaft and to have the beams punched with holes sufficiently large for pouring grout through them to fill up the small spaces between the flanges of the beam and the web. In this manner the strength of steel for shaft timbers can very readily be utilized in connection with the desirability of concrete as a permanent lining material.

The very first use of Friestedt interlocking channel-bar piling on a commercial scale was in sinking the shaft of the Johnson City and Big Muddy Coal and Mining Co., Johnson City, Ill., through quicksand 70 feet below the surface.* It has also been used by the Robert Gage Coal Co. at Bay City, Mich., in a similar construction after endeavors had been made to sink three times without success, and at an estimated saving of $35,000 if the piling had been used at the start. It has also been used by the Weaver Coal and Coke Co., at Du Quoin, Ill., Big Muddy Coal and Iron Co., at Murphysboro, Ill. etc. These installations were not all permanent. A noteworthy example of its use as a permanent lining of a rectangular shaft was at New Biggen, England, where piles of mild steel 80 feet long, built up in lengths of 20 and 30 feet, spliced so as to break joints, were driven by an ordinary pile driver through 18 feet of clay, 60 feet of running sand, and 2 feet into sandstone, with entire satisfaction and complete success. This is the greatest depth to which interlocking steel-sheet piling has ever been driven and exposed to view by excavation. The piling can be used either in rectangular or circular shafts, what is known as United States steel-sheet piling being peculiarly adapted to circular construction. The chief difficulty in respect to the use of steel-sheet piling, as also to any kind of piling, is the preservation of its verticality, especially where the clay or gravel contains large boulders. In the installations mentioned, however, the shafts remained plumb to all practical purposes without any special attention more than is necessary in good pile-driving practice. It may be added that great care is to be exercised in driving and instances have occurred, through inexperience, where it was attempted to drive the piling not to, but through, solid rock. With reasonable care in driving, however, this use of steel-sheet piling results in economical construction.

An ingenious system of forcing down piles in lining shafts has been successfully applied on several occasions in England and covered by letters patent taken out in the name of Mr. Charles Walker. It consists, briefly, in erecting a cylinder of tubbing, which is suspended from a surface platform, and

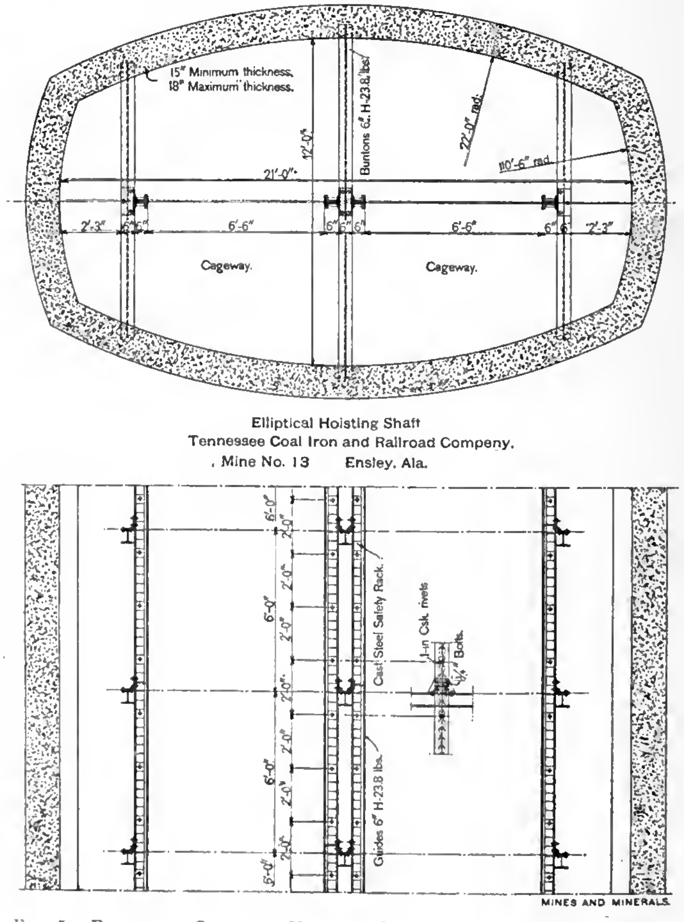


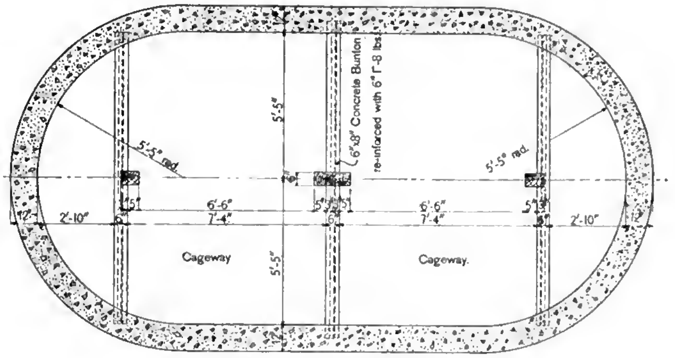
Fig. 5. Elliptical Concrete Hoisting Shaft, With **H** Section Buntons and Guides

*See Mines and Minerals, Vol. 23, page 72.

utilizing its weight to force down the circle of interlocking piles which are of special type and have riveted to them lugs or brackets against which pressure is exerted by hydraulic jacks. The verticality of the piling is assured by means of a movable or floating ring of tubbing into which they are grooved and which lies at the bottom of the shaft as it sinks into the silt. Full details of this form of construction may be found in Redmayne's "Modern Practice in Mining," Vol. 2, page 197.

The standard typical framing of rectangular shafts in the United States consists of the use of timber sets spaced 4 feet, more or less, centers, as conditions may determine, and lagged between with plank. In the mining districts of Michigan and Minnesota there are quite a number of mine shafts timbered with steel shaft sets lined with plank. These shaft sets are made along the lines of the timber set and are lowered into position by hangers as the excavation proceeds. They have been constructed with **T**-rail wall plates, 3-inch and 4-inch **I**-beam buntons and rail stuttles; also with 4″×5″ **T** wall plates reinforced by angles for proper bearing, 3-inch, 4-inch, and 6-inch double-channel buntons and rail stuttles; also with $4\frac{1}{2}''\times5\frac{1}{2}''\times8''$ Carnegie steel cross-tie wall plates with 4-inch **Z**-bar buntons, and 3-inch **I**-beam compartment dividers and rail stuttles. In the evolution of these sets the strength of steel has sometimes been overestimated by the mine foreman and sections of too small depth and too little strength substituted for the wooden sets previously in use. As an engineering proposition, after the proper size of wooden timber sets has been attained by experience, it is a very simple matter to substitute steel for their use on the basis of their relative strengths, and wherever this has been done, the substitution has been satisfactory.

There is an inclined steel-framed shaft constructed by the Newport Mining Co., at Ironwood, Mich., in which the wall



Elliptical Hoisting Shaft
Bunsen Coal Company
Clinton Coal Fields, near Clinton, Vermilion Co., Ind.

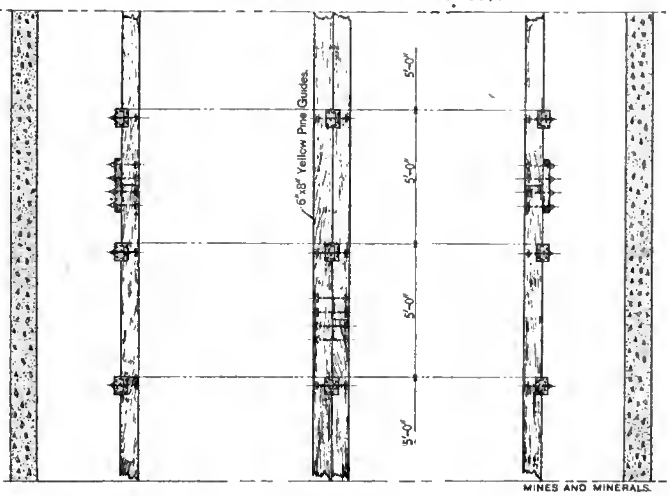
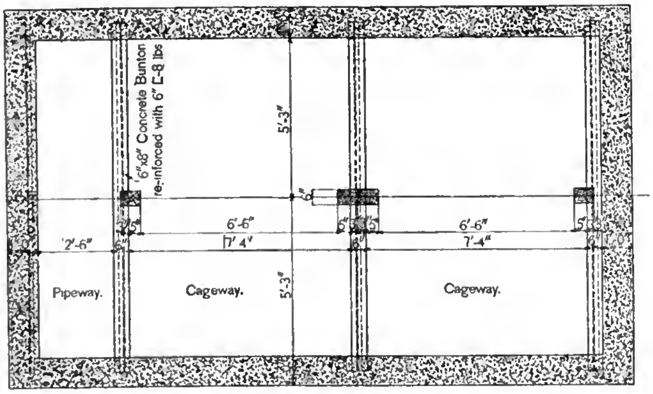


Fig. 7. Elliptical Concrete Hoisting Shaft, With Steel Channel Buntons

plates are made of 4″×3″ **T**'s, the stuttles are of angles, and the buntons of **H** sections. The use of the angle stuttle is to be recommended, as thereby special hanging rods may be eliminated in the process of sinking, and no material go into the shaft which does not remain in place permanently. With the angle stuttle also the load is divided equally on the bearers above and below, while with the rail stuttle sometimes used, the weight of the shaft comes on the lower bearers only. The **H**-section buntons are also in line with good engineering experience, while the use of **T**'s for wall plates is suitable for the installation, though as a general proposition this is not the proper form of section.

A more ideal form of construction is shown in Fig. 3, which represents a five-compartment mine shaft proposed for use in the mines of the Oliver Iron Mining Co. In this construction the broad bearing surfaces and high compressive strength of the **H** sections are utilized for wall plates, compartment dividers, and buntons, while the stuttles are made of $3\frac{1}{2}''\times3''\times\frac{3}{8}''$ angles. Ten-inch **I** beams were used for buntons instead of 5-inch **H** sections at the Hill mine, Sterling Siding, Minn., so as to allow the use of steel shaft sets in connection with wood, the buntons in the wooden sets being 10 in. × 10 in. and 10 in. × 12 in.; otherwise the 5-inch **H** sections would be much better adapted to the construction.

Recently quite a number of concrete-lined shafts have been recorded in the technical press as either completed or under construction. A representative example is to be seen in the shaft built by the Foundation Company for the Delaware, Lackawanna & Western Co., at the Woodward colliery, about a mile from Wilkes-Barre, where an open concrete caisson was sunk through 79 feet of the water-bearing soil overlying the bed rock. The concrete caisson was constructed with a cutting edge of $\frac{3}{4}$-inch plate 32 inches high, well reinforced with riveted angles, and the concrete in the walls of the shaft



Rectangular Hoisting Shaft.
Bunsen Coal Company
Clinton Coal Fields, near Clinton, Vermilion Co., Ind.

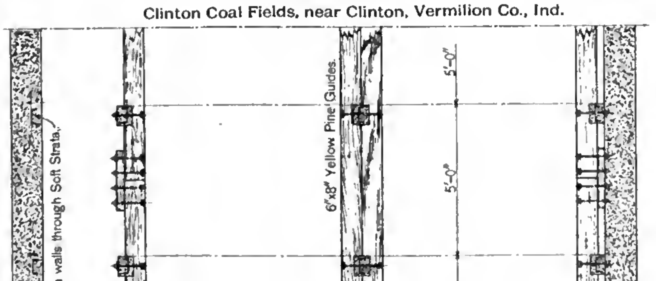


Fig. 6. Rectangular Concrete Hoisting Shaft, With Steel Channel Buntons

was likewise reinforced with 1-inch and 1¼-inch steel reinforcing rods amounting in weight to 140 tons. Instances of the same kind are the two concrete shafts built by the H. C. Frick Coke Co. at its new mines, Filbert, Brownsville, Pa.; and in the iron-mining regions may be cited the circular reinforced-concrete shaft sunk at the Morton mine, Hibbing, Minn., by the Foundation Company for the Todd-Stambaugh Co., Cleveland; also the shafts sunk by the same company for the Cleveland-Cliffs Iron Mining Co. at their Smith and Kidder mines near Princeton, Mich. The caisson at the Kidder shaft was built upon a fabricated steel shoe 24 feet outside diameter riveted to a steel-cone plate on the inside, the slope of which extended up to a height of 10 feet, making the working chamber on which were bolted steel sections 10 feet in diameter as the structure sunk.

The use of steel buntons and interior framing has much to recommend it in connection with the concrete-lined shaft,

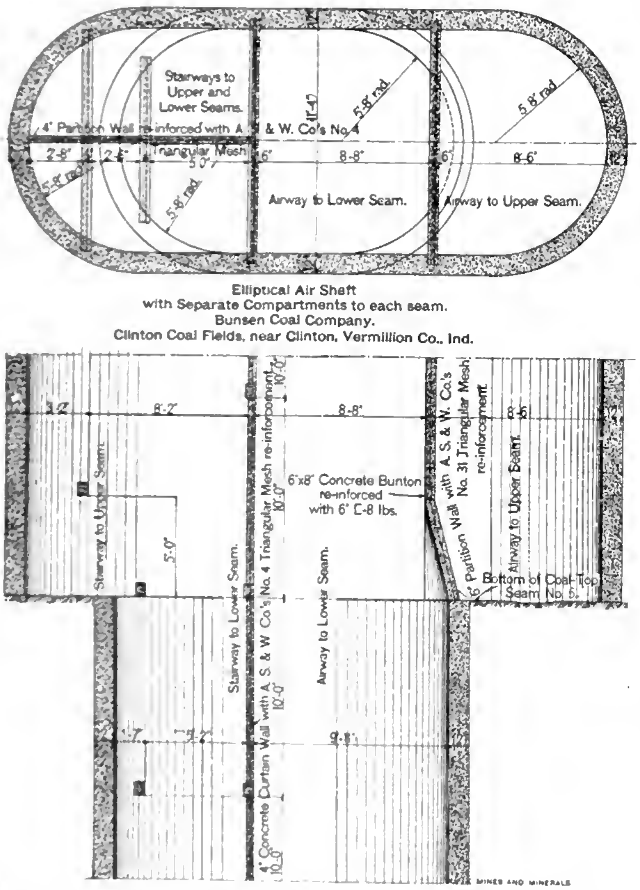


Fig. 8. Elliptical Concrete Air-Shaft, With Two Compartments, Steel Channel Buntons, and Wire Mesh Reinforcement

as it is strictly in line with observations already made as to the relative uses of these materials. Fig. 4 shows an elliptical concrete hoisting shaft under construction at Annabelle Mine, Worthington, W. Va., by the Four State Coal Co., a subsidiary of the Pittsburg-Buffalo Co. This shaft is 340 feet deep, lined with reinforced concrete. The buntons are made of 12-inch ship channels 48.4 pounds per running foot, anchored at their ends into the center of the shaft lining, but without special bearing plates. The two channels at the center of the shaft are fastened together by bolts and separators. The wooden guides are furnished in spliced lengths with broken joints, though this feature is not very plainly shown in the illustration.

Fig. 5 shows a plan and vertical cross-section of the elliptical hoisting shaft now under construction by the Tennessee Coal, Iron, and Railroad Co. at Pratt mine No. 13, near Ensley, Ala. In this case the concrete is massive and not reinforced, with a minimum thickness of 15 inches through firm material and 18 inches through soft. The buntons are made of 6-inch H sections weighing 23.8 pounds per foot, spaced 6 feet apart. The guides are likewise made of 6-inch H sections and carry bolted to them cast-steel safety racks, insuring great stiffness and rigidity in the structure, and in consequence a minimum of lost motion in operating the shaft. The shaft is located near a fault and the measures are much disturbed. It is, therefore, the intention to use steel beams for roof supports in the main gangways and tunnels around the shaft.

The Bunsen Coal Co., mining coal in the Clinton coal fields, has a shaft under construction near Clinton, Vermilion County, Ill. Two designs of this shaft are shown in Figs. 6 and 7, one rectangular and the other elliptical. The buntons in either case will be constructed of 6″×8″ concrete beams reinforced with a 6-inch 8-pound channel. Where the shafts pass through soft strata, it is the intention to reinforce the concrete at the ends also with 6-inch 8-pound channels. The thickness of the lining will be 6 inches through firm material and 12 inches through soft. The guides in both cases will be made of 6″×8″ yellow pine timbers securely bolted to the reinforced concrete buntons. It may be noted in illustration of the observations made as to the advantages of the rectangular shaft that the shaft shown in Fig. 6 has an inside area of 176 square feet with an area of excavation for a 6-inch thickness of lining of 226 square feet, while the inside area of the elliptical shaft shown in Fig. 7 is 211 square feet and the area of excavation for a 6-inch wall thickness 240 square feet.

A most excellent illustration is shown in Fig. 8, the plan and cross-section elevation of an elliptical air-shaft also under construction by the Bunsen Coal Co. in the Clinton coal fields. Two seams of coal are mined and the shaft has separate airways to each seam, as shown in the figure. The inside area of the upper shaft is 305 square feet, and of the lower shaft 165 square feet; the outside area for a 6-inch wall thickness, 341 feet for the upper shaft and 189 feet for the lower shaft. The shaft is about 400 to 500 feet deep, as also is the hoisting shaft just described. The partitions in the shaft are reinforced concrete built with the American Steel and Wire Co.'s No. 4 triangular mesh reinforcement on 6″×8″ concrete buntons reinforced with 6-inch 8-pound channels. These partitions, however, will be made 8 inches thick instead of 4 inches, as shown in the figure.

In the design of mine-shaft timbers the thing of prime importance is simplicity, which lessens the cost of fabrication and contributes to ease in erection. This simplicity is a thing which has always characterized the rectangular wooden shaft. It is a thing which just as well characterizes shaft timbers of steel. Bolts should always be used for erection rather than rivets and the hangers should be, preferably, of the angle type so as to add stiffness and facilitate plumbing. For bearers nothing better can be devised than the standard steel I beam.

An objection has been raised to the use of steel on account of its danger to corrosion. It need only be said that both the light of experience in this country and abroad and laboratory tests demonstrate amply that the subject is one of very little importance. Underground conditions are not nearly so severe on the steel as above ground conditions, and certainly painted material in the mines is not exposed to those alternating conditions of high and low temperatures, dryness, and wetness, strong light and darkness, with which above-ground construction has to contend and which are especially accelerative in the deterioration of a protective coating. The steel should be painted before it is placed in service. If the paint is well chosen and applied with care, there need be no fears as to its durability. Experience and theory all indicate that only the simplest means are necessary for the absolute guarantee of an extremely long life for steel in underground mining conditions.

The first coat applied at the shop should consist of a practically inhibitive pigment to prevent the inception of corrosion in the steel, and the second coat should be put on in

the field to protect the first from atmospheric and temperature conditions, to fill up thoroughly any voids that may occur therein, and to cover surfaces abraded in shipment. To meet these requirements a good grade of red lead or a natural oxide of iron well applied will be sufficient for the first coat. For a second, or excluding coat, there seems to be nothing better today than a first-class graphite.

It appears eminently appropriate that steel should be used in the timbering of iron mines, whether in horizontal headings or in shafts, as its use simply means putting back in finished form that which has already been dug out of the mine in order to dig out more. This is not exactly a parallel to the classical problem of a man pulling himself over a fence by his own boot straps, but is, beyond question, a step toward the solution of that problem

THE AEROLITH RESCUE APPARATUS

Written for Mines and Minerals

In the 1910 Proceedings of the Australian Institute of Mining Engineers there is an article on "Breathing Apparatus for Use in Mines," by R. T. Slee, to which he appends the table that follows, showing the conclusions of the British Royal Commission appointed to test breathing apparatus.

Results of Investigations on Use of Apparatus Carrying Liquid Air for Breathing Purposes

The common object of the various types of breathing apparatus is to enable the wearer to remain at work for varying periods of time in an atmosphere which is either deficient in oxygen or is poisonous, like the afterdamp of a coal mine explosion.

To ascertain the value attached to oxygen helmets as life savers in England, inquiries were made, and the following reply received: "I do not think it can be definitely said that the oxygen helmet has ever been the means yet of actually saving life, unless indeed the operations following the fire at Nova Scotia a year or two ago can be construed to this effect. Over here we are coming to regard the rescue apparatus in what I think has always been its proper light; viz., as the means of assisting in 'recovery work,' such as restoring ventilation, removing falls, and fighting mine fires."

Comparison of Breathing Apparatus

Apparatus	Weight in Pounds	Oxygen Capacity in Liters and Cubic Feet	Removal of Carbonic Acid	Supply of Oxygen	Air-tightness	Comfort and Convenience
Pneumatogen (Type II)	14½	163* (5.8)	Good	Poor	Good	Very good
Draeger........	36–39	251 (8.9)	Good	Good	Good	Bad
Shamrock	36	274 (9.7)	Bad	Good	Good	Good
Fleuss..........	31	228 (8.0)	Good	Good	Good	Good
Weg...........	about 30	150 (5.3)	Good	Good	Good	Good
Aerolitb........	22	?	Fair	Good	Good	Very good

*Maximum theoretical yield; 80 to 100 liters is the more probable figure in actual practice. (163 = liters, 5.8 = cubic feet.)

There are many on this side of the water who hold similar views and it has been suggested, owing to the fact that the operators encumbered with heavy helmets are unable to carry helpless men more than a short distance, that the name rescue apparatus be changed to "recovery apparatus," thereby more appropriately defining it without misleading the general public. Commenting on recovery apparatus, to be efficient, it must fulfil the following conditions:

1. It must be air-tight as regards the passage of the air from the outside to the inside. In atmosphere containing carbon dioxide a slight leak would not render the apparatus useless; but where irritating vapors, such as the smoke from burning sulphides, is met with, even the slightest leak must be avoided.

2. The oxygen supply must be ample, and so supplied that it is not checked by expiration.

3. Expiration and inspiration must not be retarded.

4. The carbon dioxide generated must be quickly and efficiently absorbed.

5. As salivation is much increased when wearing some types of breathing apparatus, effective saliva catchers should be provided in order to prevent contamination of the inspiratory air.

6. The apparatus must not be too heavy, must not impede the wearer's movements, nor impair his powers of observation. Its volume should be such as to render it easy of transport in the workings of a mine.

Modern types are constructed on the following principles:

(*a*) The expiratory air is purified by the use of a chemical which absorbs the carbon dioxide and moisture exhaled.

(*b*) The oxygen consumed in breathing is replaced from reservoirs containing the gas in a compressed or liquid form, or from chemicals liberating oxygen by means of chemical reactions taking place during the use of the apparatus.

Breathing apparatus as now used falls under one or other of the following types:

1. A helmet through which a constant current of air is passed, either from a pump or from the compressed air main being connected with the helmet by a length of hose, or from a reservoir of compressed air carried by the wearer.

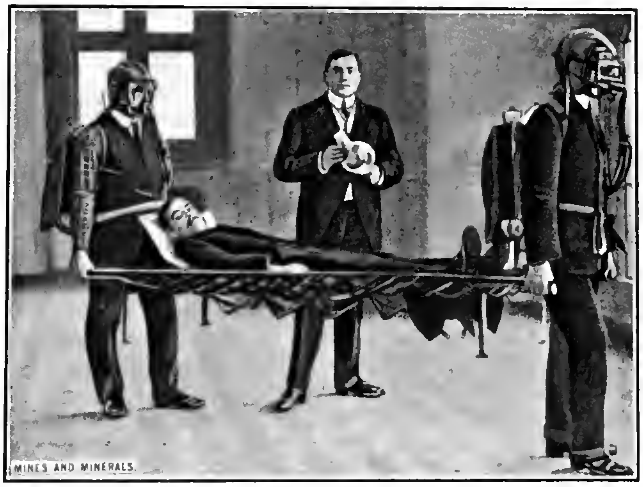


Fig. 1. The Aerolith

2. In this type the wearer breathes into and out of a bag provided with arrangements for absorbing the carbon dioxide and moisture contained in the exhaled air, while highly compressed oxygen from steel reservoirs passing through suitable mechanical arrangements replaces that consumed by the wearer, and as far as possible, assists the wearer in the act of inspiration.

3. In the third type the exhaled air passes through a cylinder containing peroxide of sodium and potassium. These compounds not only absorb carbon dioxide, but theoretically liberate sufficient oxygen to make up for that used by the wearer.

4. In this type the oxygen is derived from the evaporation of liquid air.

From the table it is seen that the Draeger apparatus affords an abundant supply of air, and that the carbon dioxide is absorbed thoroughly. On the other hand the apparatus is heavy and hung so that it is uncomfortable, tiring in use, and liable to catch on projecting timbers, etc. Also the heat is intense, which greatly detracts from the comfort, convenience, and utility of the wearer. Recognizing that vigorous action and comfort were requisites in recovery work, the aerolith has been investigated. Mr. Guy Simmonds, chief of the Durham and Northumberland Collieries Fire and Rescue Brigade, of

Elswick, Newcastle-upon-Tyne, writes as follows: "The reason we adopted the aerolith apparatus was that we found it less than half the weight of the oxygen apparatus, and the air supplied remains cool till the very end, whereas the air in the oxygen apparatus becomes very hot and unpleasant after it has been worn for a time. We have been experimenting for the last 6 months with this apparatus and find that the men are able to use it when carrying out heavy work with very little discomfort. One of the greatest difficulties we have been faced with is the transport of liquid air in glass thermos bottles. This we now think has been got over by using metal containers. I would further point out that the manufacture and storage of liquid air renders it more suitable for this rescue station than any of the others, as it is proposed in this district to use permanent men quartered at one or more stations and not attempt to keep rescue apparatus at every pit. In other districts where the supply of rescue apparatus is contemplated for every colliery the liquid air system might not be so suitable."

"The liquid air plant, which cost about £800, is capable of producing 10 liters of air an hour. With the large stock of air on hand, this plant will enable us to keep 20 aeroliths in constant

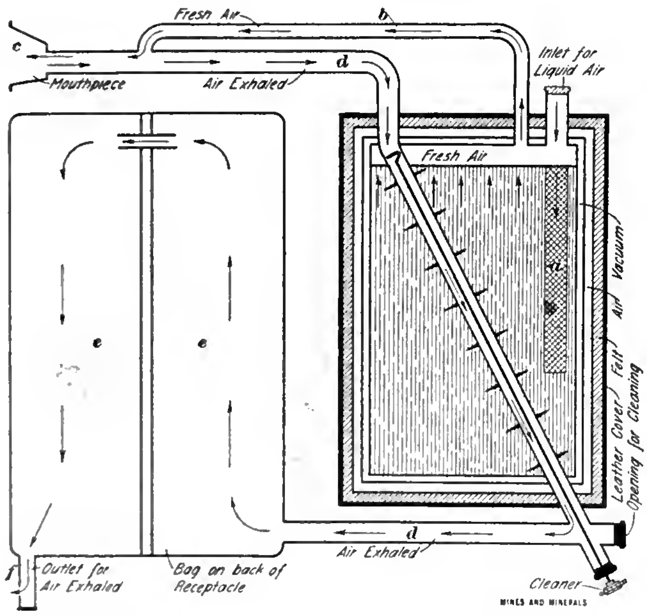


Fig. 2. Diagram Showing Circulation of Air in Aerolith

use for about 48 hours, and in that time we shall be able to get manufacturers to supply us with air in unlimited quantities."

At the central rescue station of the coal mines at Witkawitz, Austria, there are two liquid air plants, one for a normal capacity of 10 pounds of air, which may however be increased to 20 pounds per hour, and the other having a capacity of 50 liters of liquid air per hour. The smaller plant was started in May, 1907, the cost exclusive of buildings being $7,500. The larger plant was installed at a cost of $15,000, exclusive of buildings. The size of the liquid air plant is chosen according to the number of apparatuses dependent on any one center The air is compressed up to about 120 atmospheres or 1,760 pounds per square inch and purified, then through sudden expansion cooled below the critical point, thus turning it into liquid which is stored in vacuum vessels. The vacuum receptacles form such excellent insulators that the loss through evaporation is reduced to about 5 per cent. per day. A stock of air for emergency must of course be kept, but as 1 gallon of liquid air costs less than 25 cents the loss in evaporation amounts to less than 1¼ cents per gallon per day. Assuming that at a certain station the men practice 40 times weekly, and at each practice of 2 hours 1 gallon of liquid air is used, then the cost would be $10 per week. It is probable that a similar number of practices per week with an oxygen helmet would cost nearly five times the money. The composition of the gases given off by liquid air varies in a particularly convenient manner in aeroliths; for instance, the nitrogen boils off more easily than the oxygen, hence the liquid air that has been kept some time before use is richer in oxygen, and the gas received by the users toward the end of a 2-hours practice contains relatively more oxygen than at the beginning.

Plants required for making liquid air need little attention; their product is not explosive and, as used in connection with the aerolith, is harmless. Presuming there are very few in this country who have even a general idea of the appearance of the aerolith, it is shown in Fig. 1 and described by reference to Fig. 2. Fig. 1 shows the aerolith knapsack, as worn in actual work, but the headpiece is now a mask helmet entirely finished in asbestos that covers the head by means of an asbestos flap. The air-tight joint is around the face only, and is formed by a pneumatic cushion attached to an adjustable frame that will fit any type and shape of face. The eyeglasses are large and square and inside cleaners are provided that are operated from the outside. The nickel box shown in Fig. 2 is in the form of a knapsack and is loosely packed with asbestos, besides being well insulated from the outer atmosphere by felt and leather. Into this knapsack 1 gallon of liquid air *a* is poured through the inlet. After charging and closing the inlet the only outlet for the gas produced by the evaporating liquid air is through pipe *b* that opens in an air conduit extending about the knapsack to allow the air to assume a normally cool temperature, then terminates in the mouthpiece *c*, which connects with the helmet. Liquid air has a normal temperature of 180° below zero, and on evaporation expands between 700 and 800 times in volume. The gas given off is cool in temperature and contains about 60 per cent. oxygen. The exhaled air goes through *c* but is unable to enter *b* owing to a constant flow from this pipe, therefore it goes into pipe *d* which crosses the liquid-air chamber before entering the divers-cloth breathing bags *e* and then, as shown by the arrows, to the escape *f*. On the way from *d* to *e* the exhaled air helps to control the air supply by communicating its heat to the liquid air in the box, thus increasing evaporation, in other words, the harder the wearer works the more air he needs and this he receives. Any surplus of evaporated air joins this exhaust and carries its oxygen into the breathing bags. By attaching a mouthpiece to the bags another man, not wearing an aerolith, can be led for a considerable time through poisonous atmospheres.

The knapsack is made of nickel, the outer covering of leather, the breathing bags of divers cloth, and is 15 in.×11 in.×5½ in. over all. When fully charged the outfit weighs 23 pounds, but as the liquid air evaporates, the weight becomes less until when exhausted it weighs 13 pounds. There are no valves, chemicals, or regenerators, which greatly adds to the simplicity of the breathing apparatus, and from the fact that cool fresh air is supplied the operator, there seems to be no hindrance to the aerolith's introduction except the cost of the liquid air plant. The listed price of the breathing apparatus is $115 complete, which is much less than some of the others, and this added to the other good qualities suggests that it receive more consideration in the United States than it has in the past.

GAS IN VERY HOT FLAMES

The products of combustion of gases in very hot flames are not always identical with those generally met with. Thus it has been shown that in the presence of an excess of oxygen the combustion of carbon monoxide gives ozone; under the same conditions the hydrogen flame also contains ozone and a little nitrous oxide; and further, the acetylene flame contains almost 4 per cent. of this same nitrous oxide. The temperatures attained during the combustion of these gases, under the above conditions, are exceedingly high, about 2,600° C. for carbon monoxide, 2,800° C. for hydrogen, and 3,000° C. for acetylene

THE MARIANNA COAL MINES

Written for Mines and Minerals, by Hartley M. Phelps

A Modern Coal Mine Planned for a Daily Output of 5000 Tons From One Tipple

Coal mining after the most improved and modern methods may be seen at the Marianna mines of the Pittsburg-Buffalo Co., in Southwestern Pennsylvania. Practically every labor-saving device of the highest efficiency has been installed in these most interesting of workings; the machinery above ground as well as below is of the best, and no expense has been spared by the enterprising owners to make Marianna what it is, a model coal mine.

Not only is the plant constructed along the latest scientific lines but the town of Marianna must take rank as one of the finest coal miners' settlements in this country. The Jones family, chief owners of the company, are noted for their somewhat altruistic tendencies, having spent hundreds of thousands of dollars to provide many conveniences and comforts and even luxuries, for their employes. Actual uplift work is being done at this mine, The officers of the Pittsburg-Buffalo Co. are John H. Jones, President; Thomas P. Jones, Vice-President; David G. Jones, General Manager; and James Jones, father of the three, Chairman of the Board of Directors.

The Marianna mine taps a vast tract of some 800 square miles underlaid with Pittsburg, Freeport, and Kittanning coal beds, also Waynesburg and Sewickley coal beds of marketable thickness, this latter forming the future fuel supply of Pittsburg. With the exception of several mines on the Monongahela River, Marianna is the only opening in the field. It is situated on Ten-Mile Creek, a branch of the Monongahela and is 6 miles from Millsboro at the junction of the two waterways. The railroad from Pittsburg reaching Millsboro and Marianna is the Pittsburg, Virginia & Charlestown, or the Monongahela branch of the Pennsylvania Railroad. It is about 56 miles by this line from Pittsburg to the mines.

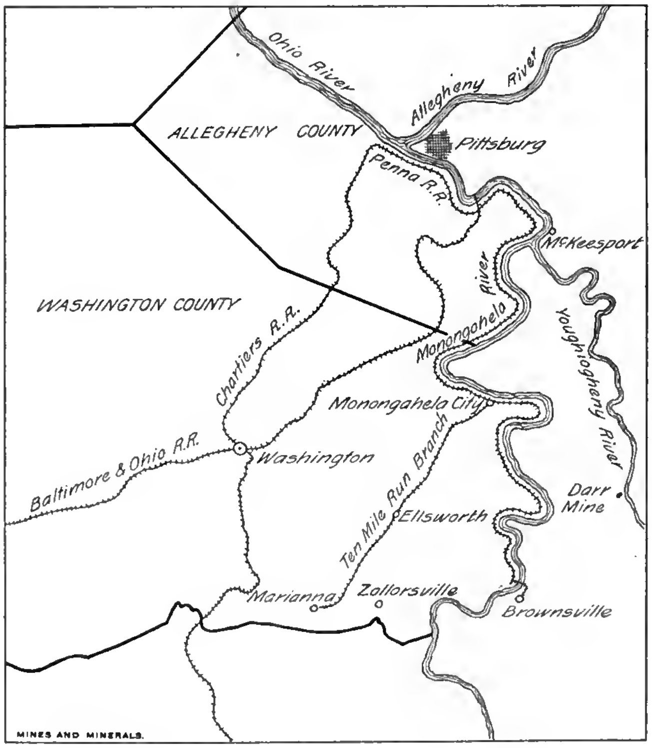


Fig. 1. Map Showing Location of Marianna

The Pittsburg coal bed at Marianna is 440 feet below the surface, is a little more than 6 feet in thickness, and of unusually high grade as gas and coking coal. The land containing the Pittsburg coal held by the company in this field totals more than 6,000 acres, and it is conservatively estimated to contain 50,000,000 tons. The Waynesburg outcropping, largely through southern Washington County, is more than 5 feet thick and makes a good grade of domestic coal. It lies about 70 feet below the surface in the Rachel shaft at Marianna. The Upper Freeport lies about 600 feet below the Pittsburg seam and measures about 5 or 6 feet in thickness. The Lower Freeport is not shown by the oil-well logs, but the two Kittannings have been located, one about 110 feet below the Upper Freeport and the other about 40 feet deeper.

The mine-run coal of the Pittsburg bed has a calorific value of about 14,000 British thermal units per pound of dry coal and cokes with less than 1 per cent. sulphur without washing, while the slack cokes with under 1 per cent. when washed

The Pittsburg-Buffalo Co., as stated, in order to operate economically so large an area, has installed some of the most modern machinery. It is said that 3 cents per ton saved in operation more than covers the entire cost of this equipment. Coal mining is now costing the company less per ton of annual capacity than the average mine in the Pittsburg district, the cost being less than $1 per ton of annual capacity, with amortization less than 3 cents per ton of coal production. The full significance of this can be appreciated when it is borne in mind that many mines having an annual capacity of from 50,000 to 100,000 tons per annum and an area of only 100 acres of coal have equipment costing as much as $100,000 or from 10 to 15 cents per ton of coal produced. The production of the Marianna mine is 5,000 tons a day, or 1,500,000 tons for a year of 300 working days, as against the average daily production of 300 tons for all mines in the United States. The magnitude of the work being done may be realized when it is stated that as much as 18 tons of coal per minute have actually been hoisted at this mine, a rate of 1,000 tons per hour.

There are two main or hoisting shafts 22 ft. × 33 ft. outside the lagging and about 450 feet deep. They were sunk about 4,800 feet apart, and are so located that the coal from the entire field can be delivered to them with the grades almost entirely in favor of the loads. Both shafts have cageways 7 ft. × 20 ft., so two cars may be hoisted on each cage in tandem. They are also provided with an airway 11 ft. × 20 ft. and with stairways from the surface to the coal. One of the shafts, the Rachel, Fig. 2, is the main hoisting shaft and here the cars in tandem holding three tons each are lifted.

The hoisting engine at the main shaft is of an original build having a lever-operated brake on each side of the drum placed in such a manner that either or both brakes may be applied, this method being much safer than the single hand brake usually employed and also more powerful. There is also a steam brake and a steam reverse. Among the other improvements is a method of taking up slack rope and thus keeping the relative positions of the cage so adjusted that there will be no delay in making landings.

The engines are figured to start an unbalanced load of 22 tons divided as follows: Cage, 11 tons; two cars empty, 4 tons; 6 tons of coal; rope, 1 ton. In order to preclude an overwind the hoister is provided with a safety stop. The engine cylinder measures 42 in. × 60 in., and the conical drums 10 ft. × 12 ft. It is possible with this equipment to lower into the mine any piece of machinery that may be necessary without taking it apart, a great advantage. The head-frame at the supply shaft, shown in Fig. 3, is of steel and designed for a load of 20,000 pounds, a factor of safety of 15 being used.

In addition to the two large shafts, known as the Blanche and Rachel, there is a small one, the Agnes, 12¾ ft. × 24 ft., used for taking men in and out of the mine and for handling slate and supplies. By thus relieving the other shafts of such

work, supplies can be lowered in daylight. A fourth shaft, the Miller, 16 feet in diameter and brick lined, is so located as to ventilate the entire mine, thus doing away with brattices and partitions in the shaft, and obviating the air leakage inevitable with such partitions, and at the same time making the air-current absolutely independent of the motion of the cages. As the mines make much water, a Jeanesville pump with a capacity of 1,000,000 gallons in 24 hours has been installed.

Fig. 2. Largest Bituminous Coal Tipple in The World, at Rachel Shaft

In the underground workings is to be seen the same attention to labor-saving devices and methods as characterizes the outside equipment. The main shaft bottom has a capacity of 250 loaded and as many empty mine cars and is arched for 1,100 feet on the rise, and 600 feet on the dip side of the shaft. By means of a chain haul, the loaded cars are fed on to the cages, and by the use of a rope haul the empties are returned to the shaft bottom tracks. The mine cars are of steel and designed with wooden bottoms, lift doors, spring draw-heads and link couplings. The sides are stiffened by bars across the top and by pressed steel stiffeners on the outside. The wheels are of special design and each car is equipped with a powerful hand brake having iron shoes.

Steel track is in all workings and mule haulage is entirely abandoned. On the main haulage road compressed-air locomotives of 15 and 19 tons weight deliver the coal to the shaft bottom. All coal is gathered by means of 6-ton compound gathering locomotives.

Four entries diverge into the mines from the foot of the air-shaft, which has an area of 201 square feet. The main entries extend to the boundary of the property, two for loaded cars, two for empties, and two for return air-courses. The cross-entries are laid out in pairs and entries have been driven from these, the mine being developed in panels. Not more than 50 men work in each panel and each panel has its separate air split. The entries are 9 feet wide and 7 feet high in the clear. All stoppings are of brick or concrete, brattice being carried to within 12 feet of the face and overcasts are built in the rock above the coal. All foul air and gas goes directly to the air-shaft and does not come in contact with the other faces.*

The method of mining is by room and pillar, the semi-retreating procedure being used, butt entries having been driven to the limit of the panel. One-half of the rooms will be turned back from these butt entries at the face end, and these rooms finished, the ribs will be drawn before the remaining half are started. The rooms have a length of 225 feet, but as the pillar-drawing proceeds they are continued across the air-course above and to the next entry, giving a total length of 265 feet.

The entire area to be operated from the two shafts is about 10,000 feet each way from the shafts along the main entries and about 3,000 feet on each side of the main entries. This area is to be divided into four parts, providing for five face entries in each section of the mine from which entries the butt entries will be driven about 1,800 feet long. Each section has its own independent haulage system, making practically four mines.

The tipple, washery, and coke ovens, being correlated in their workings, had to be situated with this end in view, the tipple, Fig. 2, furnishing slack to the washery, from whence the washed coal may be delivered to the ovens, while the bone coal is delivered to the boilers. The boiler house had to be located so waste heat from the ovens could be utilized as shown in Fig. 6.

The tipple at the main hoisting shaft is the largest bituminous mechanism of this kind in the world. It loads 5,000 tons daily with the least breakage in handling, removes impurities from the coal, and is so built that there is the least possible delay by reason of breakdowns. The coal is dumped over four cross-over dumps of heavy pattern, on to 1¼-inch bar screens, the empty cars going from the dumps to the kick-backs and thence to the transfers where they are replaced on the cages by pushers and transfer cars of the company's own design. The dumps operate in the ordinary way but are more quickly returned, being equipped with a pull-back comprising a steam cylinder piston attached by chains to the rear lever of the dump.

* Mines and Minerals, Vol. XXIX, page 272.

Fig. 3. Supply Shaft, Marianna Mine

After going over the screen the lump coal is delivered to the picking table. The impurities removed are sent to the crushing plant where they are made serviceable for use under the boilers, while the coal is delivered into railroad cars with minimum breakage. The picking tables are about 80 feet long, 6 feet wide, and run at a speed of 40 feet a minute. The slack coal falls through the screen on to conveyers whence it is elevated to revolving screens which separate the nut from the slack. Then the nut and slack go by separate conveyers to the shipping bins, the washer, or to the small bins over each picking table. With this tipple it is possible to ship the following grades of coal: Mine run, 3-inch lump, 1¼-inch lump, ¾-inch lump, 3-inch mine run, 3-inch nut, nut and slack, and slack.

The machinery of the tipple being in duplicate the chances of a breakdown throwing it idle are reduced to a minimum. Minor repairs to mine cars are made at the tipple, thus gaining time by not sending them to the shop. Two forges for sharpening heavier bits of the mining machines are located here also. The electric motors on the tipple are built dust-proof and have fans arranged so as to keep the working parts free from dust with clean air from the outside. The roofing and siding of the tipple are of asbestos-coated corrugated iron.

Near the Rachel tipple is a Connellsville fan 35 ft.×8 ft., direct-connected to two Connellsville 26″ × 48″ engines. A new departure comprises two clutches on the main shaft of the fan enabling the disconnecting of one engine and the coupling up of the other without change in speed of the fan. Thus if one engine should be idle it is not necessary to shut down the mine. The fan at 100 per cent. volumetric efficiency delivers 700,000 feet of air at an estimated water gauge of 6 inches. In order that repairs may be made quickly and cheaply, small cranes have been installed over each engine as in all other engine rooms at Marianna. The fan may be quickly reversed and air-locks are provided so the fan chamber may be entered while the fan is running.

The fine-coal washer of the Luhrig type adjoins the tipple and has a capacity of 100 tons per hour. It has 16 primary jigs and four secondary jigs which rewash the refuse from the primary jigs, thus recovering the bone coal. The washer machinery, like that of the tipple, is duplicate, and as much of the conveying machinery for the two is interchangeable a minimum stock of repair parts is necessary. One feature is the 1,500-ton reinforced concrete bin holding three days' supply of washed coal for the ovens in case of idle mines. It can be closed in underneath in winter to prevent the freezing of the slack in the bins. The gates from which the larries are charged are steam-jacketed; the larries can be run in under the bin on cold nights, the doors closed, the steam turned in, and everything put in shape to resume work unimpeded in the morning. Near the washer is a concrete bin from which refuse and bone coal is fed on to a conveyer which discharges into bins over the stokers in the boiler house.

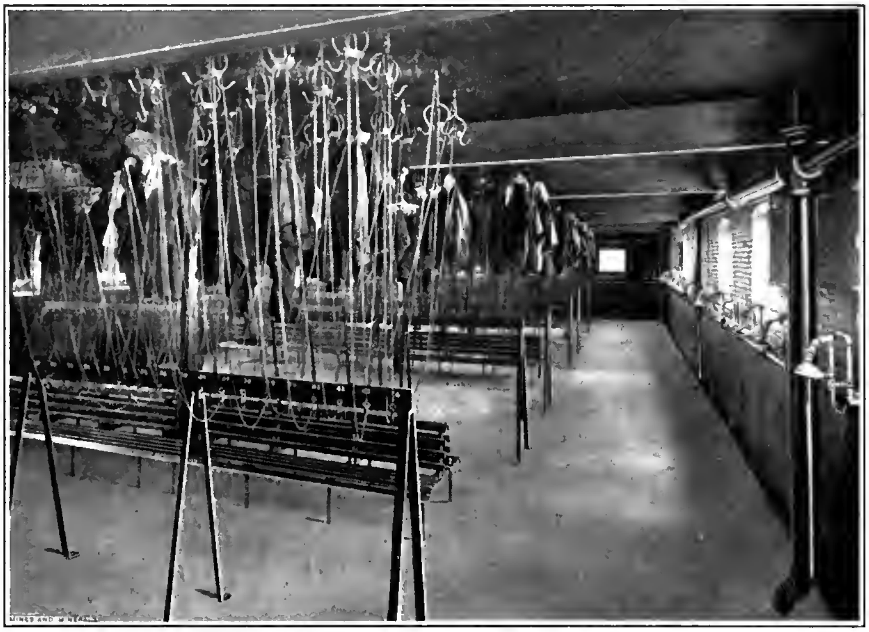


FIG. 5. INTERIOR OF BATH-HOUSE, MARIANNA MINE

The coke ovens are of the beehive type, 12 ft. 6 in.×8 ft. 6 in., equipped with the Kearns patent waste heat flue by which the waste heat is taken from the trunnel head instead of the crown, thus saving two bushels of coke per ton. The larries, having a 75-horsepower railroad motor on each axle, have a capacity of 8 tons of mine-run coal. The oven yard, 30 feet wide, is so arranged that coke-drawing machines can be installed. Provision is also made for loading box cars.

The boiler plant comprises Stirling boilers, 3,619 horsepower, old rating, of which four are equipped with chain grates and three with underfeed stokers, the latter having forced draft and being better adapted to the varying loads caused by the hoisting engine. The chain grate boilers are also fitted with waste heat flues so that coke-oven heat may be used under the boilers to generate steam. The overhead bins have a capacity of about 280 tons or 60 hours supply of coal with full load. It is the intention to install a pneumatic ash disposal plant.

The boiler house, like all Marianna build-

FIG. 4. AMUSEMENT HALL, MARIANNA MINE

ings, is of No. 2 paving brick with steel trusses and cement tile, built in such a manner that no repairs will be needed during the life of the plant.

The power building is back of and on the hill above the boiler house and in order to secure dry steam, there is a large tunnel connecting its basement with the firing floor of the boiler house. It is built of brick, concrete and steel throughout, and has also a 20-ton traveling crane with which all the engines were erected and which is conveniently used in packing, replacing gaskets, and making small repairs. The power house contains the following: One 500-kilowatt Westinghouse generator connected to a 20″×48″ Mesta Corliss engine. One Ingersoll-Rand Imperial compressor 20 in.×34 in.×32 in.×20 in.×30 in. and one Mesta 22″ × 36″ × 20″ × 34″ × 48″ cross-compound compressor, each with a displacement of 3,500 cubic feet of free air per minute, furnish all the low-pressure air required to cut the coal in the mine.

One Mesta and one Norwalk air compressor each 22 in. ×36 in.×28 in.×16 in.×9½ in.×5¼ in.×36 in., with a displacement of 2,200 cubic feet of free air per minute, furnish all the air required by the twenty 6-ton and one 19-ton Porter compound air locomotives which at present haul all coal produced at Marianna. These two high-pressure machines are the largest yet built in this country for this pressure. They are not equipped with receivers as the line itself is sufficiently large for this purpose.

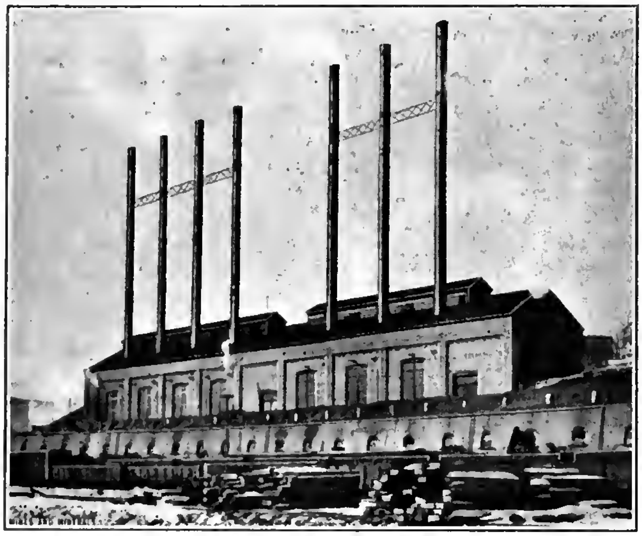


Fig. 6. Power Plant, Marianna Mine

The machine shop is near the Agnes shaft and is connected with it by tracks. In reality it is a collection of shops under one roof, including a carpenter shop, blacksmith shop, car repair shop, brass foundry, and four large storerooms. It is 182 feet long by 66 feet wide and two stories high. The ground floor is of concrete, the second floor of reinforced concrete, the storerooms being separated from the shops by fireproof walls with doors. There is one 25-ton electric traveling crane for hoisting railroad cars, three small traveling cranes and a jib crane. The blacksmith and forge department is large enough to make all manner of forgings for railroad cars, mine cars, and mine machinery.

The company has built about 400 houses in the town of Marianna, most of which are of yellow brick; the others frame. They vary in size from 4 to 14 rooms; are provided with hot and cold water, electric light and, if the tenants desire, with natural gas. They rent for $1.75 a room per month. Fireproof construction has been used where possible. Each house has a garden, the lot being enclosed with a substantial fence.

The streets are 50 feet wide, shaded and have concrete walks throughout, but the streets themselves are not paved. In each lot in the town are fruit trees. There is a sanitary sewage system, and drinking water is derived from artesian wells. The view from the town over the hills of Washington and Greene counties is beautiful.

There is an 8-room public school building under construction, and on the hill is a reservoir of 2,000,000 gallons capacity.

Two of the features at this interesting mine with its 1,500 employes are the bath house and amusement hall, both erected by the company. The former, whose interior is shown in Fig. 5, is modeled after the most advanced German shower or rain baths, is a brick structure 125 feet by 42 feet, four stories high, and is arranged so that men coming out of the mine are landed on a covered bridge leading to the fourth floor, which constitutes the assembly room. Here is the rack containing the safety lamps. Men entering the mine receive lamps prepared by the attendants, and men operating machines receive bits and oil for operating the machines. As the men come out of the mines they are checked by the timekeeper and pass down stairs to the bath rooms, their soiled mine clothing being suspended from the ceiling and ventilated while they don clean dry clothes.

The baths are of the shower kind, as a plunge or pool is dangerous for spreading disease. The inclined shower bath can be regulated for men of various heights and is the best form of hygienic cleansing bath. There are 99 shower baths for the men and two separate bath rooms for boys employed as slate pickers, etc. on the outside. There are also baths for officials and visitors in separate rooms. There is an emergency hospital on the ground floor. The lower floor is devoted entirely to the needs of the men working above ground, its equipment being identical with the two floors immediately above.

The amusement hall, shown in Fig. 4, is 106 by 62 feet and was built for the sole purpose of furnishing clean decent entertainment as well as instruction for the people of Marianna in the evening. The interior wall surfaces are painted in pleasing colors instead of being plastered. There is a theater equipped for vaudeville and moving-picture shows, with an elevated stage and dressing rooms, and seating capacity of about 275. There is also a billiard room. On the same floor, the first, is a drug store and soda fountain, while in the basement are eight standard bowling alleys, a restaurant, barber shop, and lavatories. The third floor is largely given over to a roller skating rink and dance hall, but there are also reading and lecture rooms, in the latter of which it is proposed to give instruction in mining.

The Pittsburg-Buffalo Company evidently believes that a happy workman above ground makes a contented one below ground. At least it would seem so in arranging the amusement hall. It is a pity the lectures cannot be made compulsive, since these men working in a naturally hazardous occupation and in virgin territory, more than usually gaseous, cannot be too much alive to the exigencies surrounding their own craft.

COAL AND OTHER EXPORTS

Coal and coke experts from the United States in 1910 aggregated $45,000,000 in value, and in addition to this more than $20,000,000 worth was supplied to vessels engaged in the foreign trade, making a total of $65,000,000 worth of coal passing out of the United States in the calendar year 1910. In 1900 the total value of coal and coke exported (aside from that supplied to vessels in the foreign trade) was about $23,000,000; 20 years ago, about $7,000,000; and 30 years ago, $2,000,000. Thus the value of coal exported to foreign countries in 1910 is practically double that of 10 years ago, 6 times as much as that 20 years ago, and 23 times as much as that 30 years ago. The quantity exported has grown from a little over a half-million tons in 1880, and about 2,000,000 in 1890, to approximately 8,000,000 tons in 1900 and 14,000,000 in 1910, aside from the 6,000,000 tons supplied to vessels engaged in the foreign trade.—*Department of Commerce and Labor.*

GEOLOGY OF HERRIN QUADRANGLE

*By T. E. Savage**

The Herrin coal district is a rectangle in shape, approximately 250 square miles in extent, situated in the southern portion of the state, at the southwest corner of the Illinois coal field. It lies roughly between the towns of Carbondale and Marion on the south, and Duquoin and Benton on the north; and includes portions of four counties, as follows: About 65 square miles in the northwest part of Williamson, 78 square miles in southwest Franklin, 22 square miles in southeast Perry, and 84 square miles in the northeast portion of Jackson.

Location and Importance of the Area. Correlation of the Seams. Analyses

The region has been mapped topographically by the State Geological Survey and the United States Geological Survey,* in cooperation. The district receives its name from the town of Herrin, which is a rapidly developing coal center. The Herrin quadrangle joins the West Frankfort on the west, and the coal field is a westward continuation of the Johnston City field, in Williamson County. There are operated within this area 32 coal mines, the output of which for the year ending July 1, 1908, was nearly 5,000,000 tons. Notwithstanding this large production, the development of the coal resources of this region has scarcely passed beyond the initial stage.

With the exception of about 40 square miles in the south and southwest portions, the area is underlain by two seams of coal, respectively, about 9 and 4 feet in thickness, within easy working distance below the surface. The quality of the coal in both of these seams is good, and the mining conditions in both are extremely favorable. A good market for the output is afforded by the presence of several lines of railroad within the quadrangle.

The range of surface relief within the quadrangle is about 230 feet, although the altitude over more than four-fifths of the area is included between 380 and 460 feet above the sea. The lowest point is about 3 miles southwest of De Soto, where the Big Muddy River leaves the quadrangle. The highest points are reached in the hills 2 miles northeast of Mulkeytown, which rise slightly above 540 feet.

The surface drainage of the area is entirely accomplished by Big Muddy River and its branches.

The rocks of this district consist of a mantle of loose surficial materials, overlying more consolidated beds of hard rock.

Within the quadrangle the surficial materials have an average thickness of about 40 feet. These deposits are deep over the valleys of the early Pleistocene streams whose ancient channels were broad and considerably deeper than the present valleys, as is shown where the Big Muddy and Little Muddy rivers carved channels in the strata, 75 to 90 feet below the present river beds:

Over the uplands the surficial materials consist of fine-grained silt, 18 inches to 8 or 10 feet in thickness, underlain by pebbly clay or till to a varying depth of from 8 to 25 feet. Along the larger water courses where these mantle materials are thickest, they are composed largely of fluvial, or fine, glacio-fluvial materials. A bed of quicksand, 6 to 25 feet in thickness, is often found near the base of these deposits. This sand runs easily, and has proven a serious obstacle in sinking coal shafts at a number of points in the vicinity of the larger streams. Above the quicksand the materials consist of sand and clay, often mingled in varying proportions, and frequently containing beds of till and thin bands of gravel.

The indurated rocks underlying the loose surface rocks consist of alternating beds of sandstone and shale, with occasional layers of limestone and seams of coal, and belong to the Upper Carboniferous coal measures.

In the early reports on the geology of Illinois the successive coal seams in the state were assigned numbers, beginning with No. 1 at the bottom. More recent work has demonstrated that the numbers formerly applied to the various coal seams in different parts of the state do not always indicate their true correlation. The following change in the use of the numbers is to be noted: The thick, blue-band coal, which is the principal seam worked in this area, will be designated coal No. 6 instead of No. 7, as in Worthen's reports on Williamson and Franklin counties and other counties to the east. This seam is also the equivalent of the Duquoin Coal of the Perry County report, and corresponds with coal No. 6 in the Belleville field. The 4-foot seam, occurring about 35 feet below the base of No. 6, will be referred to as coal No. 5. Another seam, about 250 feet below No. 5, will be designated coal No. 2. These respective coal seams are persistent and easily recognized horizons; the divisions of the Pennsylvania series in this region are made with reference to them, as shown below.

UPPER CARBONIFEROUS OR PENNSYLVANIAN SERIES

The McLeansboro formation embraces all of the "Pennsylvanian" beds occurring above coal No. 6 in this area.

The Petersburg formation comprises the strata between the base of coal No. 5 and the top of coal No. 6.

The La Salle formation includes the strata lying between the base of coal No. 5 and the base of coal No. 2.

The Pottsville formation consists of sandstones, thin shales, and thin coal beds lying between the base of coal No. 2 and the top of the Mississippian series. This is considered the equivalent of the Pottsville formation of the Appalachian region.

Pottsville Formation.—The strata of this lower formation of the Pennsylvanian series are known within the quadrangle from the logs of two or three borings which pass entirely through the Pennsylvanian deposits, into underlying Mississippian beds; and from the records of the few other drillings which penetrate the strata below coal No. 2 to a varying depth of from 100 to 400 feet. Further south, the full section of the beds of this formation outcrops within a dozen miles of the southern border of the quadrangle. In passing north from Cobden to Carbondale, along the Illinois Central Railroad, the outcrops of successively higher beds of hard sandstone appear as a series of conspicuous ridges. From the crests of these ridges the layers are seen to dip northward at a low angle, so that strata of successively younger age form the ridges that are encountered to the northward. The aggregate thickness of the strata of this formation in the region south of Carbondale is estimated at 750 feet. Within the quadrangle the greatest thickness recorded in the drill records shows 676 feet of Pennsylvanian strata between coal No. 2 and the bottom of the Pennsylvanian series. In the boring at St. Johns, a thickness of less than 400 feet is recorded for these beds. The formation is thickest at the south border of the basin, and the thickness decreases quite rapidly toward the north. The rocks of the formation consist for the most part of sandstones. Beds of more or less impure shale occur at a number of horizons; while occasional bands of limestone and thin seams of coal are also recorded. The thickest and most persistent coal seam in this formation occurs 40 to 70 feet below coal No. 2, and has a thickness of 6 to 18 inches.

In the La Salle formation coal bed No. 2 forms the base, while the fireclay below coal bed No. 5 constitutes the uppermost member. The average thickness of the strata embraced between these two beds is about 224 feet.

The No. 2 coal seam consists of two benches, respectively, about 22 and 24 inches in thickness, separated by about 14½ feet of shale. Since this is the important coal at Murphysboro, where the parting is thin, it may have the same character in parts of the area here described. About 55 feet above coal bed No. 2 there occurs a rather persistent coal band 6 to 28 inches thick. This coal is thinner northward and thickens toward the south. It is exposed at the surface in the east bank of Crab Orchard Creek. At a place where the coal has been mined

*Geologist Illinois State Geological Survey.

locally by drifting into the hill, the following succession of layers was seen:

Section Along Crab Orchard Creek

		Feet
5.	Yellowish-brown sandstone with numerous small brown spots	4½
4.	Layer of argillaceous limestone	1½
3.	Black, fissile shale	2¾
2.	Coal	2
1.	Gray fireclay	1½

Eight rods north of this exposure the sandstone member at the top appears to be 12 feet thick. One-fourth mile up the creek from this point there may be seen a thickness of 10 feet of the sandstone, overlain by 6 feet of gray, sandy shale.

Fig. 2. Anticlinal in Coal Measures

Three-fourths of a mile further south, at the east end of the wagon bridge over Crab Orchard Creek, the following section was measured:

Section at Bridge Over Crab Orchard Creek

		Feet
8.	Argillaceous limestone	1
7.	Black, fissile shale	3
6.	Coal	2¼
5.	Fireclay	2½
4.	Gray shale	5½
3.	Gray, sandy shale	7
2.	Yellowish-gray sandstone	8
1.	Fine-grained, shaly sandstone	10

The members 5 to 8, inclusive, in this section are considered the equivalent, respectively, of the members 1 to 4 of the section preceding. Corresponding beds outcrop about 1 mile southeast of the last exposure, near the middle of the same township, where the coal has been stripped for local use, on land belonging to Mr. Ben Lewis, and following layers were seen:

Section on the Lewis Land

		Feet
4.	Sandstone	16
3.	Impure limestone	1
2.	Black shale	4
1.	Coal	2

A thickness of 14 feet of sandstone, corresponding with No. 4 above, outcrops in a ravine about ½ mile north of the last locality, where it is succeeded by about 5 feet of shale.

The succession represented in the foregoing sections is thought to correspond with the beds associated with the coal occurring about 55 feet above coal No. 2 and 150 feet below coal No. 5. The strata intervening between these coals are in some records reported as consisting largely of shale, in others mostly of sandstone, and in still others of shale and sandstone in about equal proportions.

About 135 feet above coal No. 2 (80 feet above the coal seam last described) there occurs another fairly persistent seam, 2 feet in thickness. The strata occurring between this coal and the outcropping seam next lower consist largely of shale, although some of the records report as much as 38 feet of sandstone and sandy shale. Between the horizon of this 2-foot coal seam and the base of coal No. 5 is an interval of about 70 feet. In this interval, shales greatly predominate. In a few of the logs 8 to 10 feet of sandstone and one or two bands of limestone are recorded.

Petersburg Formation.—The average thickness of this formation, which includes the strata intervening between the base of coal No. 5 and the top of coal No. 6, is about 50 feet. Coal No. 5 is found in the quadrangle wherever borings have penetrated to the proper horizon. It is remarkably uniform in thickness, averaging 4½ feet in 40 records. The thickness rarely varies more than 6 inches from the average figures. Overlying coal No. 5, is a bed of black, fissile shale containing numerous concretions, or "niggerheads," in the lower part.

Section of Part of Petersburg Formation

		Feet
5.	Gray shale, yellowish where weathered	4
4.	Very fossiliferous, soft, gray, calcareous shale	1¾
3.	Layer of hard, bluish-gray, argillaceous limestone	1
2.	Black, fissile shale with fossils containing numerous concretions 8 to 30 inches in diameter	6¾
1.	Coal No. 5	4

The thin limestone band, No. 3 of the foregoing section, is usually present at a distance of from 4 to 8 feet above coal bed No. 5. Another limestone, 3 to 14 feet in thickness, generally occurs from 1 to 9 feet below the base of coal bed No. 6. None of the other Pennsylvanian formations in this area contains such a large proportion of limestone. Besides these limestones, and the fireclay below the No. 6 coal, the materials of this formation consist almost wholly of shale. This is more or less sandy in the middle portion, and dark, with bituminous matter, at the base.

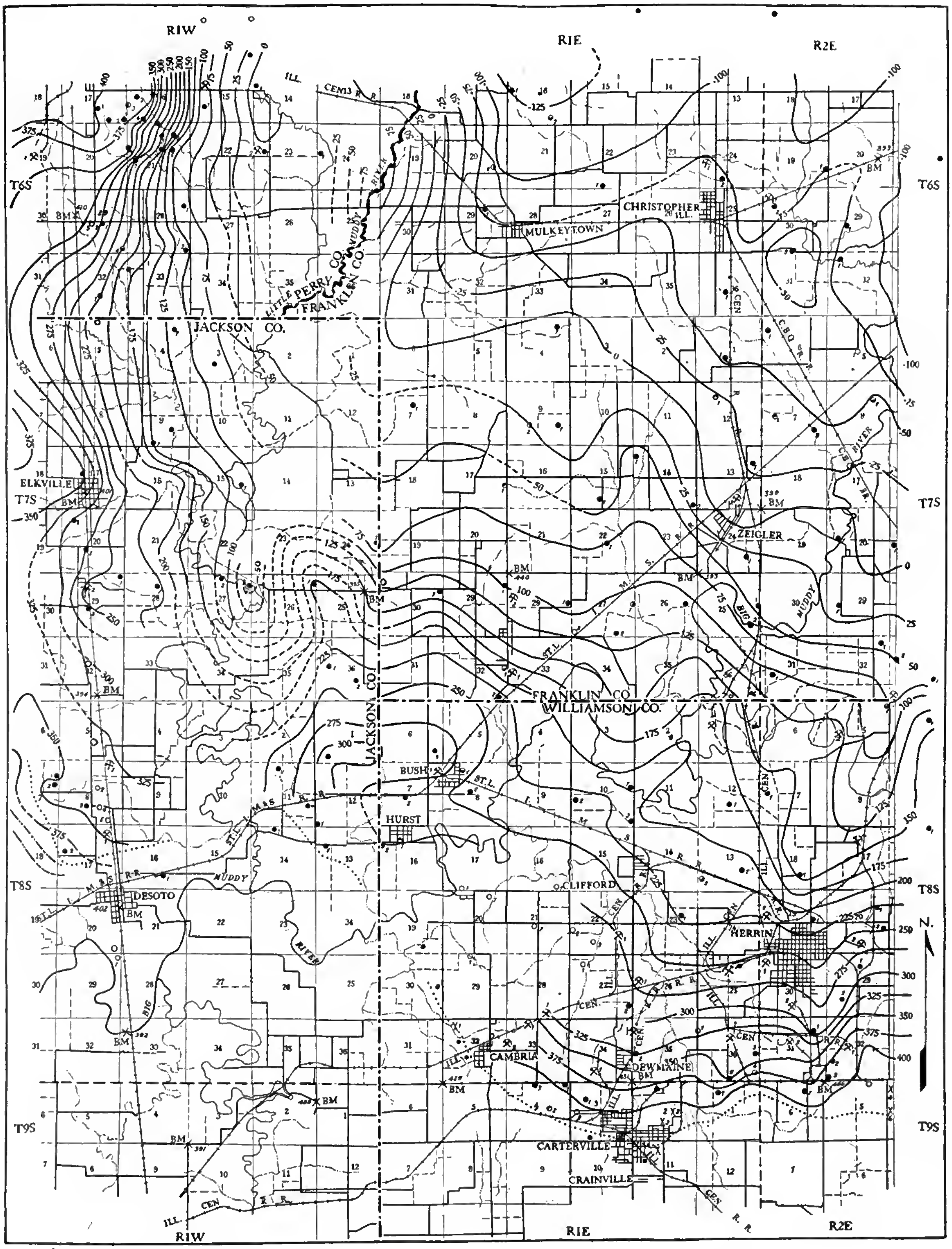
In 130 records the thickness of the No. 6 coal bed, and the intercalated blue band, averages about 9 feet and 5 inches.

McLeansboro Formation.—All of the strata lying above coal bed No. 6 in this region are assigned to the McLeansboro formation. They are more variable than the beds comprising the preceding formations. The greatest depth of the coal beneath the surface, and so the greatest thickness of the formation within the area, is near the north line of the northeast quarter of the quadrangle, where some of the borings go down 560 feet before reaching the No. 6 seam.

The No. 6 coal is succeeded by a bed of gray shale or shaly sandstone 15 to 110 feet thick. The average thickness, in 120 test holes, is 64 feet. Above this horizon is a limestone band 3 to 12 feet in thickness. About 40 feet above this limestone, and 104 feet above coal No. 6, there is generally present a 2-foot seam of coal which, for a thin bed, is remarkably persistent over the area. Still another coal band, 6 to 12 inches thick, occurs about 150 feet above coal No. 6. In the northeast portion of the quadrangle a still higher coal seam is often present about 300 feet above the No. 6 coal. The materials intervening between these several coal horizons consist very largely of shale.

Fig. 3. Coal Seam With Shale Above

A knowledge of the structure of the coal beds and associated strata in any locality is essential to an intelligent estimate of the



289 X BM Bench mark and elevation above sea level

Section lines

Public roads

Private roads

200 Contours showing elevation of base of coal No. 6 above sea level

Contours showing elevation of base of coal No. 6 above sea level, with uncertainty

Approximate outcrop of coal No. 6

2 Shipping coal mine and reference number

X Local mines

3 Boring furnishing record; and reference number

Boring of which only partial record was obtained

Reported boring; no record obtained

Fig. 1. Map of Herrin Quadrangle

31—9—4

cost of mining, and the wise selection of the location for mine shafts. The dip and attitude of the coal seams, as well as the character of the associated strata, very materially affect the expense of drainage and the cost of haulage.

The structure of the strata in the Herrin quadrangle has been determined from the study of about 200 records of test borings, coal shafts, and water wells, using for a base the elevation of the No. 6 coal seam. The position of this seam is easily recognized in the records, and it is found in nearly every test boring made within the area north of its line of outcrop. The altitude above sea level of the No. 6 coal seam in the various borings and shafts was determined as follows: The elevation of the top of the several test holes and coal shafts was generally found by hand leveling from the nearest bench mark. When no bench mark was found near the boring, the surface elevation was determined by aneroid reading which was checked with the nearest bench mark. From the surface elevation of each hole was subtracted the depth to the No. 6 coal at the respective points, as given in the logs.

The coal seams, and the intervals between them, vary somewhat in thickness, yet the strata lie essentially parallel. The layers are not generally horizontal, but dip in different directions, and at various angles, in different portions of the quadrangle. The general dip and the altitude of the No. 6

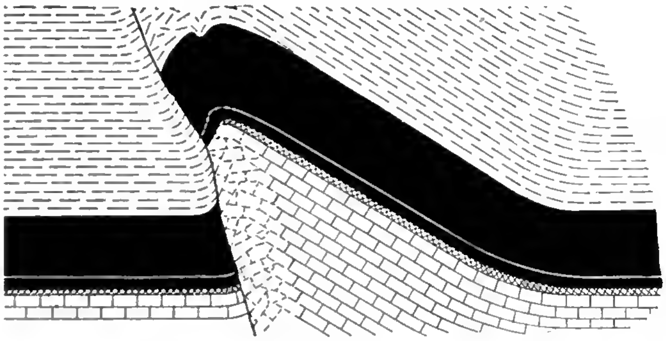


Fig. 4. Sketch Showing Thrust Fault in a Mine

coal at different points is shown on the accompanying map (Fig. 1) by the use of contour lines. The locations of the borings and shafts, the records of which furnished the most of the data used in the construction of the map, are also shown. From the legend the drill holes, mine shafts, and outcrops may be discriminated.

Between adjacent points at which the No. 6 seam was found to lie at different altitudes, the dip is assumed to be uniform. Hence on this structure map, a line connecting all of the known points at which the No. 6 coal seam occurs at an elevation of 400 feet above sea level constitutes the 400-foot contour line. In the same manner all of the points at which this coal was found to lie 375 feet above the sea, are connected by the 375-foot contour line, and so on. A dip of 25 feet is indicated between any two adjacent contour lines.

The assumption of a uniform dip for the coal seam between points where it is found to occur at different elevations is a source of slight error in a map constructed in this manner. A few faults, having a throw of 8 to 21 or 22 feet, and also local irregularities in the dip, and low folds or anticlines were found in some of the coal mines of the quadrangle, and seen in a few surface exposures. Fig. 2 shows a low arch exposed in the north bank of a stream, and Fig. 4 is a sketch showing a thrust-fault seen in Mine "B" of the Chicago-Carterville Coal Co.

The probability of error in the map is greater where the test holes, of which records were obtained, are a considerable distance apart, as in the vicinity of Little Muddy River and in the northeast portion of the quadrangle. Over such areas, where the data are not sufficient to give reasonable certainty of the structure, the contour lines are broken. However, it is practically certain that the errors from the above-mentioned causes do not exceed a few feet along any contour line; and that the general attitude of the No. 6 coal seam, and thus the general structure of the Pennsylvanian strata in this area, is essentially as shown on the map.

From this map it will be seen that in the south part of the quadrangle the No. 6 coal dips strongly toward the north, while near the west border, between Duquoin and De Soto, there is a rapid descent toward the east. Notwithstanding this dominant northward and eastward slope of the strata, local changes of direction and degree of dip are not rare. Near the northwest corner of the quadrangle is an area in which the eastward dip of the coal exceeds 200 feet to the mile. Over other areas the inclination is less than 25 feet per mile. The average dip per mile probably does not exceed 40 feet.

The direction of dip of the strata is assumed to be at right angles to the contour lines. The steepness of the slope is indicated by the distance between adjacent lines. A difference in elevation of 25 feet is shown by adjacent lines, so that the closer together the lines appear, the steeper is the dip, and the wider the distance between the lines, the more gentle is the slope of the strata in that region.

EXTENT OF THE NO. 5 AND NO. 6 COAL SEAMS

There is little doubt that the No. 6 coal underlies practically the entire surface of the quadrangle, with the exception of about 45 square miles in the southern and southwestern portions.

Practically all of the coal produced in this area is taken from the No. 6 seam. It is uniformly thick, ranging from $7\frac{1}{2}$ to 15 feet. The average thickness recorded in 130 borings is about 9 feet and 5 inches, including the blue band or dirt band. The blue band is almost universally present at a height of 18 to 30 inches above the floor. It generally consists of bone or shaly coal, or of gray shale. Its thickness is usually from $\frac{1}{2}$ to $2\frac{1}{2}$ inches. In 25 measurements of the coal seam the average thickness of the blue band was $1\frac{1}{3}$ inches. In the mines along the west border of the quadrangle this horizon is occupied by 5 to 15 inches of gray shale and bone.

A smooth parting of "mother coal" is usually present 16 to 30 inches (averaging 22 inches) below the top of the seam. Small lenses and thin streaks of bone and pyrite are occasionally found. These vary from $\frac{1}{4}$ to 1 inch in thickness, and occur most frequently between the blue band and the upper coal parting. The bony and shaly coal, and the pyrite fragments are picked from the coal by hand, where the best practice prevails.

Above the coal there is a bed of impure, gray shale, 15 to 110 feet thick, shown in Fig. 3, which in the lower part commonly contains a great number of plant impressions. In 120 borings studied, an average of 64 feet of shale intervened between the top of the coal and the overlying limestone horizon. This shale does not stand well when the coal is removed, and so the 16- to 30-inch bench of coal, above the charcoal parting, is usually left for a roof until the rooms are mined out, after which it is sometimes taken down. Timbers are generally set 3 to 5 feet apart in the rooms, and cross-bars are often placed 3 to 6 or 8 feet apart in the main and cross-entries. The fireclay beneath the coal is hard and, as a rule, thin, ranging from 4 to 50 inches. It is generally underlain by limestone and seldom causes trouble by squeezing. Rock rolls locally occur in the top of the seam, the larger ones extending down into the coal for a distance of 2 or 3 feet.

The coal is bright black in color. It is, as a rule, banded and on close inspection appears laminated with alternating bright and dull lines. Thin, more or less persistent partings and lenses, of "mother coal," $\frac{1}{2}$ to 2 inches thick, are found in the upper part of the seam. A distinct cleat is generally present, but it is not so strong as to prevent the cutting of the coal in any direction desired. The composition and fuel value of this coal may be seen from the table of analyses.

The excellent quality of the coal from the No. 5 seam is recognized in this region. Locally, it has a reputation of being superior to coal No. 6. Although it lies about 45 feet deeper, and averages somewhat less than half the thickness of the higher seam, yet its thickness is sufficient to be easily workable and the seam is sure to become of considerable commercial importance. The thickness of coal bed No. 5 is unusually uniform in all of the exposures and records examined, averaging about 4 feet 4 inches at 40 different points. There is no blue band or dirt band in this seam, and the coal appears to be more free from pyrite lenses, bone, and other impurities, than the No. 6 seam. The mining conditions are good. The fireclay below the coal is hard, and does not creep readily. Above the seam is a bed of hard, black shale that usually stands well as a roof with little or no timbering. Concretions or "niggerheads" are commonly present in considerable numbers in the lower part of the roof shale.

Farther east, in Saline County, the No. 5 coal is worked in preference to the upper seam, on account of its higher fuel value and the better mining conditions. The distribution of coal No. 5 is practically coextensive with that of No. 6 in the Herrin quadrangle, and the seam could be mined in connection with the higher coal without the expense of constructing separate shafts.

Analyses of Coals From Herrin Quadrangle and Vicinity. (Face Samples)

All Samples. Coal No. 6			
	As Received		
	High	Low	Average
Moisture	11.03	6.69	9.24
Ash	12.17	7.43	9.13
Sulphur	2.81	.61	1.47
British thermal units	12,200	11,079	11,838
Perry County			
Moisture			9.94
Ash			9.90
Sulphur			.96
British thermal units			11,485
Jackson County			
Moisture			8.96
Ash			9.45
Sulphur			1.91
British thermal units			11,851
Williamson County			
Moisture	9.99	6.69	8.87
Ash	10.83	7.62	8.88
Sulphur	2.81	.92	1.67
British thermal units	12,200	11,753	11,975

Mines and Mining Methods

Some of the mines in this area have been worked for many years, and the smaller ones still employ primitive mining methods. The larger and newer mines are generally provided with modern equipments and conveniences. Pick mining is not usually practiced, but the coal is shot from the solid, and hauled by mules on wooden cars of from 1 to 3 tons capacity. No effort is made to prevent the shattering of the coal in the process of mining. About one-third of the mines are furnished with mining machines for undercutting the coal. Compressed air is commonly used for working the drills, and electricity is, in places, supplied for lighting the mines. Electric motors, or tail-ropes, are used for the main haulage in a number of the larger mines. The deeper mines are usually dry, but the more shallow ones are sometimes troubled with water which seeps in from above. In most cases neither the necessary pumping nor sprinkling entails very great expense.

The mine shafts are generally 9 by 16 or 18 feet, in two compartments. The coal is hoisted by steam engines working a 6- to 8-foot drum. Conical drums are installed in a few mines. The cages are single decked and automatic, dumping the coal on the screens, which convey it to cars on Nos. 2, 3, or 4 tracks. Box-car loaders are employed at a number of the mines. The coal is generally screened and marketed in a number of sizes. At the larger mines washers are established, and 50 to 70 per cent. of the output is washed.

A modification of the room-and-pillar system of mining is in general practice. In rare cases the panel system is employed. In the greater number of the mines few of the pillars are pulled, and in some places where the shale drops easily the roof coal is not taken out. The amount of coal left in the ground varies from 33⅓ to 50 per cent. or more. This exceedingly large waste, in a seam averaging 9 feet in thickness, amounts to a loss of from 5,300 to 7,800 tons per acre, or more than 4,000,000 tons per square mile. In other words, under the present method of mining the No. 6 seam, the amount of coal left in the ground on every square mile is nearly equal to the total number of tons produced in this area in 1908. This very large amount of coal is permanently lost, for it is left in such a condition that it cannot be recovered later. With the price of coal land at $150 per acre, the cost of coal per ton to the company is so trifling—about 1 cent per ton—that the operators find it more economical to leave the pillars and less accessible coal in the ground, and take out only that which can be mined the cheapest. The superintendent of one of the large mines of this area is authority for the statement that more than 95 per cent. of the coal could be recovered if the mines were laid out and developed in a systematic manner with this end in view. Instead of that, in the face of excessive competition, the general practice is to get out the greatest possible amount of coal for the least possible expense, regardless of the waste in the process. A wise policy of conservation will look to the prevention of this enormous and unnecessary waste, even at a slight increase in the expense of production and of cost to the consumer.

COAL DISCOVERED IN TERRITORY OF MAGELLAN

This office is in possession of information, which seems reliable, of the discovery of a superior quality of good coal in the Brunswick Peninsula, to the south of the San Juan River, and on Dawson Island, near Lomas Bay. Samples have been analyzed by a competent party and pronounced equal to the best Cardiff coal. The analysis follows:

After water driven off percentages: Carbon, 76.47; hydrogen, 7.91; oxygen, 8.73; nitrogen, 1.93; sulphur, .97; ashes, 3.99. The specific gravity at 60° F. is 1.1342. A further analysis shows: Volatile matter, 24.85; coke (containing ashes), 73.51 per cent.; water, 1.64.

Supplementing my former dispatches concerning the existence of petroleum deposits in the Punta Arenas district, it may be added that the Chilean National Government recently sent a distinguished mining engineer to make an examination of the territory. After a thorough examination he declared that all the surface indications, judging by his past experience, show extensive petroleum beds. Petroleum has also been discovered on Cambridge Island in the Straits of Magellan, and an application for a concession to work these fields has been made by the discoverers.

All that is now definitely known is that there are surface indications of the existence of petroleum covering a large territory and vouched for by competent authority. Concessions have already been obtained for a part of the territory, and the parties having these concessions are now seeking to interest capitalists to aid them in exploiting the field.—*United States Consular Report.*

POSSIBILITIES OF A NEW LIABILITY LAW*

*By Sim Reynolds**

Mr. President and Gentlemen:—In an indirect way the subject of federal and state compensation to injured workmen has been brought to your attention before, and, doubtless will be again, for the subject is one that will not down. It will continue to agitate men's minds until there is a more humane method of treating our industrial wounded, and caring for those left dependent when the breadwinner is killed. We have in various forms in practically the entire country Employers' Liability. How unsatisfactory this is, statistics chosen at random will suffice to show.

The Question of What Shall We Do With Our Industrial Wounded?

Under the present method of paying liability less than 40 cents of every dollar paid by the employers in the United States reaches the men, women, and children for whose care and comfort it should be given. This of itself is unbusinesslike and inhuman, but what is worse, nearly one-half of the 40 cents is taken before it even gets a chance to get into the pockets of the injured workmen or their families. This, obviously, is the result of an indirect method of disbursement in practice generally—the method of paying compensation through the courts.

The bad effect of such a system can best be grasped when one finds 304 fatal accidents in one county, leaving more than 200 families dependent, and 88 of them receiving not one cent of compensation for the death of their breadwinner. Practically the entire three hundred were contributing to the support of others besides themselves. Ninety-two families received funeral expenses only. Thus 180 of the 304 cases cited, or 59 per cent. of these families were left to bear the entire income loss. Of the families remaining, only 62, or 20 per cent., secured in compensation more than $500 for the death of a provider—a sum which would approximate one year's income of the lowest-paid worker killed.

It is often pointed out that the burden must eventually be borne by the community. But wait. Out of 526 men killed in Pittsburg the city was called upon to bury six, and only seven families made any direct demand upon charity, and in these the items of relief were small. Out of 132 fatal cases, 59 of the widows left went to work, cleaning offices, taking boarders, washing, etc. Out of the 132 families, 22 children were taken out of school before they were 16, immediately after the accidents.

These are some of the things that follow industrial accidents, and, although it robs charity of much credit, it proves that as a rule the family of the American worker is not without a sense of pride and independence in taking up the burden of existence.

A well-framed and well-enforced law regulating this matter is of importance to every mining man. The future holds no reasonable hope, despite our best efforts, that the mining of coal will cease to be attended with death and injury; and it behooves us, by beneficent legislation, to alleviate the human misery inevitably resulting when accidents occur.

In nearly all industrial nations, with the exception of our own, this has in various ways been done; and in certain isolated cases—particularly by one of our own state's largest employers—the initiative has preceded the inevitable ultimate framing of such a statute. Public opinion favors a fairer liability law—one that shall deal squarely with employers, not less than employes. Our present method works quite as unjustly on the operators as it does on the men, the only difference is we, as a rule, are better able to stand it.

Many corporations—some of them in mining business and some out of it—have, however, found the injustice of our present compensation law intolerable. Such large concerns as the Cleveland-Cliffs Iron Co., the United States Coal and Coke Co., and many others, have formed plans of their own, and today are carrying them out independently of the laws provided for such cases. While such individual effort is worthy of emulation so long as present conditions exist, yet it cannot in the nature of things be the best thing possible. It speaks loudly for the need of a common law to govern all alike engaged in industrial pursuits, a law governing liability that shall make it unnecessary for one or two corporations to make their own laws in order to have one worth while.

Thorough investigations on the part of those gentlemen responsible for these self-protective measures have found a condition unparalleled in any large industrial country. They found a very pressing need of revision and cautiously but surely revised and amended to suit their own particular needs. They found that the fees exacted by casualty companies were out of reasonable proportions with the amount that reached their injured men and the widows and children of those killed. So they placed their liability to their employes on a business basis, and have the satisfaction of knowing that, while their own method may not be the best that could be drafted and enforced, yet it does result in each dollar going into the stricken homes of their working men. So far, that is very good considering the chaos and injustice it has displaced.

Unfortunately many corporations, while having the best of intentions toward their injured miners and the wives and children of those who are killed, are not in a financial position to essay the philanthropy, and perhaps are not strong enough to compel a species of insurance with an employed personnel that is here today and gone tomorrow, or at best is more or less antagonistic to any insurance fee being deducted from pay envelopes for that which may never occur to them.

The employer grieves at the harrowing spectacle of abject dependence and poverty too often presented in individual homes following the family's deprivation of its usual income. Unless a general provision is made, to give monetary relief in even the most deserving case would set a precedent the employer—being totally without reimbursement from any point—might find beyond his means to follow. This would create a difficulty the end of which none could foresee; and if the employer takes advantage of the tentative offers of certain casualty companies, and covers the possibility of accident by insurance, he generally finds the result unfavorable from every point of view. The money he would have to give to the poor family deprived of its breadwinner—the money he paid hard cash to secure, as he thought, for that very laudable purpose, he finds is used chiefly in the maintenance of a small army of expert ligitants; and the reasons these expert gentlemen can dig up from the simplest kind of testimony why the injured man or his family should not receive compensation is simply astonishing, and astonishes the employer as much as it mortifies the man seeking redress through the courts.

The one improbable chance of indemnity that the injured has is to prove the employer responsible, and even in that case the operator need not prove himself innocent; the injured man instead must prove him guilty.

Here is where the legal representative of the casualty company gets the lion's share of the argument; and in the face of this long-established phase of English common law, now obsolete in England, the employer who has paid money ostensibly for the benefit of his employes stands helpless. Until there is enacted a specific statute eliminating this idea of "contributory negligence" forever from the zone of legal equity, it will be used for this baneful purpose.

Not long ago a mule driver received internal injuries. He was sent to the hospital, and the case was placed by the coal company with the insurance corporation carrying said company's responsibility in such cases. Some time later a relative of the injured man sought the mere justice of at least hospital expenses, realizing that the accident was due largely to carelessness on the part of the injured man. He was referred to

* Delivered before the West Virginia Coal Mining Institute, Wheeling, W. Va., December, 1910.

the main office by the superintendent of the mine, knowing almost to a certainty that his errand would be fruitless. The men composing that mining corporation are far from being the human juggernauts that poor broken miner and his relatives would naturally believe. I know personally that nothing would have pleased them better than an opportunity to make fair compensation to their injured employe, to relieve him of physical anguish and financial burden at the time above all others when he was unable to bear the latter. Yet they could do nothing more than they had already done—carry insurance with the only available companies existing under our present laws. Shortly after the fruitless visit of the injured man's relative came an astute legal light from a nearby city seeking some phase of negligence which would place the whole burden on other shoulders than those of the insurance company he represented. Quite naturally the very purpose of these casualty associations is opposed to compensation. With them it is a business, just as mining coal is a business with you and me; and their chief aim is to secure the most in premiums at the least outlay. Otherwise they could not exist.

The fault is not theirs; but lies in the antiquated methods of law which leaves us only these round plugs to fit into square holes. Nor does the evil stop there; because, under the present system, the fact alone that the employers, or many of them, have the insurance company standing between them and claims for damages by injured workmen does not breed a desire for precaution that would exist in case a liability law were in effect. Unconsciously, the present system allows high pressure, and makes violation of necessary rules almost excusable.

To the average employer this illuminative idea, which, to quote an eminent English jurist, "originated with Lord Abenger, and to which the devil gave aid and increase," is not only obnoxious, but confounding. It is opposed to his sense of equity. That a worthy man, injured while on duty in an industrial pursuit, should be compensated regardless of the fact that his employer has used due care or regardless of the fact that his employe or his fellow workman has been careless and negligent, and have directly or indirectly contributed toward the accident, admits of little argument in the mind of most mining officials; in fact, it has been generally conceded by the majority of men who have had occasion to speak on the subject. Rather, to the logical mind the fact that he has been seriously injured, and through no fault of his own is disqualified to further continue the struggle for a living and the maintenance of his dependents, is not the least of reasons why the workman should receive compensation at the hands of those able and qualified to give it.

The number of employers who flatly deny the need and justice of direct compensation is so small as to be hardly worthy of notice. The fact that this strenuously struggling nation has its legal and industrial affairs in an unsettled state, is about the only logical reason why a workmen's compensation act has not been made an integral part of our national or state law.

From what certain sources shall a fund for this purpose be drawn? Even those countries in which the idea of compensation has long been a permanent feature differ in method, while being unanimous in principle.

That of Germany—which was the first to universally compensate for death or injury in industry—has some features which appeal to an American; but it has also some which would be hardly applicable here. There, accident insurance is compulsory. Funds are raised by trade associations formed specifically for that purpose. These compensation associations cover all dangerous occupations, and disburse to the workman, or the dependents of a workman totally incapacitated, an amount equaling two-thirds of his wages—two-thirds of which compensation is given out of a fund accruing from an assessment made on wages of employes, and one-third from funds of the employers. If the disability continues beyond 13 weeks the compensation comes entirely out of the employers' pockets. In case of death, compensation is provided for funeral expenses, and a pension for the widow until her death or remarriage, and for every child under 15 years of age. The total compensation in any case is not to exceed 60 per cent. of the deceased workman's yearly earnings.

In the event of a widow's remarriage she is given 60 per cent. of a year's earnings in a lump sum, which settles her account against the association. To carry this cost, trade associations form insurance associations for defined districts. All trades in each district in which compensation is a factor enter and are parties to this agreement, but pay only according to the risk involved. A board of directors computes the amount to be allowed, from whose decree an appeal may be made first to a court of arbitration, formed of equal numbers of employes and employers, lastly to the Imperial insurance office whose decision is final.

The result of this method has been highly satisfactory to the German people. In the year 1906 263,000,000 marks were distributed through the sick fund, and 145,000,000 marks for sickness and compensation. In addition, it is said nearly all the great industrial establishments add to their compulsory compensation by voluntary contributions.

The Germans—an intensely practical people—reached workmen's compensation through the same logical deductions by which we, as an industrial people, must eventually reach it. They had abundant evidence in every village, town, and city where industry on a large scale is carried on that when the workman's family is deprived of its sole means of subsistance the result is not only misery, poverty, and starvation in whole or in part to those who are dependent, but, under old methods, that the result is undeniably inimical to the best interests of the community. They reached the simple conclusion that so long as dangerous labor must be performed by human hands and human bodies, accidents would happen. Further, that an accident would not be an accident if it could be foreseen, prevented, predicted, or avoided; that accidents are the inevitable result of the trade. Therefore, since they stood always liable to occur, and, having occurred, wrought a pernicious result that money only can permanently relieve, the way to a satisfactory solution lay in providing that money by insurance, just as any other possible loss is provided for. The Germans consider an accident as such, regardless of the fact that some fool added his contributory mite to the incapacity of a fellow workman. This should not, and does not, remove the human element from the case; nor does it of itself satisfy the poor family's needs.

England, which gave us this present illogical method of defeating justice, long ago saw the fallacy of it, and removed it absolutely from her statutes. There, the employer is liable for all accidents. In case of injury to a servant the master pays his employe 50 per cent. of his weekly earnings, provided that does not come to more than 1 pound per week. The method for those under age is slightly different, but the result to the employer is similar. Compensation in case of death is 3 years' wages, or £150, whichever is larger, but in no case to exceed £300; compensation is paid to the injured workman or to the deceased workman's dependents in a weekly pro rata.

Compensation in England is made a burden on industry. Fully 13,000,000 people are now working under its provisions, and are in line for its benefits in case accident incapacitates them. A member of the British Parliament, in speaking to certain Americans, found a difference of opinion regarding the English method of placing the cost of compensation totally on the employer. He said, in its justification, that the great majority of thinking men in Great Britain believe the method right. It makes the matter a universal one of additional insurance. Under the new British Liability Law no employer in any part is favored at the expense of another, as was the case in the old law. People engaged in other than manual labor and policemen only are excluded from its benefits. Thus, every corporation and every individual employer, whether he numbers his servants by ten or ten thousand, must take this possible loss

into account at the start, just as he does regarding his boilers, his buildings, his flywheels, his plate glass. He simply extends the insurance to take in the human life used in his business for the same purpose as his boilers or his flywheels—to create profit. The regular wages are paid for the labor performed, and do not—as with us generally—also cover the personal risk. Every industry is made to bear its own cost of insurance.

This is the situation in Great Britain and in every country where compensation is in force—with the exception of Germany, Austria-Hungary, and Luxemburg. The employing interests sustain the whole structure. In the four countries named, however, employes bear part of the expense. In all countries definite compensation is fixed by law, and in all countries except Sweden it is based on the previous earning capacity of the injured workman. In Sweden a flat sum is paid regardless of previous earnings.

While the methods differ, yet the central idea remains the same. In Germany and Austria, the workman is sensibly made aware of his participancy in carrying this protection. Such a scheme might work out here, if the better one of placing a small tax on coal production were not grasped as the means to the same end. If not, the resultant burden placed wholly on the employing interests of our state could not prove much worse than the uncertain one we carry today.

If certain constitutional difficulties could be overcome, to the end that the matter of liability could be made a federal instead of an individual state matter alone, the way out of the difficulty seems plain.

It would seem that the most practical and logical way would be in the form of a slight tax on each ton of coal produced in the United States. The difference to the consumer would be scarcely noticeable; the difference to the welfare of our miners and our miner's families would be beyond computation. I speak only of this matter as it concerns our trade. If congressional provisions can be secured to cover our industry, of course, they will of necessity be more or less universal and the result be beneficial to all industries. Many signs point to a general awakening everywhere throughout this country in the respect of the public's duty to its injured toilers. Men, and women too, in constantly increasing numbers are beginning to see in a wounded soldier of industry a fellow human equally deserving of public bounty as the man who falls while bearing arms. Both are essential to the peace, the greatness, and the ultimate welfare of any nation; and the fact that the miner, the mill worker, or the man who sets in place the steel columns for our huge and towering buildings loses his arm, his limb, or his life without the attendant heroics and the brazen blare of the battlefield does not, in the minds of sober-thinking people, detract one iota from the worthiness of his action or the just deserving of reward. General Edward O'Toole says: "I may add that the public shows in many ways that it is not only willing but eager to pay a few additional cents per ton for its steel or its coal, if thereby the business of producing these necessaries may be conducted in a safer and more humane manner."

We have all felt at times the anguish of impotency in face of a great need that was too obvious, too glaring, to be hidden, and too pitifully insistent to be casually ignored. A few of us—fortunately a few—have seen this crying vital need amplified by tens and hundreds following a mine explosion. Then, being impotent, we have tried to forget; but before long it has happened again, and the same dismal scene, unrelieved by the gleam of human sunshine that efficient compensation would lend to the picture, has been gone all over again. Again and again we have been an unwilling participant in that moral and physical tension which is thrust by these accidents on the employing as well as the employed, on the official as well as the subordinate. That mining man does not live nor work, I hope, in West Virginia who, at such times, has not hoped and prayed for the possibilities of hunger to be positively removed from these stricken homes. More, perhaps, than other men, we share the common opinion which seems to sanction the step that would remove this question from the abstruse to the concrete, and set it in practice throughout the states. We have plenty of wholesome precedent to guide us if we need it, out of which we may readily take the best and leave the worst.

Yet even more glaring are the faults of our present system which should be scrupulously avoided. Briefly they are:

1. Unwise legislation, on which large sums are spent to no permanent benefit.

2. We should particularly avoid anything approximating our present roundabout system in which it is positively known that millions of dollars are annually paid for compensation, and yet not near 30 per cent. reaches the injured or their relatives. Our new liability should be nothing if not direct. There should be no necessity of lawyers in this matter; compensation should not be secured through lawsuits.

3. Another glaring feature of our present method we should take heed to remove is, that liability secured through the courts—no matter how small or large it may be—is anywhere from 4 months to 4 years in reaching the recipient. Also it breeds misunderstanding and distrust and enmity where there should be friendliness and cooperation to reach the same end. Such compensation as a newer and better liability law would allow the workman should be paid immediately—at that time when it is most needed. And the removal of all need of threshing the matter of compensation in the courts would not—as some opponents have said—breed carelessness in operation. The fact that a new liability law would take the injured employe's incapacity as sufficient evidence to award him directly his just dues would hardly instill in the wage earner a desire to deliberately get hurt or killed on purpose to get the award. I can't imagine a man saying to himself before starting to load coal under dangerous stone: "Well, here goes. If I do get my back broken my wife will get $3,000 to take care of the kiddies." A workman's carelessness is not deliberate, but spontaneous and impulsive, even though it is habitual with some men. The miner would under our present "high pressure" rules get behind in the race if he didn't take a chance now and then. But the trouble is there are no "loser's ends" in this race. If the fear of death doesn't insure caution in the workman the fear of poverty for his family will hardly do it.

The highest compensation settled on as fair award for injuries in any of the countries considered, comes far below the average earnings of the workman to whom it is paid, hence it would neither be comfortable nor profitable to be careless and thereby get in line for liability funds. Therefore we should above all institute directness in applying any new method. In what particular manner we shall get at it must be left for calm deliberate study.

In addition to the several notable companies which have voluntarily formed compensation funds several states have placed themselves on record as favoring the principle. One of the most progressive and yet uncomplicated systems looking to this end went into effect on the first day of October last in the state of Montana. The legislature there has placed a tax of one cent per ton on all coal mined in the state, to be paid by the operators, and one per cent. of the gross monthly earnings of each employe constitutes the general fund. This is invested by the state auditor. Three thousand dollars are paid to the beneficiaries of each man killed. In case of permanent disability the miner receives $1 per day.

In the state of New York members of a committee appointed to thoroughly study the question placed themselves on record as favoring compensation in at least all dangerous occupations. The report and suggestions of this committee are too elaborate to be detailed here, but, summed up, reach about the same end in a more roundabout way. What action has been taken on this report I have not at this moment means of determining; but the result will undoubtedly be a compensation law for the

state of New York. Nebraska, Maryland, and Wisconsin are among the other states which have officially considered the problem of "What Shall We Do With Our Industrial Wounded?"—all of which adds, if proof were needed, to the contention voiced previously before this Institute, that the United States are awakening to their responsibility in this respect as never before.

What I advocate, is the greatest possible utility from the liability burden we already carry—a huge load of some $23,000,000 annually. This is the tax our present antiquated, inefficient liability law imposes on the employing interests of the United States. To what end? It is computed, from reliable statistics, that less than 30 per cent. of all that huge sum of money reaches the workman for whom we pay it. Why? Because we, too, suffer from the middleman. Even that 30 per cent. must, under our present method of compensation only through a lawsuit, be divided with legal gentlemen who drag the case out to its farthest end. On every dollar we pay in liability fees the families needing it get 25 or 30 cents. Worse still, even that is, on an average, 3 to 4 years reaching them. In the meantime 9 of every 10 families dependent on compensation have for that length of time been public charges, or children who should have been in school and weak women who should have been at home have been compelled to do their poor best to provide food.

Contrast our method with that of Great Britain, where the percentage of lawsuits in this premise is one in a hundred cases, and where, out of each dollar paid by the employer for relief of his injured workmen, practically the entire 100 cents reaches those who need it, and whom he desired to get it. They, have simply rid themselves of an out-of-date, costly, unfair system of liability. In contrast with us, peace between employer and employe remains unbroken—when an accident happens both know what the result is going to be. Here, an accident sets the employer on the defensive against a jury which may prove unfair to him or unfair to his servant. Both go fighting through the courts, where costs and appeals and delays eat up valuable business days and equally valuable dividends. Either that, or a liability company does the fighting for the employer, and a shyster lawyer for the workman, with the spoils falling to the undeserving either way.

Let us fancy, for instance, all the employers in a certain definite district in the coal fields of West Virginia forming a liability association, with a board of directors in control. Let each one annul its liability contracts with outside casualty corporations and place the premiums previously paid into a common fund. Let the board of directors formulate rules and regulations, and designate the amount that shall be paid for injury, or what for death of an employe. Let the companies sharing in the contract add, pro rata with their number of employes, to the amount deficient in any year, and share in the surplus of any remainder. If expenses should be a little more, there would be the satisfaction of knowing that it had gone to the right spot. The whole economic loss would be evenly distributed—and, as a natural result, careless and inefficient employes would be weeded out. The very act of cooperation in this premise would set all the companies concerned on the alert to prevent accidents, and would inevitably, if slowly, result in a beneficent condition as to working conditions, and, doubtless, a diminution of the grievous mortality list. This plan, as others detailed briefly here, has been tried and proved good. One large company which took it upon itself alone said it not only saved money, but did away with legal vexations and curtailed labor troubles. Doubtless, in assuming this plan the board would employ a committee of inspection to work independent of the state, and to be answerable only to the association, to suggest improvements looking to safety at this place and at that, with the aim in view of compelling each member to do his share in effort to decrease liability losses. Thus would the small employer stand on a par with the greater one. By uniting for this purpose—however they hacked at one another's throats in the matter of prices for coal—the weak would become strong. The plan would enable the plant putting out only a few cars a day to share in a liability method available only to such firms as the United States Steel Corporation or those of smaller magnitude. Otherwise the employer of small plants must continue to take his choice of the two evils now open to him: fight any possible cases through the courts on his own responsibility and with his own money, or hide behind a liability company. Within the possibilities of a federal liability law there is nothing whatsoever detrimental to my employer or to yours, gentlemen. I am not urging the placing of additions where the burdens are already as much as can be borne. Rather, I would relieve the moral and financial tension. Does it not occur to you that herein lies a probable solution to a problem that has baffled every man who has to deal with wilful, disobedient, undisciplined employes? Who shall say that a general application of financial responsibility for accidents of whatsoever nature may not compel employers to be less lenient to criminally negligent miners? May it not also instill into the official mining man, and the employer himself, a more wholesome regard for the law as written? Here there occurs to me the only lamentable feature of all the provisions I have read on this subject. I see nowhere that an employe is made responsible. Why not remove the fellow servant contributory clause, and still leave in the penalty for wilful, criminal disobedience or carelessness on the part of the injured man himself? Pitiful indeed, I grant, it would be to see the wife and children deprived of sustenance because of proved criminal carelessness on the part of the husband and father; but might not a greater good accrue because of the widespread lesson? Or would the method of placing the man injured through breaking industrial law where he would not share in the compensation benefits be of avail, at the same time give to the wife and children all that the law allowed? These considerations must, of course, ultimately be dealt with. We can conjecture, but as to the certainty of their working I do not know; but the thought occurred that there should be some way of meting to the criminally negligent employe the just penalties of his injury—if positively proved—as well as its rewards, just as financially the penalty would be shared by the employer. Or would the fact of the workman being deprived of a greater income be penalty sufficient? In some cases I feel safe in saying yes; in others, no. No man—if you will pardon the apparent egotism—is better able to judge the miner than one who, from 11 years of age, has been a miner.

The subject is very wide and in many respects complicated—therefore deserves most careful thought.

SAFETY LAMPS ON BATTLESHIPS

The practical utility of the American Beard-Deputy safety lamp, equipped with the Beard-Mackie sight indicator for the detection and estimation of gas has recently received indorsement at the hands of the Bureau of Steam Engineering of the United States Navy. This Bureau, in connection with the serious explosions that have occurred from time to time on shipboard and elsewhere, in coal bunkers and storage bins, and in the operations of the large fuel-oil storage tanks now required in the navy, after careful consideration, placed a large order for the above type of safety lamp with the American Safety Lamp and Mine Supply Co., of Scranton, Pa.

The indicator requires no skill or technical knowledge in its use. It needs no highly inflammable or volatile liquid to be burned in the lamp when testing for gas. Extremely fine adjustment of the lamp flame when making a test for gas is unnecessary as the indicator shows promptly and plainly the exact condition of the air either with respect to gas or dust. There is no guessing. As little as ½ per cent. of firedamp is clearly shown by the bright incandescence of a single percentage wire, while larger percentages are indicated clearly by the glowing of one additional wire for each ½ per cent. of increase in gas.

SELF-DUMPING CAR HAULS

Written for Mines and Minerals

One of the mine manager's endeavors is to adopt only those improvements that will improve working conditions and decrease cost. This praiseworthy object does not meet the approval of those owners who are unable to understand that the first high cost of an improvement may prove the greatest in ultimate economy.

Different Arrangements Used to Dump Coal and to Handle Mine Cars at the Tipple

Take for example the horn tipple, shown in Fig. 1, which at one time was considered the most excellent device for dumping where the coal came in cars to the tipple floor. The dump was easily and cheaply constructed, and worked easily when built so as to take advantage of the load passing its center of gravity and becoming unstable.

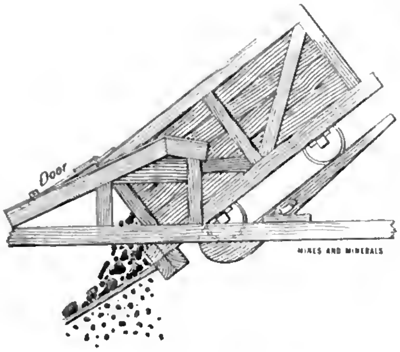


Fig. 1

The most objectionable feature in connection with this horn tipple, was that in case the dumpsman failed to use care the car would hit the horns so hard something would be strained or broken.

Suppose a car weighing 1,500 pounds containing 3,000 pounds of coal, moving at the rate of 5 feet per second is suddenly brought to a stop, its momentum would be 22,500 pounds, and this would be transmitted to the dumper horns, back to the boxes to which the dumper was fastened, to the tipple floor. The reaction due to the impact would reflect on the car boxes, to tear them loose, while the coal would rack the car body.

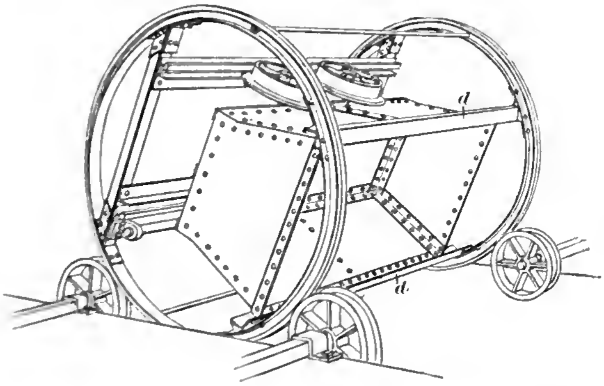


Fig. 2

In a short time when this dump is used some part of the dump and cars will need repairs, or possibly be the indirect cause of more serious matters such as wrecks.

To overcome the objections to the horn end dumper the English mine managers long ago adopted a form of circular dumper similar to that shown in Fig. 2. This to some extent has been adopted in this country, where a cross-over and back switch is used, or where there is an opportunity to approach the chute from the side and dump into it at right angles.

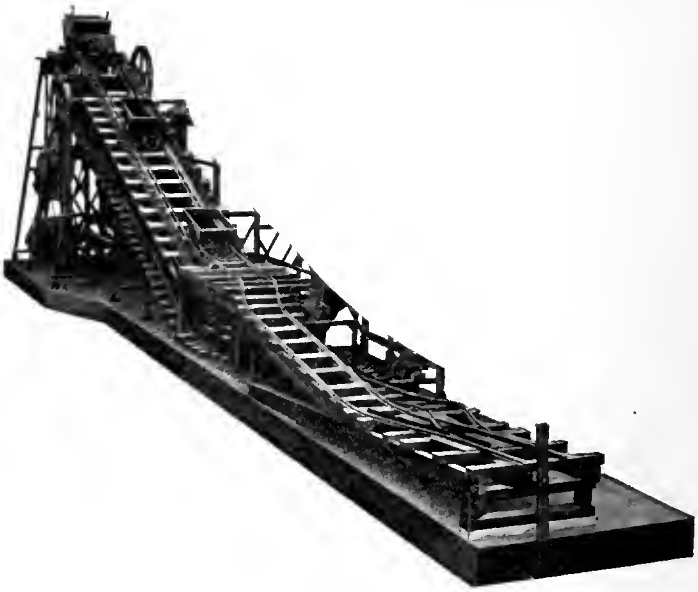


Fig. 4

One feature which adds materially to the value of the circular dumper is that no swinging door is needed on the car, which certainly decreases the cost of car up-keep; prevents loss of coal in transportation, and possible wrecks; keeps the roadways cleaner and does away with the man or boy whose duty is to attend to car doors and latches. The rotary dumper does not rack the cars or work the car boxes loose, besides it can be manipulated so as to pour the coal out into the chute and not send it with a rush that prevents it being properly screened, at least over bars. There is some objection to having fast ends on mine cars, they being harder to load, particularly where the cars are large and the coal bed low; under other conditions this does not militate against them, and in any case they should be used on inclines. Circular dumpers are readily controlled and are sufficiently fast for all practical purposes.

Many changes have been made in the horn or end dump with a view to preventing the shock, to regulate the tilting and dumping, and to prevent the shock when the car has been dumped and returned to its normal position; in every case, however, manual labor has been required to pull the empty car from the dump out of the way so the next loaded car could run on. The Reading Iron and Railway Co. introduced an end rocker dump at their Hammond breaker near Girardville, Pa., which gradually brought the approaching car to rest, and after being discharged the dump in returning to its normal position gave the empty car sufficient momentum to run it out of the way of the next loaded car.

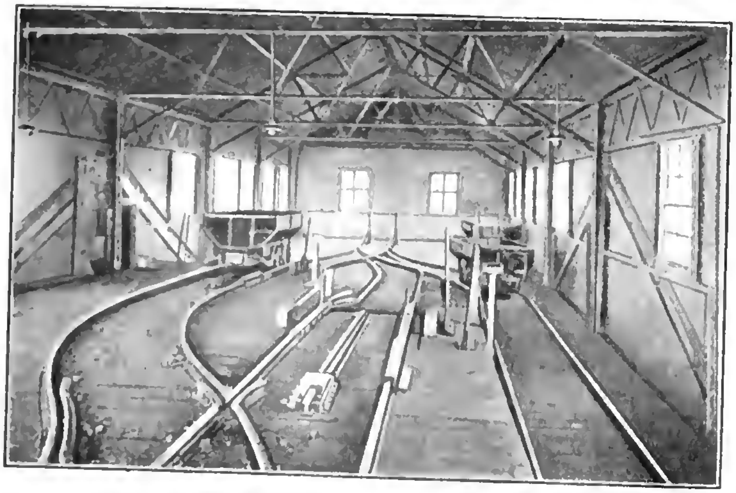


Fig. 3. Interior of Tipple House

In the United States, cross-over dumps are given preference at bituminons coal mines and also at anthracite mines where coal can

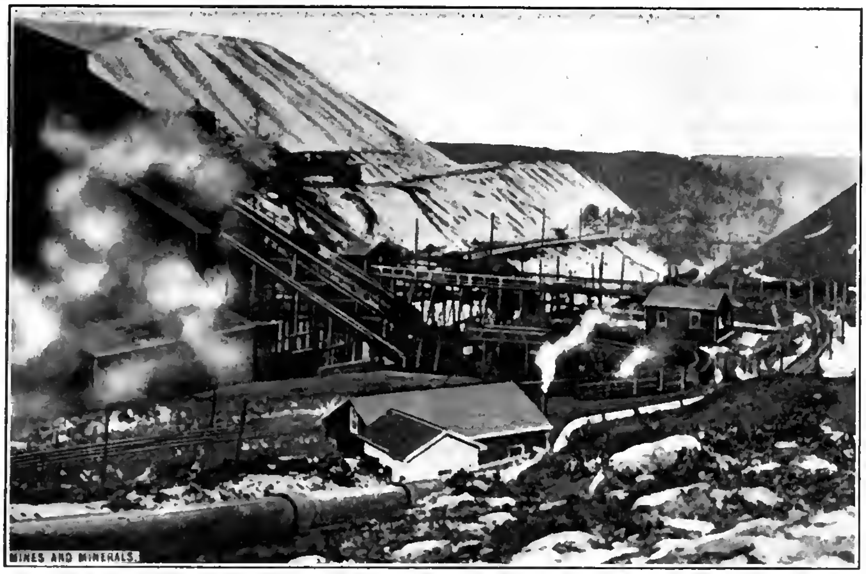


Fig. 5. Burnside Colliery, Shamokin, Pa.

be dumped to advantage at the surface. Cross-over dumpers are arranged so that they can be tilted by hand, or by means of air or steam. The use of this dumper requires that the tipple floor be made long as in Fig. 3 to provide for a kick-back to return the cars automatically to the empty track. At bituminous mines there is no particular objection to this length unless it be the cost; but at anthracite mines the extra room at the top of the breaker could not be obtained, except possibly where the breaker was built on a side hill and the coal came on to the dumping floor on a level.

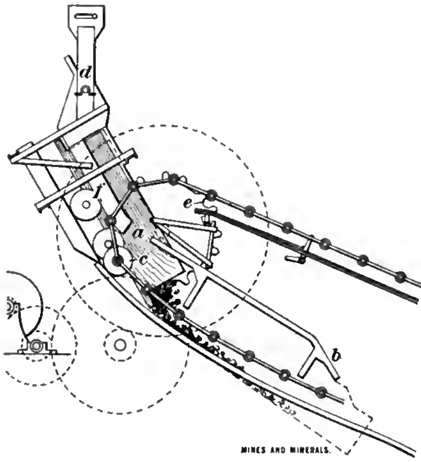


Fig. 6

Where the coal comes to the bottom of the breaker at grade it is becoming the custom in the anthracite fields to use a cross-over dump at the surface and raise the coal up to the top of the breaker by means of an inclined conveyer or scraper line, while in the bituminous fields it is customary to use a chain haul and raise the car to the tipple floor and then unload by means of the cross-over dumper. This requires the use of a bottom man, a dumpsman and helper and an empty-car pusher. It occurred to F. C. Green, a mining engineer in Cleveland, that he could reduce the length of the tipple; the expense of the installation of a cross-over; and the number of men required for the dumping, by making use of the conveyer system, and also obtain the advantages of that system, which are less breakage, cleaner coal, and less vibration to the structure. The result of his endeavors is shown by the model, Fig. 4, that was exhibited at the Wheeling meeting of the West Virginia Mining Institute, and in Fig. 5, which is illustrative of the installation at the Burnside colliery of the Philadelphia & Reading Railroad near Shamokin. The Green self-dumping car haul is constructed on the principle of the endless double-chain conveyer, with a spreader or cross-bar, in which the cars act as conveyer buckets, the loaded cars traveling up the lower runway and the empty cars traveling down the upper runway. In Fig. 8 is shown an arrangement at the foot of the incline where an automatic car feeder and car spotter delivers the loaded cars at definite intervals to the car haul. The car haul consists of a double endless chain, that travels between sprocket wheels on car wheels with axles that act as spreader bars, as pushers, and as retarders of the mine cars when on the plane. Tracks for the mine cars are inside the tracks for cross-bar wheels. When a car is delivered to the haul, the cross-bar moves up to the back car wheels and lifting them from the track assumes the responsibility of parts of the load and pushes all the load up the incline; as the car nears the discharging point the car door *a*, Fig. 6, is opened automatically by the latches *b* and the car being at the normal pitch of the slope, 23 degrees, the coal commences to run out. The full discharge is not effected at once but gradually while the car is in transit, during which time the pitch varies from 23 degrees to 60 degrees; this spreads the coal gradually over the grate bars thereby cleaning it of fine material thoroughly.

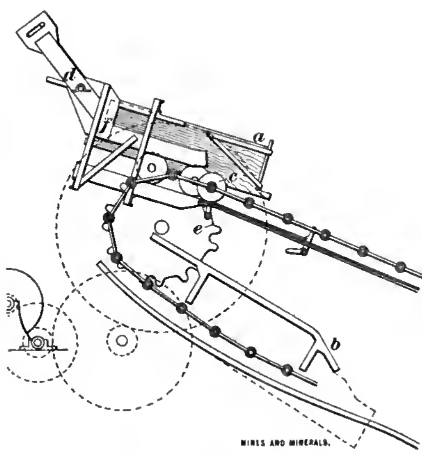


Fig. 7

The cross-bar *c* continuing to travel pushes the car on to the swing-lift transfer *d*, and as the bar must follow the sprocket wheel *e*, it carries the rear end of the car with it over the sprocket and on to the upper or return empty track. This operation is shown in Fig. 8.

The car is received by the swing-lift transfer and retained on the lift by the cross-bar. As the cross-bar starts around the head driving sprockets *e*, the cross-bar wheels engage the horns *f* on the

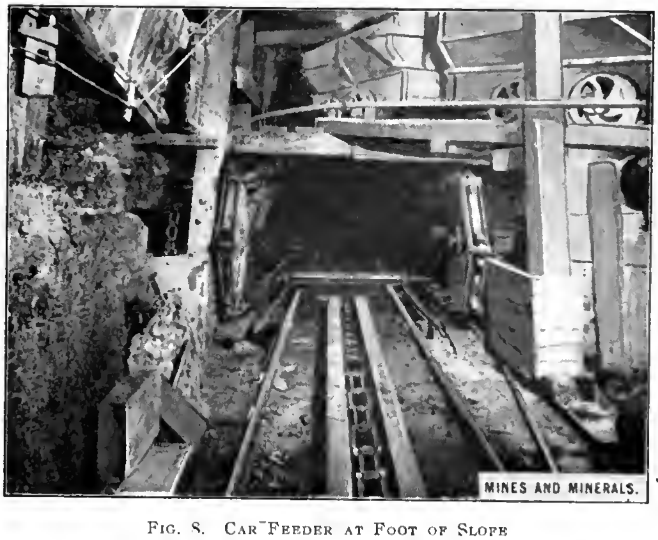


Fig. 8. Car Feeder at Foot of Slope

swing lift. The continued travel of the cross-bar around the sprockets causes the free end of the car to lift and register with the upper runway. The lift is latched in register with the upper runway until the car following the cross-bar is clear of the swing lift, when the cross-bar trips the latch, allowing the lift to swing down and register with the lower runway. The empty car continues to follow the cross-bar down the upper runway of the haul, supported as previously described, until it reaches the bottom of the upper runway.

The writer is indebted to E. J. Neville, Manager of the C. O. Bartlett-Snow Co., who manufacture this dump, for the information concerning it.

FIRST-AID FOR INJURED SPINES

Written for Mines and Minerals, by Thomas C. Harvey

As all injuries to the spine are of a serious nature, it follows that they demand most careful consideration, and especially so where the accident has occurred in the mines or other places remote from immediate medical attention.

Fig. 1

Having been a student of first-aid work for a number of years, and not finding in the various textbooks on first aid anything more definite on the subject than instructions to "keep the patient quiet and send for a physician" (or words to that effect), I feel there should be something more suggested as practicable to aid a person in this serious condition.

Such acidents occur almost without exception at the working face, which is sometimes 1 or 2 miles from the opening to which the injured workman must be carried. It is then that the question arises, how shall the man be handled without further injury and increased pain and suffering.

Since I have been connected with the mine-rescue work of the United States Bureau of Mines, I have been asked a number of times, "what is the best thing to do in case of injury to the spine?" and I have tried to answer the question in a practical way by illustration.

The first-aid man knows that in all injuries one of the essential points is to make the patient as comfortable as possible, binding him up with splints and bandages to attain that end, and it should be particularly true in a case of an injury to the spine, because one has to deal with the most vital nerve center of the body, where the least jar may mean long suffering and eventually death. I may mention that I am taking a case where the spinal ligaments or cord have not been severed.

I therefore submit a dressing which I believe will be a great help to the first-aid man in his humane work and will put his patient in as comfortable a position as possible until he is placed in professional hands.

The splints recommended for use in this particular case are in the form of a double cross and may be called a cradle splint. It consists of four pieces of board preferably 1 inch thick, 3½ inches wide, and two pieces 16 to 18 inches long, or long enough to support the shoulder and hips, as shown in Fig. 1 Also two pieces 4 feet 6 inches to 5 feet in length, as the case may be, or long enough to reach from the top of the shoulder down to the heel. The two long pieces are to be held in place by the short pieces as shown in the illustration. The long splints should be placed about 3 inches apart and directly on the back part of the leg and running up over the muscles of the back. The splints can be nailed together or held in position by bandages, as shown in the illustration.

Fig. 2

Having secured the material and made the splints, the first-aid man will proceed to pad the surface that is to be applied to the back with anything that is at hand, such as hay, cotton waste, cotton batting, or strips of clothing held in position on the splints by strips of brattice cloth, mine cotton, insulating tape or small battery wire, or anything that is at hand that will answer the purpose.

The method of applying the splint is a very important and necessarily careful operation. The first-aid man, with the assistance of three or four other workmen, will turn the patient on his side, being very careful to turn him with one motion, as shown in Fig. 2, so as not to twist the spine, placing the splints on the back, turning him back again flat on the ground before bandaging. He is now placed in the cradle, and the operator should proceed to bandage him to the splint as shown in Fig. 3, starting at the cross-pieces at the shoulder, securely fastening the shoulder to the splint, then fastening the hip joints in like manner. The bandaging should continue down to the knees and ankles, as shown in the illustration. By using inch material a very rigid splint is produced, and then again it will not be necessary to move the patient until he is securely bound, because there is an open space of an inch or more through which to pass bandages under him while he is lying on the ground.

Fig. 3

The patient is now ready for transporting to the surface, that is, after due precaution has been taken to reduce the shock as much as possible, which is severe in this case. He should be moved with the greatest of care and by a first-aid team practiced in the work; and if a person is handled in this manner, I feel confident he will be handed over to the physician in a comfortable condition and with a good chance of complete recovery, which he would not have had with the old method of handling a man who had suffered such injury.

I may add that this particular splint has met with approval wherever shown and that it can be made and kept on hand for an emergency, convenient to the working faces and under the charge of the first-aid men, so that in case it should ever be needed, the time of making it can be used to the benefit of the patient.

Consul Charles Adams Holder, Rouen, France, says: "Not only does France need coal but the United States should furnish it." The total coal production of the world for 1908 was 952,000,000 tons, approximately, made up as follows: United States, 317,323,836 tons; United Kingdom, 261,506,379 tons; Germany, 146,232,646 tons; France, 36,281,098 tons; all other countries, 136,656,041 tons. This is exclusive of lignite, of which the world's production was 100,824,000 tons.—*United States Consular Report.*

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

Note.—The following questions, selected from different examinations, are here numbered consecutively. The examination from which the question is taken and the number of the question are found at the end of each.

Selected Questions of the Iowa Mine Foremen's Examinations, Held at Des Moines, October 12, and December 6, 7, 1910

Ques. 1.—What is the meaning of "coefficient of friction"? †Q. 1.

Ans.—This expression, as used in mine ventilation, relates to the frictional resistance per square foot of rubbing surface, offered by an airway to the passage of air-current having a velocity of 1 foot per minute. It is often called the unit of resistance. Its commonly accepted value in America is .00000002. (The total resistance for any airway is found by multiplying the number of square feet of rubbing surface by this coefficient, and that product by the square of the velocity of the current expressed in feet per minute. The final product is the resistance in pounds.)

Ques. 2.—A circular airway has a diameter of 9.026 feet what must be the velocity in order to pass 21,120 cubic feet of air per minute? †Q. 2.

Ans.—The sectional area of the airway is $.7854 \times 9.026^2 = 63.985$, say 64 square feet. The velocity required to pass the given quantity of air through this airway is then $21,120 \div 64 = 330$ feet per minute.

Ques. 3.—An entry runs N 46° 54′ E, and the rooms off this entry bear S 61° 21′ E; what is the angle these rooms make with the entry? †Q. 3.

Ans.— $180° - (46° 54' + 61° 21') = 71° 45'$.

Ques. 4.—Tell how to find the latitude and departure of any course. *Q. 3.

Ans.—Multiply the horizontal length of the course by the cosine of the angle of its bearing to obtain the latitude of the course; or multiply the same by the sine of the bearing to obtain the departure.

Ques. 5.—What is the angle included between two lines whose bearings are, respectively, N 10° W and S 88° E? *Q. 6.

Ans.—The included angle is $10° + 180° - 88° = 102°$.

Ques. 6.—If the reading of a plain compass is N 43° W, what would be the true reading after correcting for a declination of the needle of 7° E? *Q. 7.

Ans.—A declination of 7° E means that the north end of the needle points 7 degrees to the east of the true north. In a plain compass, which has no arc for setting off the declination, the reading gives the angle that the line of sight makes with the direction of the needle, which in this case is evidently too great by 7 degrees. The true reading would be therefore $43° - 7° = 36°$, and the true bearing of the line of sight is N 36° W; because the true meridian lies between the line of sight and the magnetic meridian in which the needle sets.

Ques. 7.—What is the disadvantage of using a long needle for mine work? *Q. 8.

Ans.—Some prefer a short needle for use in mines, because it works more quickly and is not as liable to become bent by hard usage; also, the compass is smaller and more easily carried about the mine. The lack of these qualities may be termed the disadvantages arising from the use of a long needle. The latter, on the other hand, though setting more slowly is more reliable and can be read with greater accuracy than the former.

Ques. 8.—How would you lay a good haulage road in the main entry to secure good results and long duration? †Q. 5.

Ans.—To provide a permanent main haulage road in a mine all the chief points mentioned in reply to Ques. 11, page 280, Mines and Minerals, December, 1910, should be considered. The entry should be carefully surveyed and platted. On a profile of the entry establish a grade that shall be as nearly uniform and favor the movement of the loaded cars as far as practicable. Begin the work by straightening the entry and lightening or reducing the curves wherever practicable to do so, and grading the entry to conform to the established grade. Provide ample drainage ditches at the side of the road. Use good ties and iron of suitable size for the proposed traffic. Having laid the ties and spiked and bolted the iron, straighten and ballast the track. See that all low places where the road crosses a depression are kept well drained.

Ques. 9.—How would you proceed to guard the health and safety of the miners and the security of a mine placed in your charge? †Q. 6.

Ans.—It is important, first, to become thoroughly familiar with the conditions existing in and about the mine, and the character of the labor employed, with respect to nationality, intelligence, and general capability. After studying these conditions make and enforce such regulations as will tend to remove, as far as possible, all dangers or liability of accident found to exist. Insist on the strict enforcement of the mining laws. Hold each assistant foreman, fire boss, engineer, pump runner, cager, timberman, fireman, or other person charged with certain duties, to a strict accounting for the faithful performance of such duties. Keep constantly on hand a plentiful supply of timber and all needed material, and arrange to have this material delivered promptly in the mine when required. Watch carefully the ventilation to see that the proper quantity of air is circulated and so distributed to all portions of the working face that no dangerous accumulations of gas can occur. Encourage all men employed in and about the mines to read and study mining books and inform themselves on all matters relating to the dangers of mining.

Ques. 10.—What method is the safest to adopt in the use of dynamite in shaft sinking and which will least endanger life and limb? †Q. 8.

Ans.—Electric firing is the safest method and should always be employed in sinking a shaft. The lead wires at the top of the shaft should remain disconnected from the battery while the men are in the shaft, and should never be connected up by any one but the shot firer, who should be the last man to leave the shaft. Where electric drills are employed the power should be shut off from the drills at the surface before the holes are charged or any connection made for blasting, in order to avoid the possibility of the dynamite being set off by an accidental short circuiting of the current used to supply power to the drills.

Ques. 11.—What is the total pressure required to overcome the friction or resistance and produce a velocity of 400 feet per minute in an airway 6 ft. × 8 ft. and 3,200 feet long? †Q. 9.

Ans.—The rubbing surface in this airway is $3,200 \times 2(6+8) = 89,600$ square feet. The total ventilation pressure, then, for a velocity of 400 feet per minute $p\,a = k\,s\,v^2 = .00000002 \times 89,600 \times 400^2 = 286.72$ pounds.

Ques. 12.—In the last question, what is the pressure per square foot? †Q. 10.

Ans.—The sectional area of the airway is $6 \times 8 = 48$ square feet. The pressure per square foot is, then, $p = 286.72 \div 48 = 5.97+$ pounds.

Ques. 13.—What do you understand by the expression the natural ventilation of a mine? †Q. 13.

Ans.—By natural ventilation is meant such circulation of air as results from natural causes without the use of any artificial means.

Ques. 14.—Under what conditions and circumstances may natural ventilation occur? †Q. 14.

Ans.—Whenever two shafts or slopes, connected underground and open to the atmosphere, are filled with air of different density, there result two air columns of different weights and therefore unbalanced. The heavier column sinks and that presses the lighter one upward, thus producing a movement of the air in the passages connecting these two openings. The air

* Mine Foremen's Examination, October 12, 1910.
† Mine Foremen's Examination, December 6, 7, 1910.

flows from the colder or denser column to the warmer or lighter column.

Ques. 15.—Why should the circulation of air in mines be maintained continuously, and not during the day only? †Q. 15.

Ans.—Because the gases that accumulate during the hours when ventilation ceases in the mine find lodgment in abandoned places; the heavier carbon dioxide sinking to the swamps and filling dip workings with a foul atmosphere that is exceedingly difficult to remove, while the lighter marsh gas rises and forms exceedingly dangerous firedamp mixtures in all pitch headings and chambers, and in roof cavities, and on the falls. It will often require hours for the removal of these gases and the delay of work from this cause will generally outweigh any slight saving that would arise from closing down the ventilator for a portion of the night.

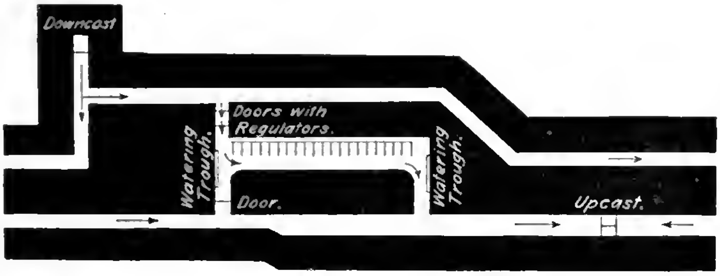


Fig. 1

Ques. 16.—What is furnace ventilation and how is it accomplished? †Q. 16.

Ans.—Furnace ventilation is produced by the use of a furnace built in the passageway, near the bottom of the shaft, which is then called the furnace shaft. The heat of the furnace serves to rarify the air passing over and around it and ascending the shaft. It is this heating of the air in the furnace shaft that makes this shaft always the upcast and produces the circulation of air through the mine, because the hot air in the furnace shaft is lighter than the cooler outside air descending the downcast shaft or slope or entering the drift mouth.

Ques. 17.—Are dangers possible with furnaces, and how may such dangers be avoided? †Q. 17.

Ans.—In the use of a mine furnace there is the danger of the flame or the heat of the furnace setting fire to the shaft timbers or the coal pillars nearby. A furnace should not be used in a mine in which gas is liable to be generated. There is danger of the gas in the return current being ignited at the furnace or in the shaft. To avoid the possible ignition of the shaft timbers, a mine furnace should be set from 10 to 15 yards back from the foot of the shaft, according to the size of the furnace and the depth of the shaft; and the shaft should be walled with masonry for a sufficient distance up from the bottom to avoid all danger. To prevent the ignition of the pillar coal a brick arch should be built in the passage extending from a few feet in front of the furnace to shaft walling. The roof strata should be supported by iron rails or beams spanning the airway and an air space left at each side and above the brickwork. Firebrick should be used for the most of this work. To avoid the ignition of gas, two methods have been employed: (1) The return current is diluted with a sufficient addition of fresh air from the intake to render the gas non-explosive before it reaches the furnace. (2) The return current containing gas is made to enter the shaft by a dumb drift at a point from 30 to 50 yards above the bottom of the shaft; the furnace in this case is fed by a separate scale of fresh air from the intake.

Ques. 18.—If a certain fan produces 65,000 cubic feet of air, at a certain mine, running at 40 revolutions per minute, what volume of air would it produce at a speed of 55 revolutions per minute? †Q. 18.

Ans.—It is generally assumed that the volume of air produced by a fan, under the same conditions, is proportional to the speed of the fan, making the quantity ratio equal to the speed ratio of the fan; or, in this case

$$\frac{x}{65,000}=\frac{55}{40}=\frac{11}{8}\text{; and } x=65,000\times\frac{11}{8}=89,375 \text{ cu. ft. per min.}$$

In practice, however, the results are more nearly,

$$x=65,000\sqrt[5]{\left(\frac{11}{8}\right)^4}=83,860 \text{ cu. ft. per min.}$$

Ques. 19.—If 170,000 cubic feet of air is passing through a fan drift per minute, under a water gauge of 2.1 inches, what is the horsepower on the air? †Q. 19.

Ans.—The power on the air is

$$H=\frac{170,000\times 2.1\times 5.2}{33,000}=56.25+ \text{ H.P.}$$

Ques. 20.—Explain the meaning of the expression "motive column" in ventilation. How is the motive column in any circulation of air expressed, and how is its value ascertained? †Q. 20.

Ans.—The word motive column, as used in ventilation, denotes an imaginary column of air of given density, 1 square foot in cross-section and of such height that its weight is equal to a given pressure expressed in pounds per square foot. Motive column is expressed in feet. The motive column corresponding to any given pressure expressed in pounds per square foot is found by dividing the given pressure by the weight of 1 cubic foot of air at the given temperature and pressure. If the pressure is given as inches of water gauge, it must first be multiplied by 5.2 to reduce it to pounds per square foot and then proceed as before. For example, assuming a temperature of 60° F. and a barometric pressure of 30 inches, find the motive column corresponding to a water gauge of 2.5 inches. The pressure corresponding to this water gauge is $2.5\times 5.2=13$ pounds per square foot. The weight of 1 cubic foot of air at the assumed temperature and pressure is $\frac{1.3273\times 30}{460+60}=.0766$ pounds. The corresponding motive column is then $13\div.0766=170$ feet, nearly.

Ques. 21.—What is the total pressure required to overcome the frictional resistance of an airway 7 ft. × 5 ft. and 6,000 feet long when the air-current is traveling with a velocity of 300 feet per minute? *Q. 5.

Ans.— 259.2 pounds. See Ques. 11.

Ques. 22.—If the ventilation is insufficient and you cannot increase your power, how will it be possible to increase the volume of air in circulation? *Q. 10.

Ans.—It may be possible to split the air at one or more points in the mine without reducing the velocity of the air-current below what is required for good ventilation. This will increase the quantity of air circulated without requiring additional power. The same effect is accomplished also by cleaning up airways, straightening air-courses, and wherever practicable shortening the distance the air-current must travel.

Ques. 23.—How would you construct and ventilate an underground stable, and where would you locate such stable in a new mine? *Q. 14.

Ans.—A mine stable should be excavated in the solid coal. It should be from 16 to 20 feet wide and 7 or 8 feet high in the clear, and as long as will meet the requirements of the mine. All the doors, partitions, roof supports, and fittings of the stable should be of incombustible material. There should be provided a water trough and an ample supply of good water piped to the stable. The floor should be hard and well drained. The stable should be ventilated by a scale of air direct from the intake current, sufficient to maintain a pure atmosphere without making it too cold. The air that passes through the stable should be conducted at once into the main return by which it will be carried out of the mine without further con-

taminating the mine air. In planning a new mine, the stable should be located in the shaft pillar, between the two shafts, so as to provide good ventilation and be easily accessible from either shaft in case of need, and convenient for the handling of the daily supplies and refuse. A good arrangement in a shaft mine is shown in Fig. 1; another arrangement is shown in Fig. 2, which, with some modification, would be applicable to a slope mine, the stable being located at the foot of the slope.

QUES. 24.—In a gangway 8 ft. × 8 ft. the water gauge shows a reading of ¾ inch when the velocity of the air-current is 280 feet per minute; what is the estimated rubbing surface? *Q. 16.

ANS.—A water gauge of ¾ inch indicates a pressure of $\frac{3}{4}\times 5.2 = 3.9$ pounds per square inch; and the total ventilating pressure is then $pa = 3.9\ (8\times 8) = 249.6$ pounds. The rubbing surface may therefore be estimated as approximately

$$s = \frac{pa}{kv^2} = \frac{249.6}{.00000002\times 280^2} = 159{,}183, \text{ say } 160{,}000 \text{ sq. ft.}$$

QUES. 25.—What is gained by splitting air-currents in mines? *Q. 17.

ANS.—A larger quantity of air is circulated per unit of horsepower; purer air is supplied to the working face; the ventilation of the mine is under better control, as the air can be

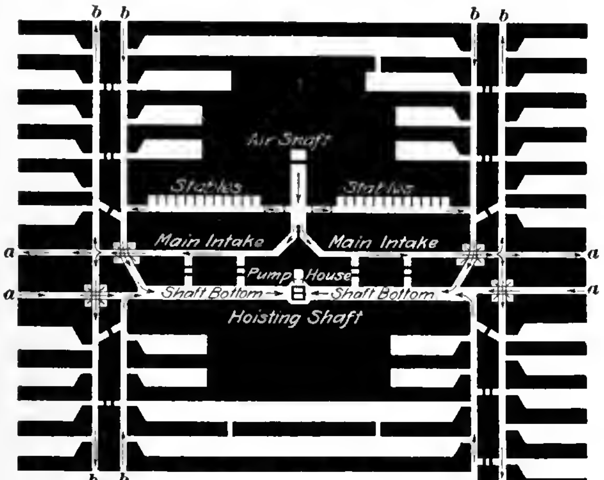


FIG. 2

distributed so as to meet the needs in different parts of the workings; trouble with gas or the circulation of air in one section does not affect other sections of the mine; and finally, it is possible, then, to concentrate the entire circulation upon any single section of the mine, temporarily, by closing down other unaffected sections till the trouble is overcome.

QUES. 26.—Explain the action of a pulsometer pump. *Q. 18.

ANS.—Steam under a pressure capable of forcing water to the desired height is admitted by an automatic valve, alternately, to two separate water chambers fed through valves by the same suction pipe. The pressure of the steam forces the water through discharge valves and up the delivery pipe to where it is discharged. At the moment when the water level in the chamber reaches the discharge valve the water and steam mix violently and the steam filling the chamber is instantly condensed, causing a vacuum; the steam valve is shifted by the excess of pressure in the other chamber. Steam is thus admitted to that chamber and performs the same work there, forcing the water from that chamber into and up the delivery pipe. At the same time, the atmospheric pressure forces water up the suction pipe and into the chamber where the vacuum was formed by the condensation of the steam. This action is automatic and continuous, alternating between the two chambers.

QUES. 27.—The plunger of a double-acting pump has a diameter of 12 inches and a stroke of 18 inches. If the pump makes 50 strokes per minute how many gallons of water will be discharged per hour, the efficiency of the water end of the pump being 80 per cent.? *Q. 4.

ANS.—The sectional area of a 12-inch plunger is $(.7854\times 12^2)\div 144 = .7854$ square feet. At a speed of 50 strokes a minute the plunger travels $50\times 18\div 12 = 75$ feet per minute, and the plunger displacement is $75\times .7854 = 58.9$ cubic feet each minute. Assuming an efficiency of 80 per cent. for the water end, this will handle $60\times .80\left(\frac{58.9\times 1{,}728}{231}\right) = 21{,}150$ gallons per hour.

QUES. 28.—Is the double notch better than the single notch for ordinary entry timbers? Give reasons. *Q. 9.

ANS.—For ordinary entry timbering the single-notch method is generally preferred, because it is simple, more easily made, and does not weaken the collar where it bears on the leg, as is apt to be the case where the double-notch method is employed. The single notch is better adapted to timbering where the roof pressure is the greater; but for a heavy side pressure, the double notch is often used to advantage.

TEST OF DOUBLE-INLET SIROCCO FAN

The following test was conducted at the Alice mine of the Monongahela River Consolidated Coal and Coke Co., November 6, 1910, under the supervision of Mr. S. A. Scott, General Manager of Mines M. R. C. C. and C. Co. Anemometer readings were taken by Mr. F. W. Cunningham, State Mine Inspector, 21st District, and Mr. Wm. G. Louder, Mine Inspector, M. R. C. C. & C. Co. Velocity readings were checked with Pitot tubes by Mr. T. Chester and Mr. F. Thorp, of the American Blower Co., also checked with "Hydro" recording velocity and volume gauge by Mr. H. Bacharach, M. E. Pressure readings were made with water gauge, checked with gasoline gauge and "Hydro" recording gauge by Mr. G. F. Osler, chief engineer M. R. C. C. and C. Co., and Mr. T. W. Fitch, Jr., of American Blower Co. Power and speed readings were taken by Mr. C. S. Davies, superintendent of electrical equipment M. R. C. C. and C. Co., and Mr. U. U. Carr, mechanical engineer of the same company. The following table shows results:

Fan Wheel 8′ 6″ diameter × 5′ 8″, driven by Morse Chain Transmission from 200 H. P. Western Electric Co.'s Motor. Fan exhausting during Test.

	Test 1	Test 2	Test 3
Time of test	1:30 to 1:45	2:55 to 3:10	3:26 to 3:41
Revolutions of fan per minute	220	248	275
Amperes	225	330	503
Volts	546	539	522
Horsepower at input	164.5	238	352
Efficiency of motor	92%	92%	90%
Efficiency of chain drive	95%	95%	95%
Horsepower delivered to fan	144	208	301
Net area of airway, square feet	98.5	98.5	98.5
Velocity of air, feet per minute	1,710	1,843	2,232
Volume of air, cubic feet	168,500	181,536	219,852
Water gauge at fan	4.3″	5.4″	6.75″
Horsepower of air	114	154.5	234
Mechanical efficiency of fan	79%	74%	78%
Mechanical efficiency of unit	69.4%	65%	66.5%
Peripheral speed, feet-minutes	5,870	6,625	7,340
Theoretical water gauge	4.37″	5.58″	6.85″
Manometric ratio	98.5%	97%	98.5%
Contents fan wheel, cubic feet	322	322	322
Volume per revolution, cubic feet	767	733	799
Volumetric ratio	238%	228%	248%
Test of Fan Closed			
Revolutions of fan per minute	222		
Water gauge	5.54″		
Theoretical water gauge	4.46″		
Manometric ratio	124%		

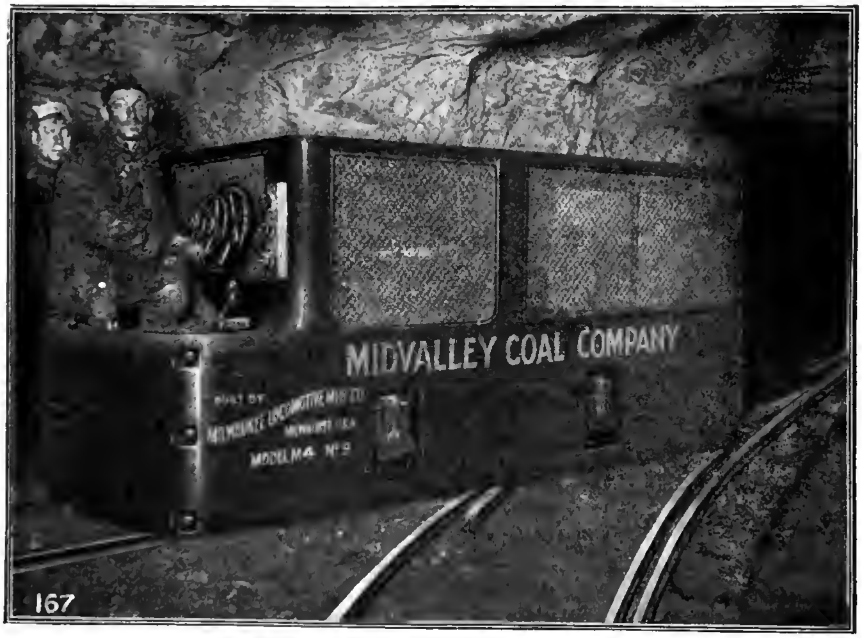
George H. Hollingsworth, F. G. S., says in page 406, of the Practical Engineer Pocketbook: "Such results as 75 per cent. of the indicated horsepower being realized in work done upon the air are manifestly impracticable." The Sirocco fan seems to have a habit of upsetting theories in fan efficiencies.

GASOLINE LOCOMOTIVE FOR MINE USE

Written for Mines and Minerals

Results of Experience at Mines of the Midvalley Coal Co., With a Milwaukee Gasoline Locomotive

In January, 1907, Editor H. H. Stoek, inserted the following in Mines and Minerals: "A large number of inquiries have been received for the address of an American manufacturer of gasoline mine locomotives. If manufacturers will communicate with the editor they will be placed in touch with inquiries from time to time." In August, 1910, there was an article on the gasoline mine locomotives used in Austrian and German mines which was very productive of inquiries concerning the manufacturer and where they were in use in this country. While the article was being written a locomotive was on the way to the Midvalley Coal Co., Wilburton, Columbia County, Pa., that was built by the Milwaukee Locomotive Mfg. Co. In the October, 1910, issue of Mines and Minerals mention was made of the first gasoline mine locomotive in the United States replacing five mules and a steam locomotive. At that time, while the Milwaukee gasoline locomotive was working satisfactorily, the Midvalley mine management considered the matter in the light of an experiment and therefore were reluctant to furnish more than general information.



Milwaukee Gasoline Mine Locomotive

Since that time a second locomotive has been installed and a third one ordered, thus showing conclusively that this company is satisfied with the results. Several beds of coal are worked at Wilburton necessitating the maintenance of a haulage system for each bed as well as each level. Every coal miner knows that the haulage system at a colliery must be flexible to comply with the demands of the colliery, also that so much depends on local conditions that scarcely at any two collieries is it possible to haul coal at the same cost figures with the same kind of motive power, yet in a general way comparisons are made which approximate theory, and are of use when estimating on the installation of new haulage plants. Probably the most accurate comparisons between haulage systems are those made at a mine where one system has been substituted for another. The management of the Midvalley Coal Co. kindly furnished data, which showed a saving of 32.2 per cent. on coal hauled by substituting the gasoline locomotive for mules.

To the writer this comparison seemed too conservative, because the locomotive was in a position where its full capacity could not be demonstrated, in fact was idle a large part of the time, making only 24 miles per day when it is capable of doing more than twice as much. The management estimates that the new locomotive ordered will save practically 50 per cent. over the present system of haulage as it is to be placed on a level where it will be supplied with sufficient work to keep it moving, and do the work of 15 mules.

By comparing the cost of haulage with the gasoline locomotive at Midvalley and the average cost of electric locomotive haulage as furnished in the Coal and Metal Miners' Pocketbook, it was found that there was a lessened cost of 27.9 per cent. in favor of gasoline. The Midvalley Coal Co.'s locomotive shown in Fig. 1 uses naphtha for fuel, it being less dangerous and better to handle than gasoline. The consumption of naphtha is about 15 gallons per day ($1.50 per day), where if gasoline were used the consumption would be about 12 gallons for the same work while the cost would be 30 cents more.

Judging from the odor produced by automobiles it would be supposed that locomotives of this description would foul the mine atmosphere. To prevent this disagreeable feature, a series of parallel pipes perforated underneath are fitted in a steel tank and then submerged in a chemical solution of chloride of calcium. The exhaust from the engine passes into these pipes, then through the perforations into the solution where all flame is extinguished and the gases of combustion cooled to about 100° F.

The mine inspector after careful examination was unable to detect any odor in the mine atmosphere that could be attributed to the locomotive. The Midvalley locomotive, rated as a 9-ton locomotive, has about the following dimensions: Length, 150 inches; width, 59 inches; height, 60 inches; wheel base, 48 inches; and diameter of driving wheels, 24 inches. The following table gives the detail of the average work performed daily by the first locomotive in 6 months, during which period approximately 2 hours out of a 9-hour day were devoted to switching, a feature which fails to show on the cost sheet:

Work Performed by Midvalley Locomotive

Average tonnage of loaded cars per day	550 tons
Average tonnage of empty cars per day	250 tons
Average mileage of loaded cars per day	12 miles
Average mileage of empty cars per day	12 miles
Weight of one loaded car	5½ tons
Weight of one empty car	2½ tons
Average number of cars per train	8 cars

Cost of Operation

Wages of operator and helper per day	$3.35
Cost of fuel per day	1.50
Cost of lubricating oils per day	.12
Consumption of fuel in gallons daily	15

Maintenance

Cost of maintenance of locomotive for six months, including repairs and labor	$65.14

The Imperial Royal Mine Chief Schardinger, of Austria, issued orders on January 5, 1907, concerning the use and working of traction explosion motors in mines in Austria. In Article II, Section 2, of this order the following will be found:

"In mines subject to firedamp such locomotives may, as a rule, be permitted for hauling only in mines in which the air-current does not contain mine gas in excess of .5 per cent. and which has not already passed through other parts of the mine. Exceptions are permissible in the latter case when the

supply of a perfectly fresh air-current would involve disproportionate costs; further, the quantity of air supplied by the air-current in question corresponds to the requirements of Sec. 9 (Abs. der h.) (a Verordnung Z. 3142/02) without consideration of the deviations permitted in Sec. 9 (Abs. 2, bez. Sec. 15, Abs. 3); and finally, if the employment of the machine hauling in the mine diggings which are aired by the air-current conducted from the local workings seems justified by the amount of hauling.

"The mine levels traversed by the locomotive must be examined in every level by the district overseer with the safety lamp, and in the event of the presence of mine gas, the locomotive traffic must be stopped. Once a week the mine gas contents of the air-current in the levels traversed by the locomotive must be established by the Pieler lamp or by analysis. In such case the result of the examination must be noted in the mine book (Befahrungsbuch).

"In mines subject to firedamp the mine levels traversed by the locomotive must be kept so moist by means of sprinkling water conduits, that dangerous coal-dust deposits in the same are constantly prevented; revocable exceptions to this may be allowed by the district mining office in so far and for as long as the mines in question are naturally moist."

ELECTRIC SHOCK IN MINES

Written for Mines and Minerals, by Syden F. Walker

(Continued from March)

Footwear and Shocks. Relative Safety of Armored and Unarmored Conductors

An important matter bearing upon the question of shock, is the footwear. A striking instance of this occurred at a colliery in Scotland, in connection with the haulage rope of an electrically driven coal-cutting machine. A man wearing leather boots trod upon the haulage rope and was killed. Another man wearing clogs trod upon it, and only received a slight shock. The important point to be noted here is, that the clogs offered a very much higher electrical resistance to the passage of the current through them than the leather boots. Leather soles, besides being thin, and sometimes of very poor material, are very much more porous than wood. At any rate there is a better chance of a larger quantity of moisture being held in the pores of a leather sole, than in that of a wooden clog. In some tests that were made some years ago, by a British Government official, of the electrical resistance offered by the human body to electrical currents, it was found that changing the boots, and changing the socks, made a very great difference. A paper was read before the British Institution of Electrical Engineers on the subject, by the official in question, and he illustrated the lecture by showing measurements of his own electrical resistance, recorded upon an instrument provided for the purpose, when he had different foot wear. When wearing socks that had been in use the whole day, and were therefore fairly impregnated with perspiration, and wearing the boots he had also worn the whole day, and were therefore also fairly moist, a comparatively large current passed through his body. When he changed his boots, the electrical resistance offered by his body increased, and the current passing through him decreased proportionately. Changing the socks to dry ones still further increased the resistance. With men working in mines, this has a very important bearing upon the question of shock. It often happens that men have to repair roads which are very wet. Also the temperature is often so high that they work with very little clothing upon them, and some of the most serious accidents that have taken place, have been due to these two causes. In one case, which occurred some years ago, some men were repairing a dip, down which a fairly continual stream of water was flowing. The water was not sufficient to stop their working, but it kept their boots in a moist state. They were also working without coats, and with very little upon their bodies. There was a cable on the side of the road, supported by insulators attached to the props carrying current to a dip pump farther toward the face. The repairs necessitated the removal of a steel H girder. In removing the girder, its sharp edge cut through the insulation of the cable, which was not armored, the girder becoming alive. The man who was handling the girder received a shock, which brought him to the ground, and a second man catching hold of the body of the first man, also received a shock. The first man was killed. Somewhat similar cases of this kind have been fairly frequent throughout British coal mines, even up till quite recently.

The question of armored versus unarmored cable has been very fully and occasionally somewhat hotly discussed. In the writer's opinion, amoring as a rule is not wise, but he would have, in place of armor, a thick covering of the insulating material, and a further thick protecting covering of some substance such as jute, the jute being rendered as fully waterproof as possible. The argument in favor of armor is, if a fall of roof occurs, the armor may probably prevent a breakage, and in a great many cases it may prevent the skinning of the insulating envelope of the cable. This is quite correct, but the reasoning is based upon the badly insulated cables that were so much used in the early days of electric light and power in mines, owing to the efforts that were made to keep the cost down, and that are now even sometimes employed, owing to the mistaken policy of the purchasing agent. The mine manager sends in a requisition for a certain quantity of cable of a certain section, and he requires it to be insulated in a certain way. The clerk who sends out for tenders, either allows himself to be persuaded by a plausible traveler, that the very much cheaper form of cable than the colliery manager has requisitioned will answer, or he takes it upon himself to save, as he thinks, his firm a certain sum, by buying a cable whose insulation is much thinner, and to which there is very little protection. Such a cable succumbs to every trifling accident. A horse may bite through the cable insulation and be killed, the sharp edge of a piece of stone falling from the roof will skin a place on the cable, which is probably not noticed, and may lead to some one receiving a fatal shock. A bell wire may accidently be rubbed against the cable, or a haulage rope, a haulage pulley, the steel edge of a mine wagon, etc. In all cases the thin insulating envelope is quickly cut through, and either the cable is left bare, and a source of danger by itself, or as more frequently happens, the conductor makes connection with the metallic body, the wire or the rope, or the pulley that has rubbed through it, and these objects become part of the electrical service of the mine, that is, become "alive" as it is termed, and transmit their killing properties to every metal body with which they come in contact. This is the source of a very large number of the accidents that have been recorded. Mine managers, finding that they have difficulty in obtaining thickly insulated, and thickly protected cables, naturally order armored cables, since the armor does give some protection. The purchasing agent is rarely able to take upon himself the altering of an order from armored to unarmored, and, consequently, armored cable has more and more come into use in mines. In the writer's view, armoring a cable introduces considerable danger. There is always the possibility that a fall of roof will drive the armor through the insulating envelope on to the conductor. The armor then becomes "alive," and transmits its killing properties to every metal with which it comes in contact. Further, it is more dangerous when it becomes "alive" in this way, because of its supposed innocence; no trouble is usually taken to prevent the armor of a cable from touching other metals, and, consequently, if the armor is "alive," they become "alive," and a whole series of connections may be made to different metallic bodies about the mine, any one of which when touched may give a fatal shock. The reply to the advocates of armor is, that if

the armor is "earthed or grounded," and it becomes "alive," the fuse or circuit breaker at the switchboard will cut off the conductor, and therefore the armor will remain harmless. Unfortunately this only introduces a new source of danger. "Grounding" is a very difficult matter in coal mines, and assuming for the moment that efficient "earth" is obtained, and that the armor is efficiently connected to "earth," if a fall occurs, and the armor and conductor are both severed, unless the armor as well as the conductor is fully jointed, that is to say, the joint in the armor is made to offer very little resistance indeed, the armor beyond the joint, if it becomes "alive," as it may easily do, is a source of danger. This will be seen in the following case. Suppose the conductor to be jointed, the joint covered, but the armor not jointed, or only imperfectly so; every one about the mine supposes that the armor is perfectly harmless; but suppose a fall occurs at some point beyond the joint, again driving the armor through the insulation, and making it "alive," any one now touching the armor will receive a shock that may be fatal.

In the writer's view, the only method which thoroughly provides efficient protection, where armoring is employed, is the concentric system of distribution, in which one of the cables forms the outer conductor, and its continuity is absolutely necessary to the continuance of the electrical service. The concentric system, which has been efficiently worked out by an eminent firm of Scotch electrical engineers, has been principally applied to continuous currents, though it has recently also been applied to three-phase services. Its working with continuous currents is very simple. One conductor forms the center of a cable, and is insulated in the usual way, its insulating envelope being protected by layers of jute or some similar substance. Outside of the layers of jute the second conductor is laid, concentric with the first, and the outer conductor is not insulated, but left free to make contact with everything it touches. Further, it is rendered of at least equal conducting power to the inner conductor. In the system mentioned, a lead tube assists to protect the insulating envelope of the inner conductor, and in some cases the outer conductor is also armored, but the armor and the conductor form one conducting mass. With this cable, if a fall occurs, and the cable is severed, it is absolutely necessary for both the inner and the outer conductors to be jointed so efficiently, that neither joint offers an appreciably increased resistance to the passage of the working current above that of the cable as it was before the fall. It may be mentioned incidentally that the resistance of any "earth" path is of very great importance in the question of shock, and in the matter of carrying the return current in such a manner that fuses or circuit breakers will operate. It will be seen that the working of the concentric system ensures that the outer conductor, and the armor, or whatever may be employed to protect it, shall always be continuous, and therefore that protection is always provided, so far as an "earthed" armor can provide it.

The concentric system has been adapted to three-phase service by one of the three cables being employed as outer conductor, much in the same manner as the rails of some of the electrical railways using three-phase currents are. The protection afforded by making one of the three cables an uninsulated outer, with three-phase working, is not so perfect as that offered by the uninsulated outer of continuous current cables, but it is far better than the protection afforded by the usual armor of the ordinary three-phase cable. The common arrangement of three-phase cables, as employed in the United Kingdom, is that the three conductors are separately insulated, the insulating envelope of each being slightly protected by yarns. The three insulated cables are then laid up into a sort of rope, around a heart of jute or similar substance. A filling of jute is added to make the rope into a cylinder, and the whole is insulated outside of all, armor usually being laid on outside of the insulating envelope. One of the peculiarities of the three-phase system is that a single cable must not be armored This applies equally to single- or two-phase systems, but neither of these are now in use, in British collieries at any rate. If a single cable carrying alternating currents is armored, the armor makes a very heavy charge upon the currents passing through the conductor, owing to the useless currents which are generated in it, and to the changes of magnetism which are also created in its molecules. A favorite form of three-phase cable for mining work in British collieries consists of three paper-insulated conductors, laid up together as explained above, the whole being drawn into one lead tube, and armoring laid on outside of the tube. It is not permissible to use separate paper-covered cables carrying alternating currents, for the same reason that separate cables may not be armored. The ordinary three-phase armored cable, the armor being "earthed," is subject to the same objections as the continuous-current armored cable. It is usually safe, so long as the armor is "earthed," and if the neutral point of the system is "earthed," but in case of the cable being severed by a fall, unless the armor is jointed as fully as the cables, a very difficult matter indeed, seeing that three joints have to be made inside the armor, besides the joint of the armor itself, and that the joints have to be made under very difficult conditions in a mine, the armor may become "alive" beyond the break, and any one touching it may receive a fatal shock, or it may transmit its killing properties to other metallic bodies about the mine, as described in connection with continuous currents

CATALOGS RECEIVED

In writing for catalogs, please mention Mines and Minerals

American Spiral Pipe Works, Chicago, Ill., Water Powers of the Third Magnitude, 15 pages.

E. I. Du Pont de Nemours Powder Co., Wilmington, Del., Du Pont Smokeless Shotgun Powder, 12 pages.

Electric Service Supplies Co., Philadelphia, Pa., Peerless Armature Tools and Car Barn Appliances, 16 pages; Garton-Daniels Lightning Arresters, 64 pages.

General Electric Co., Schenectady, N. Y., Bulletin 4787, Wires and Cables, 73 pages; Bulletin 4800, Direct Current Motors, Type CVC, 22 pages; Bulletin 4811, Drum Controllers for Industrial Service, 12 pages, Bulletin 4812, Small Direct Current Generators, Belted, Type CVC, 10 pages; Bulletin 4813, Type F, Form P, Oil Break Switches for Pole Line Service, 4 pages; Bulletin 4814, Field Rheostats, Type F, 16 pages.

Hendryx Cyanide Machinery Co., 107 William Street, New York, N. Y., Catalog No. 7 describing Cyanide Machinery 24 pages.

Hazard Mfg. Co., Wilkes-Barre, Pa., Hazard Special Plough Steel Wire Rope, 12 pages.

John Simmons Co., New York, N. Y., Baldwin Acetylene Mine Lamp, 20 pages.

The Trenton Iron Co., Trenton, N. J., The Application of Wire Rope to Surface and Underground Haulage, etc., 65 pages.

Rockwell Furnace Co., New York, N. Y., Bulletin 27, Rivet Heating Furnaces, 8 pages; Bulletin R, Combination Tool Room Gas Furnace, 4 pages; Bulletin T, Moyer Tramrail in Modern Foundry Practice, 8 pages.

Milwaukee Locomotive Mfg. Co., Milwaukee, Wis., Bulletin 101, Milwaukee Locomotives Gas Driven—Mining Type, 24 pages.

Vulcanite Portland Cement Co., Philadelphia, Pa., Pamphlet No. 7, Cement Sidewalk Paving, 32 pages.

The Montague Compressed Air Co., St. Louis, Mo., The Obear Air Lift Pump, 4 pages.

Carnegie Steel Co., Newark, N. J., Waverly Warehouses Stock List No. 6, 50 pages.

Link-Belt Company, Chicago, Ill., The Story of Two Coal Washeries, 12 pages.

Stromberg-Carlson Telephone Mfg. Co., Rochester, N. Y., Folder describing Code No. 896 Compact Type Magneto Telephone.

Holman Bros., 114 Liberty Street, New York, N. Y., Catalogs describing Holman Rock Drills; and folders on the Holman Prize Winning Rock Drill, Chapters I and II, also extracts from the South African Mining Journal on Results of the Stope Drill Contest.

FIRST-AID CONTEST IN WASHINGTON

Written for Mines and Minerals

Dr. M. J. Shields, surgeon of the field branch of the National Red Cross Society, and the "Father of First-Aid" work in the coal mines of the United States, was at Roslyn on February 6

First-Aid Team, Northwestern Improvement Co., Cle Elum, Wash.

and 7 to take part in the first annual first-aid contest of the teams of the Northwestern Improvement Co., which company operates in the Roslyn coal field.

On the evening of the 6th, Doctor Shields gave an instructive lecture on first-aid work with demonstrations of methods of handling persons overcome with gas, smoke, or electric shocks, and in adjusting splints, bandages, and tourniquets, both with volunteer subjects from the audience and by lantern-slide pictures.

This was the doctor's first trip to the coal mines of Washington, and he expressed his surprise and delight at the progress of first-aid work already made by the Northwestern Improvement Co. and their employes in the Roslyn field.

On the afternoon of the 7th, picked teams from each end of the field contested against each other for a silver cup offered by the Northwestern Improvement Co. as a permanent trophy to be contested for annually. The team representing the Roslyn and No. 3 mine was comprised of A. G. Lindsey, captain, L. Lloyd, John Heathcock, Jas. Farrington, Wm. Smith, Geo. Morris, Adam Stewart, John Graham, Robt. Bell, Tony and Nick Stanfel, Martin Coleman, and Wm. Ward, while John Heathcock, Jr., Frank Norta and Isaiah Greenhalgh were the subjects operated on in the various events. The lower end of the field, including the Cle Elum, No. 5 and No. 7 mines, was represented by John Hutchinson, captain, Dave Taylor, John Parker, Dan Murphy, Thos. Daglish, Jas. Pascoe, Ben Russell, Dan Shields, Thos. Summerill, E. McGilley, Chas. Chabert, Wm. Reay, Fred Smith, and Chas. Lee, with Wm. Smallwood and Thos. Choice as subjects.

Dr. E. W. Stimpson, chief surgeon of the Roslyn-Cle Elum Beneficial Association, who trained the men and deserves full credit for their splendid showing, acted as field manager and announced the events. Doctor Shields and D. C. Botting, the state mine inspector, acted as judges, and J. W. Anderson and J. B. Warriner, company mine inspector, acted as time keepers.

The contesting was spirited and close, and the skill of the teams provoked round after round of applause from the spectators. None but an expert could have detected any difference between the work of the contestants, and Doctor Shields was moved to admit that it was the closest contest he had ever had to judge. The cup was finally awarded to the Roslyn team by the narrow margin of 2½ points, due to their greater speed in the third event, which they completed in the remarkable time of 3 minutes and 53 seconds. They will, therefore, hold the silver cup for 1 year, and will each receive individual bronze medals setting forth facts of their victory.

First-aid work is taught at the company's Red Lodge mines in Montana, and General Manager Claghorn is arranging to have an interstate contest in the near future.

OBITUARY

GEORGE SHARP RAMSAY

George Sharp Ramsay died at St. Mary's, Pa., February 21. He was in apparent good health in the morning and it is supposed that an attack of acute indigestion affected the heart action to such an extent as to cause his demise. Mr. Ramsay was born in Dunfermline, Scotland, in 1857; came to America with his parents in 1863 and after attending public schools went to work in the mines. His wide experience made him one of the best informed coal-mining men in this country. During his career he held positions in West Virginia, Colorado, Wyoming, and British Columbia, and for 15 years prior to his death was the chief in the line of the development of the Shawmut Mining Co.'s properties.

George S. Ramsay was a man of sterling worth, his life was clean in and out of business, he was scrupulously honest with his fellow men in all things, and he had high estimates of those things that tended to an incorruptible and noble character. He was a type of man whom everybody admired, and his pleasing personality, his sympathetic impulses and high sense of honor won for him the confidence and esteem of the general public as well as establishing a friendship among his associates that will not soon be forgotten. George S. Ramsay was one of God's true noblemen and it is to be regretted that the life work to which he was so eminently fitted should be concluded before its intended achievements had opportunity of maturity. In the circles to which Mr. Ramsay devoted his best energies, he was regarded as a man of sound judgment, and in his particular line he was perhaps the best informed and most capable official of the present day.

George S. Ramsay

CHARLES T. DENNISON

Dr. Charles T. Dennison, President of the Board of Trustees of the California State Mining Bureau, died in San Francisco, February 24, aged 70 years.

MAJOR HEBER S. THOMPSON

Major Heber S. Thompson, age 71 years, for many years chief engineer for the vast coal lands and other properties of the Girard Estate in Schuylkill and adjoining counties, died March 9, at Pottsville, Pa., of general debility. He was a First Defender during the Civil War and rose from a private in the Forty-Eighth Regiment, P. V. I., to command of a battalion in the regiment.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

WESTERN EDITORIAL OFFICE: 728 Equitable Building, Denver, Colo.
CHICAGO OFFICE: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to MINES AND MINERALS, Scranton, Pa.

CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER........MANAGER
EUGENE B. WILSON, SCRANTON, PA........EDITOR
GEORGE F. DUCK, E. M., DENVER, COLO........WESTERN EDITOR
P. G. MOORE........CIRCULATION MANAGER
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., LTD., DRYDEN HOUSE, 43 GERRARD ST., SOHO, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 PITT STREET, SIDNEY.
SOUTH AFRICAN AGENT: E. HERBERT BYSSHE, CAPE TOWN, MANSION HOUSE; JOHANNESBURG, TRUST BUILDING; DURBAN, NATAL BANK CHAMBERS; BLOEMFONTEIN, FLORENCE BUILDINGS.

SUBSCRIPTION RATES

Single Copies........25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada........2.50
To all other countries........3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always gives your FORMER as well as your present address.

TABLE OF CONTENTS

PECK'S SHAFT BREAKER, by E. B. Wilson........ 513
STEEL MINE SHAFT CONSTRUCTION, by R. B. Woodworth........ 516
THE AEROLITH RESCUE APPARATUS........ 521
THE MARIANNA COAL MINES, by Hartley M. Phelps........ 523
GEOLOGY OF HERRIN QUADRANGLE, by T. E. Savage........ 527
POSSIBILITIES OF A NEW LIABILITY LAW, by Sim Reynolds........ 532
SELF-DUMPING CAR HAULS........ 536
FIRST AID FOR INJURED SPINES, by Thomas C. Harvey........ 538
COAL PRODUCTION OF WORLD, 1908........ 538
EXAMINATION QUESTIONS—
Selected Questions, Iowa Mine Foremen's Examination, Des Moines, October and December, 1910........ 539
TEST OF DOUBLE-INLET SIROCCO FAN........ 541
GASOLINE LOCOMOTIVE FOR MINE USE........ 542
ELECTRIC SHOCK IN MINES, by Sidney F. Walker........ 543
CATALOGS RECEIVED........ 544
FIRST-AID CONTEST IN WASHINGTON........ 545
OBITUARY
George Sharp Ramsay—Charles T. Dennison—Heber S. Thompson.. 545
EDITORIAL—
"Lo the Poor Prospector"—A Model Coal Mine Act—Geologists and Nature Fakirs—The Cokedale, Colo., Mine Explosion—A New Book Catalog........ 546
BOOK REVIEW........ 548
PERSONALS........ 549
ENGINEERING SOCIETIES........ 549
SELLING ZINC ORE ON CONTRACT, by Lucius L. Wittich........ 550
MINING IN THE TINTIC DISTRICT OF UTAH, by Leroy A. Palmer........ 553
REVERBERATORY COPPER SMELTING, by E. B. Wilson........ 557
COTTRELL FUME-CONTROLLING DEVICE........ 561
AIR FROM HYDRAULIC AIR COMPRESSOR........ 561
CORRESPONDENCE—
Safety in Mines—Warnings to Colliers—Rope Haulage—Steel Supports in Mines........ 562
ORE NOTES........ 564
THE MINING INDUSTRY OF NICARAGUA, by T. Lane Carter........ 566
THE INTERNATIONAL CYANIDE TRUST........ 568
MINING IRON UNDER THE SEA, by H. W. Buker........ 569
VALDEZ CREEK PLACER DISTRICT, by F. H. Moffit........ 572
AMETHYST COLORATION OF GLASS........ 573
IRON ORES IN THE SOUTHWEST, by C. Colcock Jones........ 574
TRADE NOTICES........ 576
NEW INVENTIONS........ 576
SQUIBS—
Gas in Very Hot Flames, 522—Coal and Other Exports, 526—Coal Discovered in Territory of Magellan, 531—Safety Lamps on Battle Ships, 535—Solubility of Gases in Water, 552—Silicon Steel, 552—American Tin Mining in Bolivia, 568—Route of United States Rescue Car No. 4........ 573

"LO THE POOR PROSPECTOR"

FIRST the buffalo, then the "wild" Indian, and finally the "poor prospector" has been gathered unto his fathers. But the "poor prospector," like "an infant industry" when any change in the tariff is broached, refuses to stay dead and bobs up serenely at the first attempt to improve upon our antiquated, and some darkly hint iniquitous, system of administering the public domain. How those words "public domain" roll from the tongue of the spell binder and from that of him who is out of office, and what tears can be evoked by a touching reference to the "poor prospector." The public domain is fast being absorbed and the poor prospector now travels in a Pullman, and what will the land grabber do for something as an appeal to the emotions when both are history.

A MODEL COAL MINE ACT

RECENTLY the editorial staff of MINES AND MINERALS felt compelled in the interest of fairness to those directly concerned to criticize a proposed mine law. They now take pleasure in commenting on a mine law recently passed in British Columbia. Owing to "State Rights," a particularly important phase of mine legislation has been sadly neglected in the United States, that is the part in which the general public are concerned.

If a bill is partial to the miner it is objectionable to the operator; if it is partial to the operator it is objectionable to the miner; if it favors both operator and miner it may be objectionable to the general public, who are the ones most interested. In order to draft a just and satisfactory mine law all parties concerned must be consulted and their suggestions combined as far as practicable.

The Honorable Premier Richard McBride, also Minister of Mines for the Province of British Columbia, recognizing the importance of fulfilling the three conditions imposed, withdrew the bill of 1910 for 1 year, in order to make the act as perfect as practicable. In the interim he sent his deputy with the Chief Inspector of Mines to the various coal-mining districts to confer with both operators and miners and obtain suggestions for changes or additions to the bill of 1910. The outcome was the 1911 bill, which, with three or four minor amendments, was passed as presented by the Premier.

Wm. Blakemore, a mining engineer, writes editorially in *The Week* congratulating the Premier, as Minister of Mines, on the able and exhaustive speech with which he introduced the measure, saying, among other things: "It is not a little surprising that a man of so many activities and responsibilities should have been able to acquaint himself to such a remarkable extent with the technicalities of mining and the details of the bill."

J. H. Hawthornthwaite, leader of the Socialists of the Province, although of the opposition, congratulated

the government on the spirit shown in weighing the suggestions offered from the opposition side for the improvement of this bill, and accepting such of these as were deemed to possess merit. "The bill passed was," he believed, "as strong and good a piece of practical and necessary legislation as it was possible to devise, and one of the best bills of its character in the world. Upon it the House and the entire country were entitled to congratulation. Having passed this law, it now only remains for the government to see that it is effectually carried out, which, after all, is the main thing."

Here we find the people, the operators, and the miners satisfied, a condition which no politically fostered mine legislation can bring about.

GEOLOGISTS AND NATURE FAKIRS

SHORTLY after the death of Herbert Spencer a critic observed that he had wasted his energies, money, and time in trying to impress his ideas on origin and the hereafter on people who were perfectly satisfied that they were born and had to die.

An atheist took the critic to task, and among other acrimonious remarks, said the man was not capable of criticizing so great a philosopher.

The reply was: "Spencer theorized on some matters concerning which he knew absolutely nothing, and as I am equally learned in them I certainly am able to criticize him as a 'nature fakir.'"

We wonder if when Colonel Roosevelt used the expression "nature fakir" he had in mind the practical geologist with a pet theory on ore deposits, as well as the zoologist, for it is this uncertainty, coupled with a paragraph in Doctor Peters' "Copper Smelting," that has discouraged us from furnishing some engineering society with our "Résumé of Ore Deposits."

Doctor Peters' remark was to the effect "that the practical man had infinitely more theories than the theoretical man, but the trouble was they were all wrong."

When about to become a theoretical geologist and avoid the stigma placed on the practical geologist, our supposed friend, W. R. Ingalls, informs us through an editorial that the practical geologist is not justified in taking up the time of a mining society in riding some inane hobby, or words to that effect. So again we are in a quandry how to class ourself. Since the champion geological scrapper became a "nester" there have been few public encounters; therefore, the patriarchs delight to discourse on the good old sport which originated when Cain slew Abel with a rock.

The first geological set-to in the United States took place at Poker Flats, Cal., and was reported by Bret Harte as follows: "A piece of old red sandstone hit him in the abdomen and curled him on the floor, where subsequent proceedings interested him no more."

The second memorable geological sporting event occurred shortly after the San Francisco earthquake, and if memory is not at fault, was between Tobasco Tom and Dosalane sub Rang. While the function culminated in a riot of words, the feldsparing was by far the best witnessed. There seems to have been no available place since Tobasco won the championship where a geological event could be held, except at the clubs of the mining societies, and these are about ready to limit such sports, on the grounds that while they are entertaining to some, it is immaterial to the majority whether the fracas is brought about by the Elijah thermal or Dives meteoric method of deposition. This restriction does not signify, we are informed, that papers of this kind will not be received and printed, for undoubtedly the councils know as well as others that there is much to learn about ore deposits before the subject becomes a science. It does signify, however, that where society meetings are held at long intervals, practical papers that will be helpful in producing results are to be given preference.

We desire to emphasize that these comments have no reference to geological papers that remain inside the realm of dubiety, but merely to those papers which are composed of ideas evolved during sedulous thought, and which one can neither disprove nor contra prove, not having been around at the time. There have been presented to institutes many practical geological papers that record observations and impart information on the district described in a way that allows deductions to be made and applied to other districts. The authors of such papers should receive favorable comment and thanks from all engineers. However, we still hesitate to advance our Résumé of Ore Deposits.

THE COKEDALE, COLO., MINE EXPLOSION

ACCORDING to the evidence adduced at the coroner's inquest, the explosion at the mine of the Carbon Coal and Coke Co., at Cokedale, Colo., about 9 o'clock on the night of February 9, appears to have been due to the ignition of an undetermined quantity, certainly as much as 25 pounds, of 40-per-cent. dynamite in charge of the shot firers working. Owing to the rapidity with which shot firing was going on all over the mine and the consequent amount of dust produced thereby, as well as to the open and connected character of the workings and to the large volume of air in circulation, there appears to have been a great quantity of dust of strong coking quality in suspension in the air. This was ignited by the inflammation of the dynamite and the resultant dust explosion swept practically throughout the entire workings bounded by the First and Eighth West B entries and the Third and Fifth South entries.

Investigation has shown that the systems of mining, ventilation, watering and shot firing in use, while stamped with the approval of the best practice here and abroad, each contained an unsuspected "weak"

feature, and these weaknesses being in evidence or in force all at the same instant brought about the accident in question.

Owing to lack of space and to the fact that certain radical changes are being made at the mine as a result of the experience gained at this accident, the extended discussion of the subject will be left for a later issue.

A NEW BOOK CATALOG

FOR the convenience of our readers who may want the most up to date and reliable books on Mining, Mineralogy, Metallurgy, Prospecting, and Assaying; and on Mechanics, Electricity, Chemistry, etc. as related to mining and metallurgy, we have prepared for the Technical Supply Co., of Scranton, Pa., a comprehensive and convenient catalog of "*books worth while*," which will be sent free on request to any reader on application to either Mines and Minerals or the Technical Supply Co.

The prices quoted in the catalog are strictly publisher's prices and any books ordered from the Technical Supply Co., will be sent postpaid. Orders received by Mines and Minerals will be promptly filled by the Technical Supply Co.

BOOK REVIEW

Colliery Guardian Co., 30 Furnival Street, Holborn London, E. C., have printed abstracts on American Coal Dust Experiments conducted at the Pittsburg Experimental Station, and on the Austrian Coal Dust Experiments conducted at Rossitz Experimental Station. The prices of these abstracts are each 1s.

Practical Mineralogy Simplified, by Jesse P. Rowe, Ph. D. This book is intended for mining students, miners, and prospectors. It is divided into ten chapters and has tables for the determination of minerals. The writer realizes the number of splendid textbooks on Mineralogy that are now published, but from his long experience with mining students, miners, and prospectors not familiar with scientific methods, and with other college students in this kind of work, he realizes that there is ample room for a simple scientific work on the ores and minerals. There is no question but that a work of this kind will appeal to those who wish to obtain the practical part of Mineralogy. A little explanation of each mineral goes a long way with the prospectors who must use their eyes and make ordinary physical and elementary chemical tests. The book contains 155 pages, in addition to the mineral tables. In this work Professor Rowe limits the number of minerals to the more common of any class and does not deal with the extremely rare complicated minerals. It is believed that this method of presenting mineral information will appeal to a large number of those interested in metal mining. The price of this book is $1.25 net. It is published by John Wiley & Sons, New York City.

We are in receipt of Bulletin 2, 1911, West Virginia Geological Survey, Levels and Coal Analyses, by I. C. White. Part I is devoted entirely to levels, Part II to coal and coke analyses of the Pottsville series, Kanawha series, the Allegheny series, the Conemaugh series, the Monongahela series, and the Dunkard series. In addition there are statistics of coal production, coal areas of the several counties, and the classification of coals. Address I. C. White, Morgantown, W. Va.

Geology of the Iron River District of Michigan, by R. C. Allen, Director Michigan Geological and Biological Survey, Lansing, Mich. From a geological standpoint this is an exceedingly interesting book on the geology of the iron formation of Michigan.

Features of Producer Gas Power Plant Development in Europe, by R. H. Fernald, Bulletin No. 4, Bureau of Mines, Washington, D. C. This bulletin will be found exceedingly useful to those who are interested in the use of gas for power plants. It is stated that suction gas producers fed with anthracite or coke are in general use at the small power plants in Great Britain and on the continent. At the large producer gas power plants in Europe, bituminous coal is generally used both at those which save by-products and at those which do not. At a plant in Wales an installation of the by-product type comprises two pressure producers, each of 1,250 horsepower. At this plant an inferior coal is used containing 23 per cent. ash, 27 per cent. volatile matter, and 2 to 3 per cent. sulphur. Owing to the fact that the wholesale price of the sulphate of ammonia in the principal markets is from $55 to $60 a ton, the recovery of ammonia as a by-product in the manufacture of producer gas is a very tempting project abroad. Mr. Fernald states that one of the most interesting by-product installations in Europe is a 16,000-horsepower Mond gas plant at Dudley Port, South Staffordshire, England. He says that the leading manufacturers of gas producers are all working on suction plants for bituminous coal, but that each manufacturer places special restriction on the kinds of fuel that may be used.

The general situation in Europe regarding the use of low-grade bituminous coal is not much unlike the situation here. For the most part the better-grade coals are being used and the poorer grades are being left in the mines. The lignite or brown-coal briquets made in Germany form an excellent producer gas fuel and are in common use in the small suction plants, generally of the double zone type, that are put out by the various manufacturers. This brown coal closely corresponds to some varieties of the American lignite. The first peat producer gas plant stands in the center of a peat bog near Skabersjo, Sweden. Another peat-burning plant that is attracting the attention of engineers interested in producer gas development is at Visby, Gothland, Sweden.

Annual Report of the Board of Regents of the Smithsonian Institution for 1909, Charles D. Walcott, Secretary, Smithsonian Institution, Washington, D. C. The general appendix to the report contains high-class articles and brief accounts of scientific discovery in particular directions, and memoirs of a general character that are of interest and value to the numerous correspondents of the institution. There is an article on The Future of Mathematics, by Henri Poincare; What Constitutes Superiority in an Air Ship, by Paul Renard; Researches in Radiotelegraphy, by Prof. J. A. Fleming; Recent Progress in Physics, by Prof. J. J. Thomson; Production of Low Temperatures and Refrigeration, by L. Marchis; The Nitrogen Question From the Military Standpoint, by Charles E. Munroe; Solar Radiation Researches, by Jules C. Janssen; The Return of Halley's Comet, by W. W. Campbell; The Upper Air, by E. Gold and W. A. Harwood; The Distribution of the Elements in Igneous Rocks, by Henry S. Washington; The Mechanism of Volcanic Action, by H. J. Johnston-Lavis; Some Results of the British Antarctic Expedition, by E. H. Shackleton; Antiquity of Man in Europe, by George Grant MacCurdy; The Republic of Panama and Its People with Special Reference to the Indians, by Eleanor Yorke Bell; Ceramic Decoration, Its Evolution and Its Applications, by Louis Franchet; Some Notes on Roman Architecture, by F. T. Baggallay; The Relation of Science to Human Life, by Prof. Adam Sedgwick; Intellectual Work Among the Blind, by Pierre Villey; The Relation of Mosquitoes, Flies, Ticks, Fleas, and Other Arthropods to

Pathology, by G. Marotel; Natural Resistance to Infectious Disease and Its Reinforcement, by Simon Flexner; Conservation of Natural Resources, by James Douglas, in addition to several geographical articles.

BOOKS RECEIVED

United States Geological Survey Publications, Washington, D. C.: The Production of Anthracite Coal in 1909, by Edward W. Parker. United States Geological Atlas, Johnstown Folio, Pa., by W. C. Phalen, United States Geological Survey, Washington, D. C. This also furnishes an extra good idea of the coal fields in that vicinity, giving sections and geology. The price of this Atlas is 25 cents.

Map of Wood, Ritchie, and Pleasants Counties Oil and Gas Fields of West Virginia, including structural contours, West Virginia Geological Survey, Morgantown, W. Va.

Permissible Explosives, Miners Circular No. 2, by Clarence Hall, Bureau of Mines, Washington, D. C.

Oil Resources of California, by M. L. Requa, University of California, Berkeley, Cal.

The Prevention of Coal Dust Explosions by Zone Systems, by E. O. Simcock. Excerpt from the Transactions of the Institution of Mining Engineers, Newcastle-Upon-Tyne, England.

Some Relations Between the Composition of a Mineral and Its Physical Properties, by Prof. G. H. Cox and E. P. Murray, Vol. 3, No. 1, University of Missouri, School of Mines and Metallurgy, Rolla, Mo.

An Act to Consolidate and Amend the Coal Mines Regulation Act and Amending Acts, Province of British Columbia, Honorable Minister of Mines, Victoria, B. C., Canada.

Preliminary Review and Estimate of Mineral Production of Mines, William Fleet Robertson, Provincial Mineralogist, Victoria, B. C.

The California State Mining Bureau has issued a map of Whittier-Olenda Oil Fields, Louis E. Aubury, San Francisco, Cal.

Preliminary Report of the Oil and Gas Developments in Tennessee, by M. J. Munn. Address George H. Ashley, Nashville, Tenn.

The Farmers Handbook of Explosives, E. I. Dupont de Nemours Powder Co., Wilmington. Del.

PERSONALS

Mr. Wren, of the Canadian Pacific Railway, is receiving praise for furnishing a special train for the rescue party at the time of the Bellevue explosion.

Henry M. Lancaster, E. M., has removed his office from Wallace, Idaho, to 715–716 Hutton Building, Spokane, Wash., where he will continue the practice of his profession as mining and consulting engineer.

Sidney Norman has severed his connection with the Los Angeles *Mining Review*, having disposed of his interests in the paper to Mr. George H. Scott, formerly connected with Munsey and McClure magazines.

Ralph E. Davies, recently connected with the Montana School of Mines, has been appointed director of the Wisconsin State Mining School. Mr. Davies has had much practical experience in copper mining and is a proficient instructor.

W. R. Bauder has been appointed assistant general superintendent of the Breitung-Kaufman mining interests on the Marquette, Menominee, and Michipicoten iron ranges.

Frank Koester, sometime connected with the American Smelting and Refining Co., has opened an office at 115 Broadway, New York, as consulting engineer.

George E. Sylvester, of Rockwood, Tenn., has been appointed chief mine inspector of that state. He succeeds R. A. Shiflett, who has so ably performed the duties of that office, which in Tennessee includes all kinds of mines.

Warren R. Roberts, class of 1888, delivered a lecture at the University of Illinois on "The Engineering Features of Coal Mining and Handling."

James Epperson (Rep.), who for 16 years past has successfully filled the office of chief mine inspector in Indiana, will be succeeded March 11 by one of his able assistants, Frank I. Pearce (Dem.), of Brazil, Ind. This is an aftermath of the clean sweep the Democrats made last November in Indiana, when they elected all the state officers and an overwhelming majority in the state legislature.

Edgar M. Tuttle, E. M., of the Department of Mining and Metallurgy, New Mexico School of Mines, has changed his eastern address from 30 E. Logan St., Germantown, Philadelphia, to 1024 E. 9th St., Brooklyn, N. Y. He will unless he receives a commission in the meantime be at his Brooklyn home about May 1.

ENGINEERING SOCIETIES

The sixty-third meeting of the American Society of Mechanical Engineers will be held in Pittsburg, Pa., from May 30 to June 2, inclusive. The society has not met in that city since 1884. An executive committee consisting of E. M. Herr, chairman, George Mesta, J. M. Tate, Jr., Chester B. Albree, D. F. Crawford, Morris Knowles, and Elmer K. Hiles, secretary, will have charge of the Pittsburg meetings. It is expected that from 300 to 400 members and ladies will be in attendance.

It is proposed to celebrate the fiftieth anniversary of the Massachusetts Institute of Technology on April 10 and 11. At the same time there will be held a Congress of Technology, during which a large number of technology graduates will present papers dealing with various phases of the country's industrial problems as they exist today and promise to shape in the future. The papers, separately, will discuss the conditions and prospects in specific industries, and will therefore be of exceptional interest to the great number of men practically engaged in them. No similar discussion of the industries has been attempted on such a scale, and the record promises to be of unique suggestive value to the manufacturers of the country. The meetings will be open to the public.

The National Gas and Gasoline Engine Trades Association hold their convention June 20 to 23, inclusive, at Detroit, Mich., Albert Stritmatter, secretary, Cincinnati, Ohio.

The ninth annual meeting of the Western Branch of the Canadian Mining Institute was held in the Provincial Court House, Nanaimo, on February 17. Mr. W. F. Robertson, provincial mineralogist, of Victoria, B. C., was made chairman, and E. Jacobs, Victoria, B. C., secretary. E. B. McKay addressed the meeting, giving some interesting reminiscences of early days in Nanaimo. He stated that he first came to the district in 1875, bringing with him the first diamond boring machine, in order to prove the coal below what is known as the pitch in the old Douglass, and it was found at a depth of 240 feet. The paper by Charles Graham, superintendent of the Middlesboro colliery in Nicola, on "First Aid and Its Relations to Coal Mining," was read by Secretary Jacobs. C. F. J. Galloway, of Vancouver, read a paper on "The Beginning of Coal Mining in British Columbia." The first coal mined in British Columbia was in Rupert, in 1835. These workings were abandoned, however, for the Nanaimo workings in 1850. Mr. F. Napier Dennison, F. R. N. S., of the meteorological office, Victoria, delivered an address entitled, "Earthquakes, Strains, and Stresses in Relation to Coal-Mine Explosions." Mr. Jacobs read a report relative to the death of Fred Alderson, the Hosmer miner, who lost his life in attempting to rescue another. A vote of thanks was tendered the director of the United States Geological Survey for sending Mr. E. W. Parker to attend the meeting of the institute. It is proposed to hold the next branch council meeting at Trail, B. C., in May of 1912.

SELLING ZINC ORE ON CONTRACT

Written for Mines and Minerals, by Lucius L. Wittich

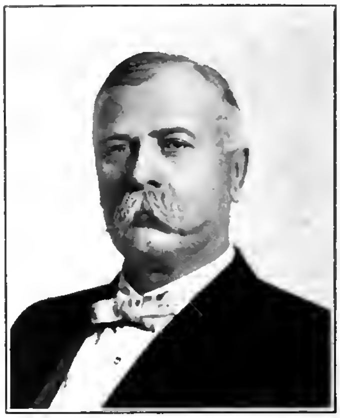
Method by which Prices for Zinc Ore are Based upon Price of Spelter and Analyses of Ores

Of almost 20 smelting companies buying zinc ore in the Missouri–Kansas–Oklahoma district, two, the American Zinc, Lead, and Smelting Co., and the Grasselli Chemical Co., are making purchases on a contract system, based on the average price of spelter at East St. Louis; this innovation having been introduced through the efforts of the Zinc Ore Producers' Association, an organization of mine operators of which C. E. Matthews, of Webb City, Mo., is president.

It is the first time in the history of this district, which has been a producer of zinc ores for almost 50 years, that a heavy tonnage of ore has been sold at a price determined to the fraction of a cent by the price of spelter. Of approximately 6,000 tons of zinc ore produced weekly in this district, almost 1,000 tons are now being disposed of on the contract basis, the sheet-ground producers of the North Webb City, Mo., field being conspicuous as the first to sign contracts on the new system. Fig. 1 shows a group of Webb City sheet-ground producers, the daylight workings of the Bradford–Kansas City Mining Co. being noted at the right of the picture.

Whether the contract system will result in the ore producer receiving a greater or less figure for his output than if he sold in the open market to the highest bidder is a question that is resulting in a storm of discussion, the majority of high-grade ore producers maintaining that the open market has its advantages, while the prevalent sentiment of low-grade ore producers is that the contract system will give the operator the best of the market. Comparative figures of prices paid, in the open market, and by contract, since the new system has been introduced, would indicate that both methods have their merits; but it is the contention of the Producers' Association, almost to a man, that cooperation of all producers will, in the end, result in better conditions and will create more harmonious relationship between smelter and ore producer.

After months of preliminary negotiations between the Zinc Ore Producers' Association and the various zinc-smelting companies buying ore in this district, which produces more than 50 per cent. of the zinc ore of the United States, which is converted into primary spelter, the first sales of ore on a contract basis were made in the week ending January 14, this year, and were based on the average price of spelter, East St. Louis, for the previous week. This spelter average was $5.37½. The contract specifies that zinc concentrates (except sludges) carrying 60 units of metallic zinc, shall bring $37 per ton (2,000 pounds, dry weight), when the previous week's spelter average shall have been $5 per 100 pounds, East St. Louis. The unit variation for metallic contents above or below 60 per cent. is to be $1 for each unit. That is, zinc blende containing 64 per cent. metallic zinc would command a premium of $4 per ton.



C. E. Matthews, President of Zinc Producers' Association

The price variation is to be 8.5 cents on or from the ore price per ton when spelter shall go up or down 1 cent per 100 pounds. The contract basis price for the first week of the experimental selling therefore was $37 per ton plus $3.19 (37.5 cents multiplied by 8.5 cents) which made the basis $40.19. The open market price for that week ranged from $40 to $44 a ton, basis of 60 per cent., which would appear, at first glance to indicate that those selling in open market received a better figure. However, several circumstances combined at that time to boost the open market bids, which, under other conditions might have been absent. For instance, the demand for ore was unusually strong at a period when spelter was weakened. At

Fig. 1 Sheet Ground Zinc Mines, Webb City, Mo.

another time, excluding the possibility of a manipulated spelter market of course, the metal would be high at a time when ore might not necessarily be in strong demand. Results under such conditions might give the producer selling on contract a big margin of gain.

Another feature of the contract welcomed by the producer of ores carrying 2 or more per cent. of iron is the clause which reads:

"Iron allowance: 2 per cent., no penalty; above 2 per cent., to and including 5 per cent., $1 per ton per unit penalty; above 5 per cent., 50 cents per unit penalty."

Under the present system of open market selling, only 1 per cent. iron is allowed without penalizing; for each additional per cent. $1 is deducted. So this clause in reality adds $1 a ton to the contract price; hence the bulk of ore that sold for $40.19 basis was of a class that would have been penalized at least $1, or more, in the open market.

Still another clause that appeals to the ore producer is this: "The producer shall have the right to withhold from delivery any ores and concentrates covered by this agreement, at any time during the continuance hereof when the base price under this agreement is below the sum of $40 per ton, but the producer shall notify, in writing, and not later than Tuesday of the week of production of which he desires to withhold, the agent of the smelter of his intention to so withhold such production; and provided further, that such production shall not be withheld from delivery for a longer period than 4 successive weeks."

Another clause specifies that the operator shall not be compelled to operate his mines steadily and that he may discontinue operations temporarily or permanently without affecting the terms of the contract.

In the week ending January 14, the first epoch of contract selling, the average price of all zinc blende sold was $41.49 per ton, which was higher than the contract basis price. The next week, ending January 21, saw the contract price jump to $40.49 a ton, reckoned on a spelter average of $5.41 for the previous week. The average price, all lots, was $41.06, a decrease, while the contract price had ascended. The basis price in the open market also was weaker, ranging from $38 to $43. For the week ending January 28, the contract price per ton was $40.23 on a spelter average of $5.38, but in this week the open market offerings slumped heavily, the basis ranging from $36 to $42.50 per ton, while the average price of all zinc blende sold was $38.78 a ton, considerably less, and for the first time, than the contract basis.

Producers of exceptionally choice zinc blende find a ready market for their output, the Matthiessen & Hegeler Co., operating five furnaces and more than 4,000 retorts, at LaSalle, Ill., and the Illinois Zinc Co., operating seven furnaces and almost 5,000 retorts at Peru, Ill., being willing to raise prevailing market quotations by several dollars per ton. The high basis prices paid by these companies in the open market are not criterions of the basis price paid by other companies, and because of this condition, the prevailing basis of all ores sold will be considerably lower than the top prices reported week by week. And because the producers of this grade of ore find such a ready market at prices in excess of that paid for second quality ore, they are opposed to the contract system, and justifiedly.

But to the mass of small producers, and to those whose ores are not in great demand in the open market, the contract system apparently offers relief, gives assurance of constant demand, and places a value on the producer's ore which will enable the producer to find bankers more willing to advance money on the output than they have been in the past. The contract system abolishes the competitive bidding, relieves the producer of the anxiety concerning his possible ability to sell his ore at a profitable figure, and gives him the privilege to withhold from the market his ore for a period of 4 weeks.

The contract system is not the result of months of mere discussion; it is the fruit of a carefully laid campaign during which the Producers' Association has sent agents to the plants of virtually every zinc-smelting company in the United States. Data on spelter production have been secured; every detail of the metal industry as well as the mining department has been given careful study, and upon the information so secured the scale was determined. Comparisons of what contract prices "might have been," had the contract system prevailed in years gone by, show a margin in favor of the open market system of selling; but this fact must not be forgotten: Had the contract been in force in the past, had the zinc ore producers of this district enjoyed control of the situation, would it not be reasonable to presume that the average price of spelter would have been sufficiently higher to have shown a margin in favor of the contract prices?

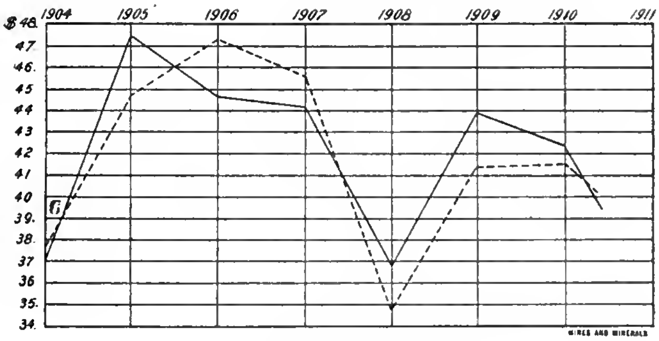


Fig. 2. Zinc Ore Prices

Relative to the smelter's margin, the Zinc Ore Producers' Association gives out the statement that $14.80 on 5-cent metal is reasonable and just; although this figure appears rather high to a great many zinc-ore producers of this district who have not forgotten the recent battle at Washington for a protective tariff on zinc ore, and on that occasion the supposition gained currency that the smelter asked for only an $8 margin.

Had the producers of the Joplin district been selling their ore on contract for the past 7 years, and had the same smelter prices prevailed that did prevail, the district would have received an average of $41.87 a ton for all ore sold instead of $42.42, the actual average basis, and this would have meant a loss to the district producers of hundreds of thousands of dollars. Fig. 2 shows a diagrammatic scale of actual and hypothetical prices, the solid line indicating actual basis offerings, while the dotted line shows the hypothetical basis, figured on the contract system, giving an idea of how the producers, at the existing spelter prices, would have profited in certain years by the contract system and would have lost in other years. The average for the whole period, however, is a strong argument in favor of the existing method of ore buying.

Table 1 shows the average price of spelter, East St. Louis, and actual basis price of 60-per-cent. zinc ores from and including 1904 to 1910; the value of metal in a ton, calculated on a recovery of 85 per cent. from 60-per-cent. zinc ores; the actual working margin of the smelters, and the calculated ore prices and smelter's margin had the contract been in force,

Table 1

Year	Average Price Spelter, St. Louis	Value 1,020 Pounds Spelter	(Old Scale) Ore Price 60 Per Cent. Zinc	(NewScale) Ore Price 60 Per Cent. Zinc	(Old Scale) Smelters Working Margin	(NewScale) Smelters Working Margin
1904....	$4.97	$50.69	$37.40	$37.75	$13.29	$12.94
1905....	5.79	59.06	47.40	44.72	11.66	14.34
1906....	6.10	62.22	44.82	47.35	17.40	14.87
1907....	5.90	60.18	44.36	45.65	15.82	14.53
1908....	4.62	47.12	36.63	34.77	10.48	12.35
1909....	5.39	54.98	43.98	41.32	11.00	13.66
1910....	5.42	55.07	42.35	41.57	12.72	13.50
Average 7 years		55.61	42.42	41.87	13.19	13.89

The proportions between spelter and ore base price under the old system figures out 7.76 to 1, while under the new system or scale the proportion is 7.65 to 1. That is, when spelter is quoted at a certain figure per 100 pounds, the price of ore, per ton, has been 7.76 times as great. While the contract price is slightly lower than the other, the contract scale is more than compensated by the difference in penalties exacted. The allowance of another per cent. of iron without penalty, the lowering of the penalty to 50 cents per unit after the fifth unit, the premium of $1 for less than ½ of 1 per cent. of lead, make ample provision for the difference in ratio, even had spelter prices been the same. The association maintains that the producers, through curtailment, would have been enabled to have "pegged" the spelter market at a much higher point than the low quotations of the past 7 years. The association maintains, for instance, that in 1904, when spelter averaged $4.97, the average could have been held to $5.25 or better. That a curtailment of zinc ore from this district will have a tendency to stop the slump in spelter has been evidenced this year, the spelter market having been held above $5.25 when there was every indication that it was scheduled to hit the $5 mark before the end of January. The metal now is ascending, and so long as this condition exists the producers selling on contract will enjoy stronger prices for their ore, week by week.

In compiling the new scale, the association and the smelters start at a point where it is claimed both the miner and the smelter are just breaking even. When spelter goes below $5 it is claimed neither smelter nor ore producer will make a profit, except in the case of exceptionally rich mines; but, as the price of spelter ascends, the miner's revenue not only increases proportionately, at the ratio of 8.5 cents for each 1 cent increase in the price of spelter, but the smelter's margin likewise widens. Table 2, shows the calculated price for zinc ore carrying 2 per cent. or under in iron and less than ½ of 1 per cent. lead on a spelter basis varying from $5 to $6.50; also shows working margin to smelter with 88 per cent. metal recovery.

Table 2

Spelter Price Per 100 Pounds	Metal Return and 88 Per Cent. Recovery in Pounds	Spelter Value	Ore Price Under Scale	Working Margin to Smelter
$5.00	1,056	$52.80	$38.00	$14.80
5.50	1,056	58.08	42.25	15.83
6.00	1,056	63.36	46.50	16.86
6.50	1,056	68.64	50.75	17.89

"Under this scale would it not seem reasonable," declares the Producers Association, "that the smelter would rather pay $50.75 for his ore and reap a bigger profit, his margin being $17.89, than to pay $38 for his ore and have a margin of only $14.80?"

From this it would appear that both ore producer and smelter would be benefited; but the objection is raised that $14.80 is a princely margin for the smelter, that in 7 years under the old system the margin was $13.19, and that if the smelter could prosper on this latter margin, and construct new plants in all portions of the country with the view of producing more spelter, why could he not profit on a $14.80 margin, keeping the price of spelter at a figure where the demand must necessarily be heavier than would be the case if the metal price were higher. Just as a merchant would sell more eggs at 10 cents a dozen than at 20 cents a dozen, so would the smelter sell more metal at $5 per 100 pounds than at $6.50—enough more, in fact, some contend, to more than overbalance the added price per 100 pounds. And again, a very pertinent question is propounded by those who are prone to question the practicability of the new scale: "Is there not sufficient scope for manipulation in the spelter market to permit the metal producer to buy at one figure (the reported price week by week) and sell at another price, in advance of the quoted one. That manipulation of the spelter market is not impossible is the common supposition in this district; and this presumption is strengthened by the reported efforts of certain operators to purchase metal at the actual East St. Louis quotations. For example, spelter at East St. Louis, at the close of January was reported to be ranging between $5.25 and $5.30, but to purchase any considerable amount at this figure is said to have been an impossibility. The private contracts of numerous concerns, especially those producing sheet zinc, hinge to a small extent on spelter quotations. It is true that in this district there is much guesswork relative to the secret workings—if such secret workings actually exist—of the powers higher up in the spelter industry; but be this as it may the spelter market quotations upon which the scale will be based each week are of vital interest. The contracts call for settlement upon the average of the preceding week's metal prices, East St. Louis. That the quotations received will be the actual quotations there should be little question—for no insinuation is made that alterations will be made in giving the actual figures. Considering that a heavy tonnage of metal is sold on private contracts at higher figures than the quoted metal prices, as has been explained, it is clearly seen how the ore producer will be selling on a lower quotation than the one upon which the actual sales of spelter are made. The ore producer thus loses and, in nine cases out of ten, would not know when he lost. In this connection it is well to explain that the spelter made from Joplin ores is for the most part more choice than that from other fields, and a larger portion of it is sold as special brands and at a premium over ordinary prime western spelter. For instance, three smelters handling Joplin ores make a "brass special" for which they receive a premium over the prevailing price. Others make a spelter for sheet zinc, the Joplin ores being especially desirable for this purpose. That the producers of the choicer grades of ore shall have incentive for cooperating in the contract system movement, the association is negotiating with the smelters of high-grade ores with the view of securing a more liberal contract which will appeal to others besides the producers of second-grade zinc ores.

SOLUBILITY OF THE RARE GASES IN WATER

The solubility of xenon, krypton, argon, neon, and helium in water was determined at temperatures from 0° to 50° C. The solubility rises with the increase of the atomic weight. At 20° C. the solubilities are He., .0138; Ne., .0147; Ar., .0379; Kr., .0670 and .0788 (the reason for this difference in values is not known), Xe., .1109. The solubility of xenon is greater than that of any other gas, which does not form a compound with the solvent. In the case of all the rare gases, there is a distinct minimum of solubility. The minimum for xenon and argon lies at 40° C.; for krypton between 30° and 40° C.; for helium at 10° C.; and for neon probably at 0° C. A. V. Autropoff. (Proc. Roy. Soc., A. 38, 744.)

SILICON STEEL

This article will well repay careful perusal, as it gives a careful resume of tests to show its superiority over other kinds. Among other things it is stated that high silicon steel was largely used in the construction of the Mauretania and Lusitania, the requirements being, ultimate tensile strength 74,000 to 85,000 pounds per square inch; elongation in 8 inches, 18 per cent. The United States naval requirements are: Minimum tensile strength and elongations of 60,000 pounds and 25 per cent.; and 75,000 pounds and 17 per cent. for medium steel. Manufacturers have found that the requirements for high tensile steel can be met much more readily by high silicon than by high carbon steel. G. A. Bisset, Naval Constructor, U. S. N. (Iron Age, lxxxvi, 8.)

MINING IN THE TINTIC DISTRICT OF UTAH

Written for Mines and Minerals, by Leroy A. Palmer

The Tintic district is one of the oldest mining regions of Utah, and has been one of its most productive. Notwithstanding this fact, some of its greatest mines are scarcely more than names to many well informed on mining subjects.

The Eureka Zone and Some of the Principal Mines—Conditions and Methods of Operating

The district is south of Salt Lake City, in the central portion of the Tintic Mountains, the easternmost of the narrow ranges rising out of the basin of the old lake. This range is about 40 miles long and is practically continued in the Oquirrh Range on the north and the Cañon Range on the south. These mountains are not very high, Tintic Mountain, the highest peak, rising to an altitude of 8,214 feet. The region is very dry, there being but one perennial stream. Such water as is used is obtained from springs and wells and by piping from Cherry Creek, 18 miles distant in Tintic Valley. Naturally, vegetation is sparse, consisting mostly of sage and stunted and gnarled pines. In this district, in a portion about 12 miles long and half as wide, are found the mines that have yielded the richest ores in Utah, and its most stubbornly contested lawsuits.

The first discovery was made by a party of prospectors returning from Western Utah, who located the Sunbeam claim on December 13, 1869, thus making Tintic the fourth mining district in the state, Bingham, Stockton, and Alta having been located at that time. The district was handicapped for years by the fact that it had no railroad, but the richer surface ores were hauled to Salt Lake, and from there shipped to Reno, San Francisco, Baltimore, and even Swansea, Wales. The average was too low for such expensive shipment and many attempts were made from time to time to mill or smelt in the district, and two mills and two smelters were erected in 1871. From that time up to 1895 several mills were erected, but owing to the refractory nature of the ore, amalgamation met with little success and two attempts at leaching were even less successful. These failures were due in part to a scarcity of water, but an attempt to operate a dry concentrator met with only indifferent success. From 1871 to 1908 seven smelters were built in the district but none of them survived more than a short run. The last was a modern plant of 1,200 tons capacity adapted to both lead and copper ores, which closed down just a year from the time it started. This was due largely to the fact that the district supplies ore that is mainly silicious, and favorable freight rates on outside ores which would have made suitable mixtures were not forthcoming. So it may be said that the Tintic district has been a very successful one from the miner's standpoint, but a dismal failure from the point of the metallurgist.

The Union Pacific built into the district from the west in 1883 and this aided materially in its development. In 1891 the Rio Grande Western built in from the east so that for the past 20 years the mines have had two outlets to the outside world.

Fig. 1. Part of Eureka-Tintic District
a, Bullion-Beck Shaft; *b*, Bullion-Beck and Champion Shaft; *c*, Gemini Shaft; *d*, Eureka Hill Shaft

The ores of the district occur in both the sedimentary and the igneous rocks; and for the first few years production was about equal from each. As the oxidized ores were exhausted in the igneous rocks a number of the mines were abandoned. Later shafts were sunk that encountered good sulphides, and mining in this formation was resumed after an interval of practically 20 years. The mines in the sedimentary formations have had a steady and increasing production and some of them are developed for years ahead.

The Tintic Range is a composite range of different rocks in which the sedimentaries are partly buried by volcanic outpourings and contain intrusions of igneous rocks, all being more or less concealed by surface accumulations, which condition has tended to hamper prospecting.

The sedimentary rocks, which are of the Paleozoic era, consist of limestones and quartzites of a total thickness of 14,000 feet. They are divided into the Tintic quartzite, 7,000 feet so far as exposed, a pure, compact, fine-grained rock of the Cambrian period with occasional beds of fine quartz pebbles; the Mammoth limestone, 4,000 feet, of dolomitic, cherty and shaly lime of the lower Carboniferous age; the Godiva limestone, 3,000 feet of upper Carboniferous blue and gray limestone with sandy beds in the lower portion and carbonaceous beds in the upper; and the Humbug formation, which consists of 250 feet of fossiliferous limestone alternating with arenaceous limestone and calcareous sandstone overlying and conformable with the Godiva limestone. The section presenting these strata is the western limb of a syncline which pitches toward the north. This limb is characterized by steep dips with beds vertical or even overturned, while on the eastern limb the dips rarely exceed 35 degrees.

There are a number of faults, the most of them with throw of only a few feet, but others with throw up to 400 feet, and cutting the limestones near the town of Mammoth is the Spy-Ajax fault with a throw of 1,000 feet. There may have been some displacement along the many fissures in the lime, but it is hard to detect, and, in any case the faulting in this district seems to have been of less importance than the folding.

The volcanic activity probably took place in the Neocene period of the Tertiary era. The igneous rocks, which are both intrusive and extrusive, are, in the order of their eruption, rhyolite, monzonite, and andesite (same age), and basalt. These rocks cover the greater part of the district, andesite being the most characteristic rock of the Tintic Mountains, about 100 square miles being overlaid by it and its tuffs and breccias. The rhyolite is the next most widely distributed, while the basalt and the monzonite occur only in limited areas, chiefly in sheets and dikes, and both of them of a composition closely resembling trachyte.

The rocks give evidence of alteration, both by physical and chemical agencies. In some belts the massive limestones have been reduced to a shale-like rock, which has in turn decomposed to a plastic clay, by sheeting and crushing. Hydrometamorphism has played an even more important part than dynamic action, and superficial alteration has caused extensive

surface deposits, as well as affected the character of the ores above the ground water level. The igneous rocks were not erupted until after the period of folding, so do not show much metamorphism.

There are two classes of fracture systems, those in the sedimentaries and those in the igneous rocks. The fracturing and mineralization of the sedimentaries took place before the period of volcanic activity and consequently before the fracturing and mineralization of the volcanic rocks. The fractures are very numerous and are easily traceable in the quartzites and hard limestones. They are also most numerous in the vicinity of the mineral-bearing zones of which there are three in the district, the Eureka, Mammoth, and Godiva-Sioux. Most of the fractures occur in the northeast-southwest quadrant and the most of these are almost due north and south. The fracture planes are nearly vertical, the dip in general being not less than 70 degrees. Of the east-west fractures the Spy-Ajax fault is the most prominent. Most of the fractures cross but do not fault one another. The faulting fractures are later than mineralization, but, as they do not occur in the igneous rock, were previous to volcanic action. The general direction of the ore bodies is along the north-south fractures.

The ore occurs along the fracture zones in both the sedimentary and the igneous rocks. In the sedimentaries it is confined to the three zones mentioned, and in the igneous rocks to short veins widely separated. The more common minerals are pyrite, sphalerite, galena, enargite, and their oxidized forms carrying gold and silver in a quartz or barite gangue. In the

Fig. 2 Centennial-Eureka Shaft Buildings

igneous rocks the ground water level was found at a depth of 200 to 700 feet, the line being very marked by the complete oxidation of the ore above and the unaltered condition below. In the sedimentary rocks the oxidized zone extends to great depths, in fact only two mines have entered it, the Gemini at 1,670 feet, and the Centennial-Eureka at 2,250 feet. In the igneous rocks the ore occurs only in the walls of fissures; in the sedimentary rocks it fills irregular spaces or chambers along the fissure zones or replaces the country rock. At the northern end of the zones lead and silver ores predominate and at the southern end gold and copper.

A feature of this district is the number of ore bodies that occur filling caves in the limestone. The supposition is that these were originally replacement deposits or deposits in cavities caused by dynamic action. They underwent leaching, which left the ore body in a more or less porous condition and caused contraction. Then occurred some disturbance which caused the loose mass to settle leaving a cave with ore on the bottom and clinging to the sides and roof.

The deposition of the ore was apparently made by ascending heated solutions which contained hydrogen and alkaline and arsenical sulphides, with some forms of silica, barium, lead, copper, silver, and gold. The replacements in the lime were due to chemical reactions and to the checking and cooling of the solution by some already existing cavity, or to contact with surface water containing oxygen or carbonic acid or organic matter. The separation of the minerals and the original complex ore bodies into those containing only one was due to a difference in the stability and solubility of the original minerals.

Taking into consideration the actively operated mines, the Eureka zone may be considered as extending from the Ridge and Valley on the north to the Apex in Mammoth Basin on the south, thus taking in the Gemini, Eureka Hill, Bullion Beck, and Champion, Centennial-Eureka, King William, and others of less importance. The greatest fracturing in this zone is between the Eureka Hill, Bullion-Beck, and Champion shafts. The ore bodies follow the fracture planes, which have a general north-south trend in the southern end, but turn toward the west in the southern end of the Eureka Hill, and between the two shafts mentioned have a course N 35° W from which they turn north of the Bullion Beck and Champion to almost due north. Toward the southern end of the zone the ore bodies are continuous and extend to great depth, but toward the northern end they are not so continuous and are in lens-shaped masses scattered through the zone. The Centennial-Eureka is the largest shipper in the Eureka zone, and in fact in the Tintic district, so will be taken as an example of operation in this portion.

The Centennial-Eureka.—This mine lies southwest of the town of Eureka toward the southern end of the Eureka zone, being the farthest south of the important producers of this belt. The ore is mostly copper with good gold values, the copper occurring as a carbonate or oxide with a very little native, although some small bodies of argentiferous galena are found. The shaft has recently reached the water level and it is supposed that below this point the values will be found as sulphides. The occurrence of the ore follows the general rules mentioned in the notes on geology of the district and is usually found near the intersection of two fissures. The ore bodies have a dip nearly vertical and vary in width up to 18 to 20 sets. The occurrence is more regular than in some of the other mines and the grade of the ore is such that, in general, all is extracted and the stopes do not present the irregular outlines found in some of the mines in the northern end of the zone.

For years the main working entry has been the Centennial-Eureka shaft, a two-compartment and manway vertical shaft that has been carried to a depth of 2,250 feet, but the Holden tunnel, recently completed, which connects with the shaft at a depth of 535 feet, is now used for the transportation of supplies and ore. The shaft is carried down in firm limestone, the two compartments and manway all having a measurement of 4 ft. × 4 ft. in the clear. Timbering is done with 8″×8″ sawed timbers, with 2-inch plank lagging. The tunnel will effect a considerable saving in the handling of ore and supplies, besides being much more convenient for the men. Reaching the shaft at the 535-foot level, it cuts off just so much from the height to which ore and water must be hoisted and permits of delivery of the ore directly to the shipping bins from the mine cars, whereas, in the past this delivery has been by aerial tram. The saving in the delivery of supplies is proportionate, timber, for example, being framed in a shed at the tunnel mouth and sent directly to place in the mine instead of being hauled by a winding wagon road to the shaft house.

There seems to have been no regular system in spacing the upper levels of the mine, which were driven some years ago. The first is 160 feet below the surface and from there to the 1,400-foot level the average distance is 112 feet. At this depth the shaft is a considerable distance from the ore body, so, in order to avoid a great amount of dead work, levels have been driven from the shaft at intervals of 225 feet, with blind working levels between. The levels so driven are designated as the 16, 18, and 20, the latter having a depth of 2,250 feet below the surface. Below this level a sump has been sunk 65 feet.

The drifts, which run through the lime formation, are 5 ft. × 7 ft. with an 18-inch track well laid and maintained. They require little timbering, a few sets, or posts and head-boards being all that is required. Ingersoll-Rand piston drills are used for drifting and most of the stoping is done by Waugh stopers.

Stoping is done by the underhand method. A slice is carried the width of the vein on the drift level and timbered with square sets. The stope is then carried upward and supported by square sets of 8″×8″ timbers. As a rule the lime walls have a steep dip and are firm, so that no lagging is required, but, as will be seen later, some very heavy ground which calls for special timbering is encountered. Most of the stopes are large, allowing the men plenty of room in which to work, so that the ore breaks well and one man can bring down 15 to 18 tons per shift. The broken ore is wheeled to chutes to the next through level on which it is hauled by mules in 20-foot cars to the shaft. The chutes are 14 in. × 14 in., of steel, and have given very satisfactory results, the increased cost being more than returned by the freedom from repairs and the longer life than timber chutes.

Timbering is an interesting feature in this mine and has been conducted with great care so as to preserve the ground and afford absolute safety to the workmen. In ordinary ground square sets are used. These are 8″×8″ sawed timber, spaced 5 feet in the clear with posts cut with a 1-inch tenon and caps butting. Much of the ground is heavy and some special forms of timbering have been used. If the pressure is not too great an ordinary doubled-up set is used, a set of eight by eights being placed inside the regular set. If there is a tendency to heavy side pressure, angle braces are carried from one cap to another so that the pressure is transferred in part to the posts. On the hanging wall every alternate set is carried clear to the wall, at which point the caps are connected to a long stull which covers the two sets and rests against the wall. A strut is carried from the intermediate cap to this stull and perpendicular to it so that the entire pressure of the hanging wall is transferred through the caps to the foot-wall. The foot-wall timbering consists simply of short sets tightly wedged to the wall with such braces as may be necessary. All sets are floored with 2-inch plank and these floors are kept tight while the stope is being worked, the manways being the only open part. As a stope is worked out it is filled with waste that is taken out in development work. If a stope is of such size that the timber shows signs of working before all of the ore is extracted, that portion that is worked out is bulkheaded off and filled while the remainder is being worked. Only such waste as is taken out during development is used for the filling and none is mined especially for the purpose. A sufficient amount is obtained in this way to fill most of the larger stopes. In some portions of the mine, notably on the 1,000-foot level, extreme pressure was encountered, as shown by the crushing of doubled-up sets that had been reinforced by angle braces. Here resort was had to cribbing to keep the stopes open while the ore was being extracted, after which they were filled as quickly as possible. The cribbing is made of eight by eights laid crossways with 8-inch spaces between and the whole filled with waste and waste filling between the sets. These cribs are built as close to the walls as possible and heavily braced to them. In times past some effort has been made to recover the timbers from the stopes, but without success, and the practice has been given up.

From the foregoing it will be seen that the ore is easily mined, although the timbering necessary makes its cost high. While there are not many places where cribs are necessary, it is probable that mining cost, including development and general expense, is in the neighborhood of $3.50 per ton.

The surface equipment is quite complete and is well maintained. The shaft house is of corrugated iron over a steel frame. In the boiler room are four 150-horsepower Stirling water-tube boilers generating steam at 100 pounds. They are equipped with American stokers, belt-driven by a small vertical engine, and draft is supplied by a 4-foot diameter centrifugal blower driven by a Sturtevant piston-valve engine.

The hoisting engine is a 20 in. × 60 in. Fraser & Chalmers duplex first motion, with 12-foot reels which can be operated independently or clutched together and run in balance. The reels wind a ½″×5″ flat rope. The head-frame is 60 feet high, consisting of two 12″×24″ timbers bolted together, with a batter of 1 inch to the foot. These uprights are braced on each side with an 18″×24″ timber tied to each upright by a 12″×24″ strut, thus making a staunch and rigid structure. The sheaves are wrought iron, 8 feet in diameter. The cages are double-decked, attached to the ropes by four clamps, each with four bolts, and are equipped with safety catches which operate automatically if the rope breaks. The compressor is a two-stage Sullivan 13 in. × 26 in. ×44 in. ×20 in. ×32 in. with a capacity of 1,500 cubic feet per minute. The machine and timber shops are under the same roof as the shaft. They are operated by a 12″×24″ Corliss engine.

The use of flat ropes is a general practice in the deep shafts of Utah. The objections that have been made to them are their great first cost and the shorter life caused by the wear of the rope winding on itself and rubbing on the edges. This latter feature can be eliminated to a large extent by carefully lining the reel and the head-sheave, and, if the lead from the shaft to the engine is a long one, providing guides to keep the rope from swaying. The Tintic operators profess themselves well satisfied with the flat ropes, and they are also in general use at the largest Park City mines. One of the latter claims an average life for a ½″×5″ rope of 5 years. The advantages are the saving in space and the fact that when the hoist is begun and the inertia of the load must be overcome, the drum winds on a small diameter, thus giving the engine a large leverage, and as the hoist is being made and the cage gathers momentum, this diameter is increased, allowing an increase of speed. For such a shaft as the Centennial-Eureka each drum would have to be 6½ ft. × 15 ft. to accommodate a round rope of the proper diameter and length. The reels therefore effect a large saving in space as well as allow the use of a much shorter and consequently smaller shaft.

Fig. 3. Timber Connection to Hanging Wall in Centennial-Eureka Shaft

One of the characteristics of the Tintic district is the great depth attained before water is encountered. In the case of the Centennial-Eureka this occurred while extending a drift at a depth of 2,250 feet. The drift, which was being run to the south, encountered a water course, supposed to be the same as that in the Gemini, the two being only 2 feet different in elevation, which gave a flow of 300 gallons per minute.

The mine had no pumping equipment, so a concrete bulkhead was built across the drift and the flow was cut off except for such portion as seeped through the formation into the sump. This amount was only such as could be handled by bailing skips. Bailing was carried on between the regular shifts, a skip being hung in each compartment in place of the cage. These skips are cylindrical with shoes working in the cage

guides. Each is 3 ft. × 10 ft. of ⅛-inch iron, with a large butterfly valve forming the bottom. Below the bottom is a pointed hood so that the skip enters the water easily, this hood being open on two sides so that the valve opens and the skip fills. Each skip has a capacity of 500 gallons and they gave good results in handling the water in the sump while the pumps were being installed.

The shaft is equipped with a No. 9 Cameron sinking pump which raises the water to a concrete sump 6 ft. × 9 ft. × 24 ft., with a capacity of 10,000 gallons, on the 2,250-foot level. Here a large station has been cut out and two 13-stage centrifugal pumps installed. This is, so far as the writer's information goes, the first installation of centrifugal pumps for station work against a high lift in this country. This type of pump was used by the United States Mining Co. in Mexico with such success that it was recommended for this installation. The station has provision for four pumps, two of which are installed. Each has a capacity of 500 gallons per minute. They are built in two sections, one of five and one of eight stages, with a 400-horsepower 4,000-volt General Electric induction motor mounted on the shaft between the two. The five-stage section receives the water through a 10-inch suction pipe, the intake being about 5 feet above the surface of the water when the sump is full. This discharges to the eight-stage section which in turn discharges to the 10-inch water column terminating at the Holden tunnel 1,715 feet above.

Fig. 4. Cribbing in Centennial-Eureka Stope

The concrete bulkhead was shot out when the pumping equipment was in running order and the drift extended. A cross-cut was driven and from this a drift to the sump. These drifts have 4′×6′ cast-iron water doors with dressed edges close-fitting against a cast-iron frame. They swing inward against the flow of water and provide protection against a sudden increase of water flow such as may be expected in a lime formation.

The Holden tunnel starts near the bottom of the gulch below the town of Eureka, and terminates at the 535-foot point in the shaft. It is 2,207 feet long, 9′×8′ in cross-section, with a grade of .40 per cent. It was driven through the hard lime all of the way, so that after the surface formation is passed only 12 sets of timbers are required in the entire length. It is laid with a 36-inch track of 35-pound steel for electric haulage, this track running over a trestle to the ore bins which are beside a spur of the San Pedro Railroad. At the station at the shaft is a 700-ton ore bin with eight air-operated gates, and beneath the track is a coal bin from which the coal can be unloaded to cars and hoisted to the shaft house. All freight will be hauled in through the tunnel, and timbers will be framed at a shop at the mouth.

Power is generated in a brick building with concrete floors near the portal of the tunnel. Coal for the power plant is unloaded to bins beneath the railroad tracks, from which an inclined conveyer carries it to storage bins from which it is hand trammed to the stoker bins. The boiler room has three 225-horsepower Stirling water-tube boilers with American stokers. The feedwater is heated in a Hoppes heater which receives the exhaust of the feed-pumps, two Epping & Carpenter duplex-plunger 7½ in. × 4 in. × 5 in. × 10 in. × 10 in.

Electricity is generated by a 400-kilowatt, 4,000-volt, General Electric three-phase, 60-cycle alternating current generator mounted on the shaft of 19″×37″×24″ cross-compound Ball-Wood engine. There are two General Electric exciters, one 20-kilowatt, 125-volt, chain-driven by a 7″×7″ Sturtevant slide-valve engine and one 17½-kilowatt, 125-volt, direct-connected to a 25 horsepower, 400-volt, induction motor. The power for the haulage system is supplied by a 25-kilowatt, 250-volt, General Electric direct-current generator, direct-connected to a 35-horsepower, 440-volt induction motor. The engine is connected to a Worthington barometric jet condenser with a 10″×18″×12″ vacuum pump. The current is led direct to the mine pumps and to two 30-kilowatt General Electric transformers stepping down from 4,000 to 440-volts, which furnish current to the exciter and motor-generator sets. There are also some smaller transformers stepping down to 120 volts for the lighting systems. As an additional source of power, current can be taken from the Utah County Power Co.'s lines through three 250-kilowatt, 11,000- and 5,500-volt primary and 4,000-volt secondary transformers located in a small brick substation.

The Centennial-Eureka has been described in detail as the largest mine of the district and the one presenting the most interesting operating features. At the northern end of the district where the production is of silver-lead ore the Gemini is the most active. This mine has a two-compartment shaft to the 1,670-foot level, and a winze from there to the 1,970-foot level. The ore is extracted by square setting and the stopes have an irregular shape owing to the fact that the ore is not of uniform grade and only that which will ship is extracted. Water was encountered on the 1,670-foot level, and below this point copper ore running high in silver has been found. Pumping is done by a 12″ × 12″ Aldrich quintuplex pump on the 1,970-foot level and a similar pump on the 1,670-foot level. The surface equipment includes three 200-horsepower Stirling boilers, an 18″×60″ Fraser & Chalmers duplex first-motion hoist with ½″×5″ flat rope and an 18″×36″×20″ Noerdberg duplex compressor. North and east of the Gemini is the Ridge and Valley, which is worked from the 1,670-foot level of the Gemini. The formation is the same as that in the Gemini and, in general, the same method of operation is followed.

Across the cañon from the Gemini, and adjoining it on the south, is the Bullion Beck and Champion, now owned by the United States Mining Co. Butting on the Bullion Beck and Champion is the Eureka Hill, which joins the Centennial-Eureka on the south. The Bullion Beck and Champion is working only a few company men, and the Eureka Hill none at all. A number of leasers are working in both mines. Each mine has a depth of about 1,500 feet, with two-compartment shafts, and 20″×60″ duplex Fraser & Chalmers hoists. It was between these two companies that one of the most stubbornly contested legal suits of the district was fought.

The Eureka zone has produced very liberally, but just what the production has been it would be hard to state. Many of the mines are held by close corporations which have an aversion to giving out figures. Also in the early days of the camp some of the mines were worked as partnerships, and no one knew just what was taken out. The known dividends of the mines on the Eureka zone are as follows, but the actual profits doubtless exceed this considerably: Gemini, $2,000,000; Eureka Hill, $1,500,000; Bullion Beck and Champion, $2,738,400; Centennial-Eureka, $5,317,000.

The Ridge and Valley is supposed to have paid some profits which are not of record. Beyond the Centennial-Eureka are the Apex and King William, which are still in the prospect stage and the only mines on the zone that have not shown a profit.

REVERBERATORY COPPER SMELTING

Written for Mines and Minerals, by E. B. Wilson

Methods of Roasting Ores and Smelting in Reverberatory Furnaces

This first of a series of copper smelting articles is descriptive of roasting and modern reverberatory practice. Welsh copper smelting, while it may be described in a general way, involves 12 or 15 operations before the copper is ready for the market. The successful operation of the Welsh method requires an experience which cannot be described, and as it is not practiced in the United States and is going out of style in Swansea, the reader is referred to Dr. Edward D. Peters' "Metallurgy of Copper" for further particulars. In the Lake Superior copper district, where practically metallic copper is melted, skimmed, and refined, all in reverberatory furnaces, the smelting is comparatively simple, and has changed little since its introduction, except in mechanical features.

The ore, previous to smelting, is cleaned as thoroughly as is consistent with economy, in which condition it carries probably about 35 per cent. of rock impurities that cling to it and are difficult in some instances to remove. The furnaces are not made large, being capable of smelting 625 tons of ore and producing about 400 tons of copper per month, which is a remarkable contrast between the Washoe, Cananea, and Arizona smelting companies' reverberatories, which, running on sulphide ores, smelt to matte from 250- to 350-ton charges in 24 hours.

The hearth of the Lake Superior furnace is 12 ft. × 18 ft., that of the Cananea furnace 100 ft. × 19 ft.

The ore is charged into the furnace through a hopper in the top; the mass copper through the working doors in the side; and large mass too heavy or large to pass through the doors is charged by a derrick when the roof is repaired. The copper commences to melt soon after it is charged, and slag skimming commences as soon as it accumulates in sufficient quantity.

After skimming the melted mass on the hearth it is flapped to oxidize the impurities and send them into the slag. Where flapping was once carried on with paddle-shaped tools thrust into the bath and then raised, it is now generally done by

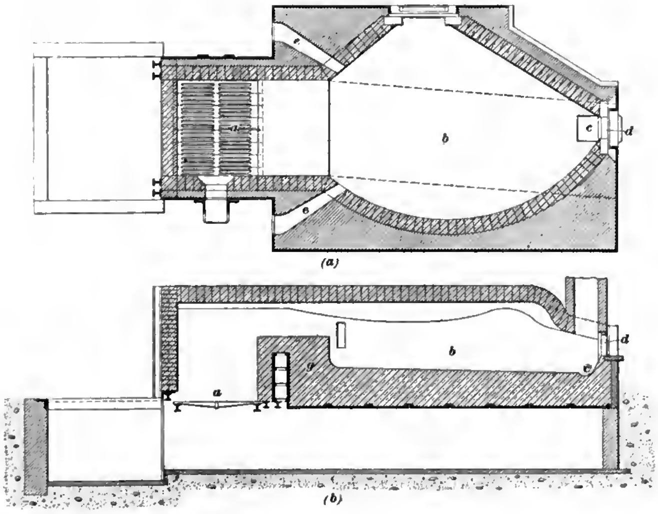


FIG. 1. LAKE SUPERIOR REVERBERATORY FURNACE FOR COPPER

thrusting pipes through which steam is forced under pressure into the bath. In flapping, the rabble or steam pipe should not be thrust deep into the bath, but only a little below the surface, where the lighter impurities are, and even then considerable copper is oxidized and sent into the slag. The slag is removed from time to time during the flapping process, which lasts several hours. Test samples are taken during the operation by means of small ladles and examined to ascertain the progress of the melt. When the impurities are thoroughly oxidized, the test sample on being cooled and broken shows a slight depression in the middle and the fracture has a mottled appearance. The next operation consists in spreading charcoal over the top and then thrusting a green sapling into the bath. This operation, termed "poling," has for its object the stirring of the metal and the reduction of any copper oxides remaining by bringing them in contact with the reducing action of carbon in the charcoal and in the sapling. If this operation

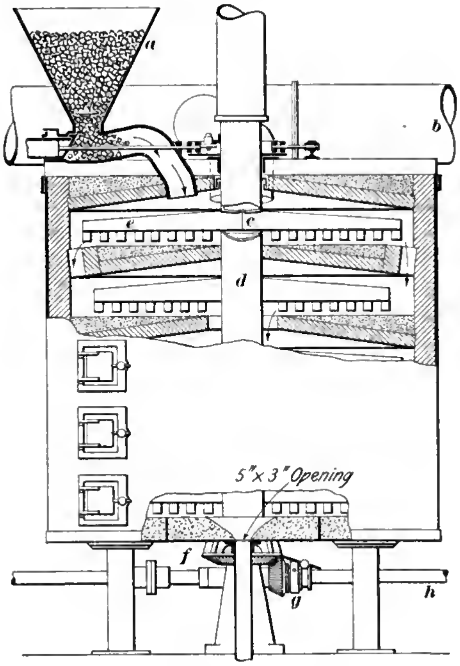


FIG. 2. HERRESHOFF FURNACE

is carried to just the right point a sample of the metal will not have a depressed surface, the fracture will show a fine texture, and the metal a reddish color. The metal is now cast into ingots and after pickling is ready for the market.

Slag from the first skimming of the reverberatories contains from 10 to 20 per cent. copper and is worked over in blast furnaces, while the rich slag from the final flapping and the poling, which contains from 20 to 35 per cent. copper is remelted in the reverberatories.

The furnace operations will be now about as follows:

1. Mineral, black copper from cupola, limestone flux and rich copper slag from *d*.

2. *a*, fusion; *b*, removal of slag; *c*, rabbling; *d*, poling; and *e*, casting and pickling.

3. Slag from operations *b* and *c* is sent to the blast furnace; slag from operation *d* is charged into the reverberatory.

The blast furnace mentioned is designed to recover the copper from the reverberatory slags and is more properly termed a cupola (about 36 in. × 76 in.), because of its size and intermittent working. It is charged with slag from *b* and *c*; two-fifths as much limestone as slag and about one-seventh as much coal as slag and limestone. The product of the cupola is "black copper" containing from 70 to 80 per cent. metal and slag. The black copper is charged into the reverberatories and the slag is wasted after standing some time in a settling pot.

When the rich copper ores of the West were at the surface and mining was carried on in carbonates and oxides, as at Clifton and Bisbee, Ariz., the blast furnaces produced what was termed black copper, a product carrying from 70 to 90 per cent. copper; with depth, however, the ores changed to sulphides, and the furnaces were changed to produce a complex sulphide product termed "matte." To free the sulphides from rock, crushing and wet concentration were introduced, and, because the concentrate was too high in sulphur and iron, and

comparatively low in copper, a combination that made a matte too poor in copper and too rich in iron, part of the ore was roasted, or otherwise part of the matte had to be crushed and oxidized. At first hand-rabbled reverberatory furnaces were used for roasting, then the Bruckner cylinders, the Brown horseshoe furnace, the Holtoff-Wethey, and others, until at last the majority of the copper smelters seem to have settled on the Herreshoff furnaces, or others somewhat similar known as the McDougall and the Klepetko-Evans-McDougall.

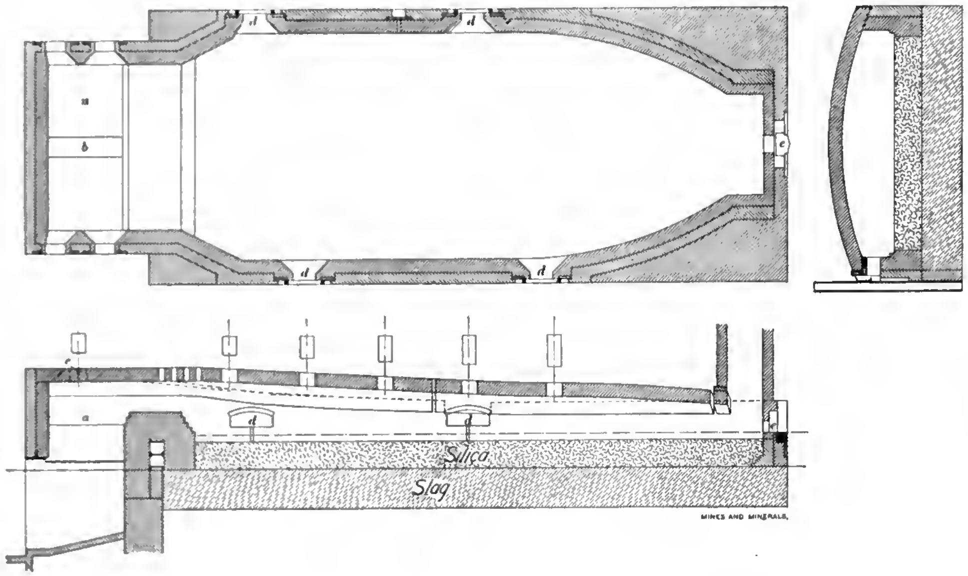


FIG. 3. LARGE REVERBERATORY FURNACE FOR SMELTING MATTE

The favorable acknowledgment that this style of furnace has received from copper metallurgists is owing to the cheapness with which it roasts ore compared with some others. The Herreshoff roaster is shown in Fig. 2. It consists of an iron cylinder mounted on pedestals, lined inside with firebrick. At intervals concave-convex arches are built of firebrick which answer as hearths on the upper side, and as reverberatories on the lower. In the center of the roaster is a hollow shaft through which air circulates, and attached to this shaft are rabble arms carrying teeth pitched so that when the shaft is revolved the teeth on the arms will move the ore to or from the center of the hearths. The shaft is turned by the gears shown underneath the lower hearth. When the furnace is put in operation the ore is admitted to the top hearth near its center, and by means of the revolving arms is worked slowly from the center to the circumference, where it falls to the second hearth.

TABLE 1

Name	Size	Per Cent. Sulphur in Ore	Per Cent. Sulphur in Product	Tons Roasted in 24 Hours	Cost Per Ton
Hand reverberatory	69.5′ × 16′	35	7.5	13.0	\$2.00
Allan-O'Hara	2.94′ × 9′	35	8.0	51.0	.78
Bruckner, 8½ revolutions per hour	8′ × 16′	37	9.5	10.0	1.25
Pearce double-decked	7′ hearths	35	6.5	42.0	.98
Herreshoff 5 hearths	10′ 10″ diam.	35	6.0	6.0	.50
McDougall 6 hearths	15′ 10″ diam.	35	7.0	40.0	.35
Klepetko-Evans-McDougall		35	7.5	37.5	.34

On the second hearth the teeth are pitched to work the ore from the circumference to the center, where it falls to the third hearth, and so on alternately from the center to the circumference and back to the center of the next hearth, until it is finally discharged at the center of the lower hearth. After the sulphur in the ore has been ignited in the lower hearth, no fuel is required, as the heat rising upwards follows the roofs to the exits and thus heats the descending ore, until the sulphur in the latter takes fire and keeps up the heat necessary for the roast.

The rabble arms of the Herreshoff roaster are solid, and owing to their corrosion by the sulphur and sulphur dioxide do not last indefinitely. When one is badly corroded the revolving shaft is stopped so that the arm will be in line with a door, which is then opened so that the arm can be withdrawn and another substituted. In order to reduce the cost of repairs the McDougall roasting furnace was constructed with water-cooled arms, and has now become more popular than the Herreshoff.

Table 1 herewith gives the size, capacity, cost per ton of roasted ore, and the percentages of sulphur in the ore before and after treatment in the roasters named.

Quite a number of reactions have been advanced to show the various chemical changes that occur during the roasting period, based on the fact that pyrite, FeS_2, loses one atom of sulphur when heated in air to a temperature of 516° C., and also on another fact that copper has more affinity for sulphur than any other metal. Starting with a chalcopyrite, $CuFeS_2$, the following reactions are assumed to occur:

1. $CuFeS_2 + \text{heat} = CuS + FeS$
2. $CuS + FeS + 3O = CuS + FeO + SO_2$
   158 160
3. $2CuS + 2FeO + 3O = Cu_2S + Fe_2O_3 + SO_2$
   158 160 48
4. $Cu_2S + Fe_2O_3 + 3O = Cu_2O + SO_2 + Fe_2O_3$

On examining the table it will be seen that the sulphur remaining in the different roaster products varies from 6 to 9.5 per cent. This shows that the roasting was incomplete; and as copper has more affinity for sulphur than iron, the product will contain sufficient copper sulphide to satisfy the sulphur.

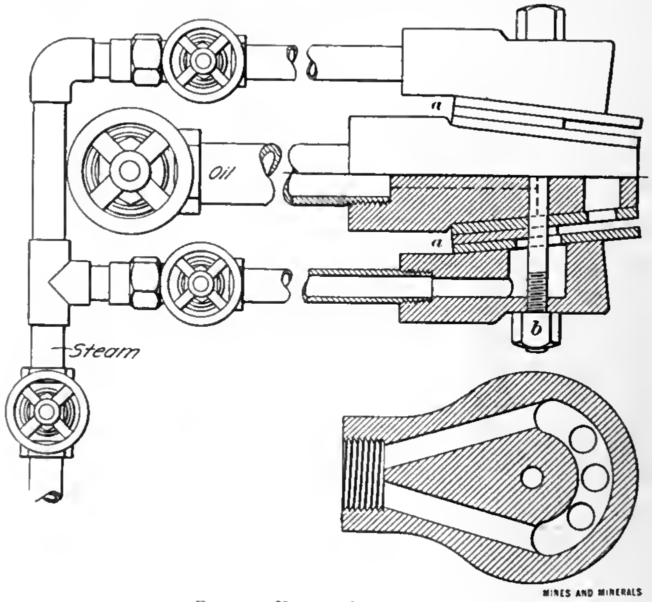


FIG 4. KITTLE OIL BURNER

From the molecular weights given in equation 4 there is 43.17 per cent. of the ore cuprous sulphide, or in every 100 pounds there are 43.17 pounds. Since 20.25 per cent. of Cu_2S is sulphur, there is 34.43 pounds copper in the ore and

8.74 per cent., or pounds, of sulphur; but 6 per cent. sulphur, or 6 pounds, remained in the roasted product, and to satisfy this it will require 8.74 : 6 = 34.43 : 23.64 pounds copper, and the copper remaining to form cuprous oxide is 10.79 pounds. Instead of equation 4 the roaster would furnish the following product per 100 pounds:

5. Fe_2O_3 = 43.71 pounds
Cu_2S = 29.64 pounds
Cu_2O = 15.16 pounds
SO_2 = 11.49 pounds

because there is 13.12 pounds of oxygen in equation 4 to be satisfied. One of the principal laws in chemistry may be stated as follows: "It requires as much heat to separate a mineral as to form it," but so far that quantity of heat has not been determined for chalcopyrite, consequently thermochemical data are wanting for equation 1.

The heat of formation for ferrous sulphide, FeS, is 429 calories; for cupric sulphide, CuS, 159 calories; for cuprous sulphide, Cu_2S, 160 calories; for ferrous oxide, FeO, 1,173 calories; for ferric oxide, Fe_2O_3, 1,746 calories; for cuprous oxide, Cu_2O, 344 calories; and for sulphur dioxide, SO_2, 2,164 calories. All the heat units or calories mentioned have been determined by experimenters.

In the second equation the copper is unchanged, but the ferrous sulphide is broken up and oxidized to FeO and SO_2.

The heat absorbed to separate FeS = 429 calories
The heat evolved to form FeO = 1,173 calories
The heat evolved to form $SO_2 = \frac{32 \times 2{,}164}{56} = 1{,}236$ calories

Net heat evolved 198.0 calories

In the third equation cupric sulphide furnishes an endothermic reaction, and cuprous sulphide an exothermic; at the same time the ferrous oxide is endothermic and the ferric oxide exothermic.

The heat absorbed to separate CuS = 159 calories
The heat absorbed to separate FeO = 1,173 calories
The heat evolved to form Cu_2S = 160 calories
The heat evolved to form Fe_2O_3 = 1,746 calories

Net heat evolved 574 calories

In equation 4 the cuprous sulphide is endothermic and the cuprous oxide and sulphur dioxide exothermic.

The heat absorbed to separate Cu_2S = 160 calories
The heat evolved to form Cu_2O = 344 calories
The heat evolved to form $SO_2 = \frac{32 \times 2{,}164}{126} = 549$ calories

Net heat evolved 733 calories

FIG. 6. TAPPING AND GRANULATING SLAG, ANACONDA FURNACE

FIG. 5. INTERIOR OF ANACONDA REVERBERATORY FURNACE

The calculations have been formed using pound calories; that is, the heat that will raise 1 pound of water 1° C., and have been based entirely on the theoretical supposition that the furnace roasted pure sulphides when in practice. This is merely approximated. Again, the calculations have been based on a dead or oxidizing roast, when the resulting product from the roaster was assumed to contain 29.64 per cent. Cu_2S; 15.16 per cent. Cu_2O, and 11.49 per cent. SO_2, therefore, no matter how pure the minerals may be, the calculations should not be carried beyond the sulphur in the roasted ore. To approach practice, the final calculations are based on the 6 per cent. sulphur in the roasted product; but it is to be remembered that the ore before the final roasting in 4 contained 43.17 per cent. Cu_2S, and after roasting 29.64 per cent. Cu_2S.

The heat absorbed to separate
13.53 per cent. $Cu_2S = 160 \times .1353 = 21.65$ calories
The heat evolved to form Cu_2O ($.1516 \times 344$) = 52.15 calories
The heat evolved to form SO_2 ($.1149 \times 594$) = 63.08 calories

Net heat evolved in last operation 93.58 calories

The deductions from such calculations, while merely approximate, show the metallurgist that for a dead roast he must use the hand-rabbled furnace with the extra expense for fuel and labor attached; that in a mechanical roaster he must use the purest possible concentrate; and that without iron sulphide he must use fuel to generate sufficient heat to roast.

There would be great difficulty in roasting the mixed sulphide product termed matte, if it were not for the iron it contained; and where, as in Chili, the minerals atacamite, $CuCl_2\,3CuO + 3H_2O$, and covellite, CuS_2, are concentrated, pyrite carrying copper is a necessity in order to obtain a suitable matte.

It was assumed for the purpose of heat calculations that the final product from the roasters was reached when the 6 per cent. sulphur was distributed to satisfy the copper; this, however, is unlikely, because the ore cannot be roasted so uniformly as to produce theoretical results, and it is undoubtedly true that there will be iron and copper sulphides, and iron and copper sulphates, as well as oxides of both metals, in the product coming from the roaster. The net heat units evolved in the final calculations where sulphates are formed in such small proportions would not vary much from the 93.58 calories found; on the other hand, if the sulphides were oxidized to sulphates directly and completely, two or three times

as much heat would be given off as in oxidation, and this would create incipient fusion and loup the ore.

At the Yampa smelter*, Bingham, Utah, there are nine McDougall roasters, 18 feet in diameter. The rabbling arms to these roasters make one revolution every 58 seconds, and as there are six decks it requires about 1 hour and 15 minutes to work a charge through. Each roaster has a daily capacity of 47 tons of concentrate from which it eliminates about 83 per cent. of the sulphur. The McDougall dust assays on an average: Copper 2.36 per cent., iron 30.5 per cent., silica 25.8 per cent., sulphur 21.2 per cent., lime 2.6 per cent., alumina 7.4 per cent.

The accretions which form on the sides of the roasters are barred off and trammed to a bin from which they are mixed with the charges going into the blast furnace.

A 20-horsepower direct-curent motor drives the McDougall roasters.

The roasted ore is discharged from the McDougall roasters into cars, having a capacity of 7,300 pounds each, and hauled by a motor to a building adjacent, where there are three reverberatory furnaces two of which are in continuous operation. The furnaces (a drawing of which is shown in Fig. 3), are 17 feet wide, 47, 53, and 59 feet long, each with a rated smelting capacity of from 150 to 175 tons of raw ore per day, but they actually treat much more. The fireboxes *a* are of uniform

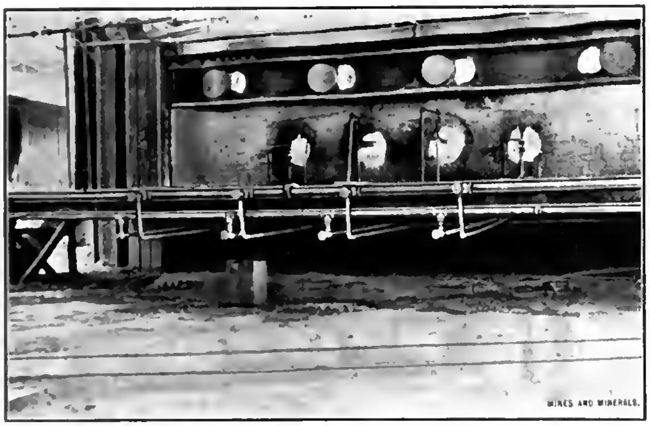


Fig. 7. Oil-Fired Reverberatory Furnace

size, 9 ft. × 12 ft., divided into two compartments by a brick wall *b*, so that each compartment is 5 ft. 6 in. × 9 ft. Each side of the firebox is connected to a 15-inch air line, while the fuel is fed from the roof through three 15-inch openings *c*. Draft for the fireboxes is supplied by a 16-inch centrifugal blower at a pressure of considerably less than 1 inch of water. A 20-horsepower induction motor drives the blower.

The furnaces are of brick, stayed with 4-inch I beams tied across the top and spaced on 16-inch centers, except where two doors *d* 15 in. × 24 in. have been left in each side to allow the furnace men to feel the charge. A 12″ × 15″ door *e* is in the back of each furnace for skimming.

Between each furnace and the stack is a 300-horsepower water-tube boiler utilizing the waste heat from the furnace to generate steam at 110 pounds pressure. These boilers deliver an average of 180 horsepower, effecting thereby an economy equal to 150 tons of coal for each furnace per month. This 180 horsepower represents an efficiency of only 60 per cent., which is small

The construction of the reverberatory furnace of the modern type requires much more skill and material than one of the smaller ones. This is due to their width and enlarged fireboxes, to the systematic methods of charging material and fuel, followed by handling the matte and slag. Furnaces are made as wide as 19 feet, and this feature necessitates great care in constructing the arched roof which in any case is low and flat; and in placing buckstaves as near together as possible that they may resist the expansion which ensues from the flat arch and heated brickwork.

*Leroy Palmer in Mines and Minerals, Vol. 31, page 14.

In Fig. 5 is shown the interior of one of the large furnaces at Anaconda, Mont., which illustrates to those accustomed to smaller reverberatories the absolute need of the increased care that is used to reinforce the sides.

The roof of the reverberatory is made of firebrick, as is the inner wall, which comes in contact with the flame and the hot material. The bridge wall between the firebox and the hearth is of firebrick, and has flues for admitting air for the combustion of gases arising from the coal. Flues for the same purpose are placed above the firebox, the object being to furnish sufficient air to completely burn the volatile hydrocarbons of the fuel and prevent smoke, which cools the furnace walls and the top of the charge. While the waste heat of the furnace is utilized to generate steam, and thus economize in fuel, no particular or serious attempt has been made to preheat the air previous to combustion and further utilize the heat that escapes. In this connection it may be stated that, from experiments made by Dr. E. D. Peters, it was found that one-quarter of the coal consumed by the reverberatory smelters performed useful work, that is, heated the charge, while three-quarters of the coal was required to heat up the walls of the furnace. Owing to the peculiar construction of reverberatory smelting furnaces there is great radiation, and ready chilling takes place when the working doors are opened for any purpose. Any heat, therefore, be it in the air for combustion or in the charge, saves a certain quantity of coal.

The hearth of a modern copper reverberatory furnace is made of quartz sand from 24 to 36 inches deep. The top layers are dried and sintered and are not corroded so long as oxides do not reach them; and in hot-matte covered hearths oxides are not likely to sink as deep as the hearth. Silica will not unite with sulphides, but will with oxides, and since there is not a blast of air passing through the matte to form oxides, no corrosion occurs except at the level of the liquid material in the furnace. The firebox is made of a size to conform to the area of the furnace, the object being to burn as much coal as possible in as short a time at possible. The Anaconda furnace, which is 102 feet long by 19 feet in width, has a firebox 7 feet long and 16 feet wide. This ratio of the hearth to the grate, 16 : 1, all silver men will agree on as being the proper one. In this reverberatory furnace about 285 tons of charge is smelted daily with 57 tons of coal; or 5 pounds of ore are smelted to every 1 pound of coal burned. When this ratio of fuel to ore is compared with blast-furnace smelting where 1 pound of fuel smelts from 8 to 9 pounds of charge, economy in reverberatory smelting seems to be wanting; however, fuel is not the only consideration, and when the details of machinery, kind of ore smelted, and the higher priced fuel are taken into the calculation, the comparative cost will, under usual circumstances, favor the reverberatory. In order that the fuel may burn freely, there must be a free draft to keep the products of combustion moving out of the furnace, and as the heat expands the hot gases enormously, the area of the grate to the area of the stack is an important ratio, which in general may be stated as follows: The smaller the area of the grate is in proportion to the area of the stack the greater will be the draft; but it must be remembered that the larger the area of the grate the more coal can be burned in a given time, and that combustion increases about in the inverse ratio of the area of the grate to the area of the chimney. A good ratio for a large reverberatory would be where the area of the grate was twice as large as the area of the stack, as this ratio would permit the combustion of from .65 to .75 pound of coal per minute on each square foot of grate area. The Anaconda furnace burns about .7 pound of coal per minute per square foot of grate area.

In modern reverberatory practice the matte is never entirely tapped from the furnace for three good reasons: (1) It protects the hearth bottom from oxidized ore; (2) it retains heat and keeps the furnace hot, thereby hastening the melting process; (3) it does away with spreading the ore by hand, as fresh ore dropped upon a bath of matte spreads over the matte without assistance.

The third reason also saves heat, and as the ore is generally heated when charged, considerable economy is reached by this method over the old style method of spreading the charge by means of tools inserted through the working doors in the sides of the furnace.

As the reverberatory process of matte production is one of melting and liquidation rather than one of reduction, the charges are not calculated, neither is flux added, but there must be conditions favorable to quick melting, and these are brought about by the use of fine ore, not lumps, composed of silicious sulphide concentrate, roasted concentrate, and flue dust. This combination will furnish a richer matte, but a more refractory slag containing more copper than the blast furnace products. Several ore charges are allowed to melt down and then the matte is tapped as the converters require it. The slag is allowed to flow off from time to time through a trough, as shown in Fig. 5, into a stream of water which granulates it, but not enough is allowed to flow away at any one time to more than lower the contents of the hearth to a proper level and make room for a fresh charge of ore.

Where 30 years ago reverberatory smelting was an intermittent and tedious operation, at the present time it is practically continuous.

Within the last few years oil has been substituted for coal at Cananea and at the Arizona smelting companies' plants with excellent results. It is said that with this fuel higher silica slags and lumps of ore up to 2½ inches in diameter are used without materially retarding the furnace. At one of the Arizona smelting companies, two oil-burning furnaces smelt from 300 to 350 tons of charge in 24 hours, producing 40 per cent. copper matte from 5 per cent. copper ore and 12 per cent. sulphur. To smelt such ore 1¼ barrels of California oil is used, costing $1.25 per barrel of 42 gallons.

In the January, 1910, Mines and Minerals, there is an article by R. L. Herrick on the use of oil in the large reverberatory furnace at Cananea. Four burners are used, as shown in Fig. 7, placed in the front end of the furnace. Details of these burners are shown in Fig. 4. The oil is atomized by steam coming from the waste-heat boilers, 1 pound of oil requiring .4 pound of steam for the purpose. In reverberatory furnace work the aim is to produce a flame that will roll the length of the furnace, and this is accomplished by the burner shown. Where this furnace treated 100 tons of flue dust and calcines per day with coal, the capacity was increased to 250 tons, with the consumption of .9 barrel oil per ton smelted.

COTTRELL FUME-CONTROLLING DEVICE

The Balaklala Copper Co. has installed the fume-controlling device invented by F. C. Cottrell, of the University of California. In this method the solid particles, which cause the damage to crops, are separated by static electric discharge. The plant will have an electrode area of 100 square feet, and is designed to handle 70,000 cubic feet of smoke per minute. The device consists of a precipitating chamber in which are placed 126 electrodes. These are strips of iron 4 feet long by ¾ inch wide, with saw teeth their entire length. Between the electrodes are placed flat plates of iron about 6 inches wide. An electrical current of 27,000 volts is delivered to the electrodes, and as the smoke circulates between the positive and negative electrodes, the solid particles are precipitated on the adjacent plates.

The experimental plant shows that 90 per cent. of the solid particles carried by the smoke are precipitated by this method. The gases from the main flues are drawn into the precipitating chamber and are driven into the main stack by two fans 186 inches in diameter, whose combined capacity is 500,000 cubic feet per minute. There will be nine permanent units in the plant, seven for constant service, and two for reserve purposes. The total cost will approximate $125,000.

AIR FROM A HYDRAULIC COMPRESSOR

Written for Mines and Minerals

Readers of Mines and Minerals will remember that the lights would not burn in the mines at Cobalt that were supplied with compressed air from the Ragged Chutes hydraulic air compressor. Messrs. F. W. McNair and G. A. Koenig were commissioned to examine the conditions that existed at the Victoria Mine, Ontonagon County, Mich. The results of their determinations were given out in a paper read before Section D of the American Association for Advancement of Science at the Minneapolis meeting, parts of which are quoted.

The fact that water in intimate contact with air not only dissolves it but takes up a greater percentage of its oxygen than of its nitrogen, has long been known. Each successive solution will render the dissolved air richer in oxygen. Since the hydraulic air compressor delivers, not the air dissolved, but what is left, it follows that the air delivered must be poorer in oxygen than normal air.

Determinations of the hydraulic air made by means of a Hempel pipette, charged with thin sticks of phosphorus, showed an oxygen content of 17.7 volumes to the 100 volumes of compressor air, whereas in 100 volumes of normal air there are about 21 volumes of oxygen. The air from the Cobalt compressor gives 17.7 volumes of oxygen to 100 of air, which agrees with the determination at Victoria. This is what may be expected from a single solution. If, out of 100 volumes of normal air, the water extracts 2.15 volumes of nitrogen and 4.47 volumes of oxygen, the ratio of solution will be that given in textbooks, and the resulting percentage of oxygen will be found as by the determination. It would at first seem doubtful that this shortage of 3.3 volumes of oxygen could cause the reported trouble with the lights.

To investigate the action of the Victoria air six candles were chosen. The candles were first lighted and observed on the surface to judge the normal rate of burning, the height and general appearance of the flame, together with the cup at the bottom of the wick. On lighting these same candles underground a marked difference in their burning was recognized. The flame was in every case lower than normal and much more blunt. The tail, partly of semiluminous and partly of sooty carbon, so characteristic of the flame in normal air, was as a rule wholly absent. The cup was only meagerly supplied with melted wax and showed a frozen appearance around the edge. The candle was also easily extinguished by a sudden sidewise movement.

Evidently the deficiency of oxygen in the compressor air is wholly responsible for the difficulty with the candles. The lower oxygen content allows only a slower combustion, heat is supplied more slowly, the temperature remains lower, the wax melts at a slower rate and is liquid over a smaller area.

Regarding the effect of this air on persons working in it, conclusive evidence is not easily obtained; however, it had no effect on the experimenters, although they purposely exerted themselves to ascertain its toxic action. It is interesting to note that this solution of air lowers what might otherwise be the efficiency of the hydraulic compressor. For instance, if all the air at Victoria mines was drawn in and delivered compressed to 114 pounds gauge, the compressor when absorbing 1,961 horsepower would show an efficiency of 82 per cent. If it is assumed, as above, that 6.6 volumes of each 100 drawn in are diverted, the efficiency cannot be above 76.6 per cent.

CORRESPONDENCE

Safety in Mines

Editor Mines and Minerals:

Sir:—The following are just a few hints, with the hope of bringing forth from others some new and better ideas.

In conversation with an old miner recently, the question of accidents and recent disasters were freely discussed, and several points were brought forth concerning the safety of coal mines, some of which may be worth while reading. While these questions have been discussed many times, it may be well to again mention some of them in order to arouse those interested in mining, and bring out some new and better ideas.

When I think of the many catastrophies of the past in coal mines, it makes me wish that I were young again to get into line and try to employ some means that would eliminate at least some of these, for in my opinion many of them, with a little foresight, and with forcible leaders, could be avoided.

From time to time we read opinions from our most able observers as to the dangers of the dust, gas, and inflammable structures, but few seem to advance any forcible ideas that are stringent enough to eliminate their dangers. Until the government officers get the experience, I am afraid these things will go on, because our present system of mine inspecting is not able to handle the question as it should be. Rigid measures, regardless of cost, should be used. Wherever safety appliances and progressive improvement are installed and tried out, you will find the strictest economy, fewer accidents and cheaper coal with better general results. Conditions locally are a large part concerning these matters. Our under-managers have a habit of traveling a beaten path and too often carelessly leave very important points to lieutenants who, from lack of experience, take everything for granted. This is another common result of lack of force. If you were to go through some of the larger mines, you would frequently hear the foreman, or underforeman, inquire of some of the employes as to the conditions of a certain point, or ask a miner about the condition of the roof of the place in which he works, the answer would invariably be that the places inquired about needed repairs and that the needed repairs would be made as soon as the miner could get time to make them. The miner will, in nearly every case, promise to take down the bad roof, or to set a prop under it in order to make it safe, "after I have loaded another car," or that "he will see to it first thing in the morning." These answers are absolutely wrong. The men in charge should have these repairs done without delay. The bad roof should have been safely propped up or taken down as soon as it was reported, and other dangers, which are apt to result in the loss of life, attended to at once. Any one refusing to act promptly in such cases, should be immediately suspended. If the putting off until tomorrow was eliminated, more than one-half of the local accidents would be avoided. You will often hear the remark, from men concerned, when an accident occurs, that it was the victim's own fault. "I told him to make his place safe." That order, telling him to make his place safe, might be all that the law required, but it would have been better if the foreman had seen to it that the order was carried out at once. All such orders should be rigidly enforced.

Yes, all mines should be so planned and the coal so removed that a sufficient pillar will be left for the walling off as soon as any portion of the mine is abandoned. Dust and fires in old workings cause more disastrous explosions than any one thing that I know of. When such places are properly walled off and the stoppings carefully looked after, there is no more trouble from them. Keep the air away from the old works and you will have no trouble with them. Also keep a good current of air playing against the old work stoppings in case of a leak and there will be no cause to worry over them. This, with keeping the mine in a damp condition, will make an extended explosion impossible. The trouble, if there be any, will be local and its spread will be very limited.

I have had several local explosions during my time, caused from a strong feeder of gas, or a door left open carelessly, but it never extended for any distance, and with no results more serious than scorching a few men. But if the mine had been dusty as a result of not wetting the workings, at the time of a local explosion, and opened old workings near by, I dread to think of the results. There is the key, give it a start and there is no telling the end.

I know it is impossible to eliminate all the dangers or explosions, as oftimes the one you place the most confidence in proves to be the most negligent and with dire results. With all these dangers in mind, why not try to figure out something that will give the man under ground more heart? Since he places all confidence in the management, it seems a duty of the men in charge to try something automatically to help him. I would like to see a system of piping tried out. A system so arranged as to escape the force of an explosion as much as possible. These pipes could be taken, or laid from the surface into the mine with branches laid in every entry with laterals and proper connections at the most convenient points, to which a hose could be attached that would reach any point of danger. In case of an explosion, could it not be arranged that an air compressor be installed at each and every mine and connected to the water mains to furnish air to large chambers, installed at different points in the mine, where imprisoned miners could get a supply of compressed air and find refuge until rescued? From this system of piping through which air could be pumped into the mine, some at least of the men could live instead of being suffocated in the air deprived of its oxygen.

Cherry and Leyden and many others have taught us lessons that we should never forget. We should try to enforce some vigorous action to better the present conditions and some measures to overcome the agonies of the afterdamp.

Ogden, Utah — A Miner

Warnings to Colliers

Editor Mines and Minerals:

Sir:—Under the title "Warnings to Colliers," there used to be issued in newspapers circulating in the Midland and Yorkshire coal field of England 45 to 40 years ago, notices signed by William Cooper, F. M. S. (Fellow of the Meteorological Society), calling the attention to coming falls of the barometer, the inception of which had been observed.

In these notices, firemen were warned to be extra vigilant in their inspections of the workings, and to exercise special care in seeing that all brattices and stoppings were tight, and in good order; and sometimes they were enjoined to particularly examine the lips of the gob, which at the time meant crawling into the gob and reaching up the faces of the breaks.

Some of these colliery managers, who were then unfortunately in charge of pits where the ventilation was chronically bad—and this was a numerous class—were liable to judge these warnings as of something special and beyond what could be observed in the pit itself.

The warning which the mine gave always anticipated those in the newspapers, sometimes by many hours.

The writer was, in the fall of 1868, newly in charge of a colliery in North Derbyshire, which gave off firedamp freely, with very bad ventilation, which could only be changed slowly. The motive column, 500 feet, was set up by a fire-basket. There were over 300 men employed underground.

We were sailing very close to the wind of disaster, and were very conscious of it, and a heavy fall of the barometer was always looked forward to with anxiety, and when it came the

addition to the area over which the gas constantly showed was carefully watched and noted.

Locked safety lamps were exclusively used and strict discipline was easily maintained, the overshadowing calamity at the Oakes colliery being still fresh in the memory of both officials and colliers.

The chief night foreman was somewhat of a wag, as well as being a very capable and cautious examiner. So, in addition to making a circumstantial report in the official book when the gas cap showed any sensible lengthening, at the close of the shift, this man would come across the fields to my house and call out from outside, "Look out for Mr. F. M. S." This would occur at 5 o'clock in the morning, and the writer would quickly make his way to the colliery. If the barometric disturbance occurred early in the night, the warning would appear in the newspaper which we would get at 9 A. M. If it occurred after say 11 o'clock, the newspaper would give it the next day; 35 hours after it had been observed already in the pit.

It is a simple conclusion that an incipient fall of the barometer will show itself very readily in the mine, such as is described above, by the expansion of the gas and air in the goaves, fully as soon as it will on any instrument at the surface, and *this* warning comes direct to the collier, whereas observations at the surface, like Mr. Cooper's, must be read, recorded, transmitted to the newspaper office, set up in type, printed, and sent out through the ordinary channels of circulation, entailing a loss of many hours.

At the above-mentioned time there was one colliery where the height of the barometer determined whether it should work or play. This was Shipley Gate colliery, Derbyshire. When the barometer fell below a certain level (which cannot now be remembered), the men were withdrawn, the pit laid idle, and the fan—a large Guibal—increased in speed.

Jeannette, Pa. W. Clifford

Rope Haulage

Editor Mines and Minerals:

Sir:—We are interested in the letter of Mr. Chester Moon published on page 425 in Mines and Minerals for February, relating to the practicability of operating an endless rope haulage 1½ miles long in conjunction with a 35-degree incline plane to a tipple, and passing around a curve at the foot of the incline.

The rope in any endless-rope haulage must be run under a certain tension, and while it is practicable to run around curves on a level track, there would be an objection to extending the line up the incline plane, owing to the upward lift of the rope, which might cause trouble at the curve by the derailing of the cars, and we think therefore that it would be desirable to use an independent rope for raising the cars up the slope.

The proper sizes of ropes for the haulage and incline plane would depend on the output, and the number of cars hauled or hoisted at each trip, which information is not volunteered in Mr. Moon's letter.

We mail you with this two copies of our book on "The Application of Wire Rope to Surface and Underground Haulage," which may be of assistance in determining the best arrangement for Mr. Moon's purpose.

The Trenton Iron Co.
Wm. Hewitt

Steel Supports in Mines

Editor Mines and Minerals:

Sir:—Under the caption "Steel Supports in Coal Mines" in your February issue Mr. R. B. Woodworth has an excellent summary of the use of steel in place of timber for permanent mine workings. A good deal of the material is not new, but it is the most complete review of the subject that I have seen and it will prove of value.

There is one point, however, upon which Mr. Woodworth is notably silent, and that is as to the impossibility of using steel as a mine support in properties where the ore is a sulphide, undergoing oxidation in the upper levels. This condition it seems to me absolutely prohibits the use of steel supports (steel timbers sounds altogether too anomalous), and without knowing the circumstances in detail, one is strongly inclined to think that the failure of the "few adjustable sets" in one of the western copper mines of which Mr. Woodworth speaks, was caused rather by the action of the H_2SO_4 and $CuSO_4$ laden waters than by any structural defect.

That preservative coatings count for little, we have ample evidence in the case of pump columns, where various non-corrosive alloys or else wood-stave pipe have to be used, steel or iron pipe in spite of most careful coating often lasting not more than a week or two. I question very seriously whether any ordinary paint such as is described will serve the purpose; slight movements of the ground, the rubbing of men's clothing covered with grit and dirt, cars getting off the track, hitting with pick and shovel and the like, soon remove the paint in spots and once the acid mine waters begin to attack the iron they will speedily creep along behind the paint layers exactly as rust gets beneath the paint on steel structures at the surface, only much more rapidly.

This problem is a serious one and will undoubtedly be solved in time—possibly by some heat-treatment of the steel—for steel is bound to replace timber. Concrete, either plain or reinforced, is too heavy for most uses and has the serious disadvantage of giving no warning when it is taking heavy pressure. Francis A. Thomson

State College, Pullman, Wash.

Editor Mines and Minerals:

Sir:—I have duly noted Professor Thomson's letter of February 15, with the criticisms which he has to offer on my paper entitled "Steel Supports in Coal Mines." There are three points in this letter calling for consideration:

1. I use the words "roof supports" to indicate beams or channels laid on the coal resting on side walls or supported at one or both ends by wooden props. The words "gangway support" I construe to indicate the three-piece gangway set or a steel roof support with one steel leg, the whole structure in either case being a rigid one, which may be considered as a unit. The designation "steel mine timbers" I have coined to indicate the regular use of steel in mine timbering uses as contradistinguished from wooden mine timbers, or possibly as contradistinguished from concrete mine timbers. At the first blush the term may appear anomalous. It is, however, a very handy trade designation, is coming into quite extended use, and has the merit of exactly designating the particular use to which the material is put.

2. My information as to the failure of the few adjustable sets installed in one of the copper mines of the Far West is precise. I do not care to name the installation, but the reason stated in my paper is correct. The failure was not in any way due to the action of acid-laden waters.

3. I have not made any experiments in the protection of steel against waters laden with copper sulphate. So far as waters laden with free sulphuric acid or ferrous sulphate in solution, which is even worse, are concerned, I have no reason to doubt that proper protection to the steel is afforded indefinitely by the simple means stated. The Professor will find a brief notice of my experiments and conclusions in this direction in Cushman and Gardner's recent work on the "Corrosion and Preservation of Iron and Steel," to which I contributed some half a dozen pages, and he will note that my conclusions in this respect are indorsed by the editors of that work. The theorectical considerations mentioned by the Professor are not corroborated by experience in the actual use of steel mine timbers in this country and abroad. My own recommendations

in the proper preservative treatment of steel have back of them both laboratory experiments and wide experience. I have sufficient confidence in the results of advanced research into the preservation of iron and steel structures to think that even in copper mines it is possible to preserve steel mine timbers intact over any reasonable period of use without fears of any structural weakness due to the action of copper solutions. The use of steel for pump columns is by no means a parallel case.

I am unable to see in what way heat treatment of the steel can protect it, provided the chemical constituents remain unchanged. Certainly the successful introduction of steel for mine timbering purposes is based on the elimination of any factor which would in any way tend to increase its relative cost, and experience in other lines indicates that the problem of underground mine timbering is in no way different from the use of structural steel in buildings and bridges generally. In fact, the large merit of the proposition from the standpoint of the mine owner and operator is the interchangeability of steel with a material with which they are already perfectly familiar, and the consequent simplicity of its use.

R. B. Woodworth

427 Carnegie Building, Pittsburg, Pa.

ORE NOTES

After 6 months spent in development and the placing of new machinery, shipments have been renewed from the Colonel Sellers, of Leadville, which is owned by the Empire Zinc Co. It is reported that the company has taken $350,000 out of the zinc ore on the dump.

Plans for another drainage tunnel in the Cripple Creek district are under consideration. T. R. Countryman, the construction engineer of the Roosevelt tunnel, has been asked to prepare a report. The Roosevelt tunnel will add 730 feet to the workable depth of the Cripple Creek mines, and it will probably be 6 or 8 years before the drainage effected by this tunnel is completed. According to the plans now under consideration, a tunnel 500 feet deeper is to be driven. By taking plenty of time for the work, it is believed that the cost of driving the proposed tunnel can be kept within $500,000.

The Standard mine, near Silverton, B. C., is said to have run into a body of silver-lead ore 41 feet wide. In the same vicinity the Rambler-Cariboo mine ran into the largest and richest ore shoot ever found in that property on the 1,050-foot level. The ore is said to average 250 ounces silver and 75 per cent. lead. The ore shoot is about 5 feet wide and has been drifted 150 feet on level.

Patrick Stewart, manager of the Canadian Consolidated Mining and Smelting Co., has entered upon a vigorous policy of encouragement to mine owners, by reducing by 33⅓ per cent. the rates on freight and treatment to the smelter at Trail, B. C., on certain classes of ore. The camps which will benefit at once by this departure are those in the Salmon River section of Kootenai, including Ymir, Sheep Creek, and the North Fork.

Advices from Grand Forks, B. C., say that the shipments of blister copper from the Granby smelter there for the week ended February 2 were the largest in the history of the plant. The output was 440,000 pounds—45,000 pounds more than in the preceding week, and 1,000 pounds more than the best previous records. Receipts of ore from the company's mines aggregated 22,638 tons, which is just about the weekly average since January 1, when shipments were greatly increased to feed the full battery of eight furnaces. Receipts of ore from foreign mines totaled 535 tons, or 100 tons above the average.

J. L. Whitney, who has just returned to Spokane from Missoula, Mont., says that the Marsh mine is the talk of mining men in that city. United States Senator Dixon and Charles Cowell are largely interested in it.

Federal Mining and Smelting Co. has placed an order for a sorting plant of 150 tons daily capacity at the Morning mine near Mullan, Idaho, to prepare ore for shipment direct to the smelter. Much ore has been bared on the property.

Advices from Baker City, Ore., are that 2 feet of high-grade ore has been encountered in the Susan D. mine, formerly known as the White Swan. The ore was found in the new workings on a drift from the bottom of a shaft of 122 feet. It is announced from the same source that the Virtue mine, with a record of producing more than $8,000,000, has resumed operations after an enforced idleness of several years as the result of litigation.

Bunker Hill & Sullivan Mining and Concentrating Co., operating at Kellog, Idaho, brought its total disbursements to $12,456,000 by the payment of dividend No. 161, amounting to $81,750, in February. Its ore reserves are extensive and the grade of its ore much superior in its deepest levels to what it was nearer the surface. It is predicted that it can continue to pay dividends at the rate of $81,750 a month for at least 20 years.

Follow the Ore.—Many instances are recorded in mining where the ore seemed to have entirely run out, and many mines have been abandoned on this account. This is particularly noticeable in dolomitic-limestone, where the ore is found in large bodies, but it is also the case in some other rocks. If in such places a careful inspection is made of the walls there will probably be found a crevice through which the ore-bearing solutions escaped, and if this be followed there are good chances of finding another deposit. The Moose mine, Colorado, and the Ontario mine, Utah, both silver-lead producers, lost their ore, but in both cases another pocket was found by following mere cracks. Ten years ago when the Silver Ledge mine, in San Juan County, Colo., was shut down it was supposed that there was not a pound of ore left in it. At the same time, on Nos. 3 and 5 levels a stringer of ore yielded all the ore it contained. It was not until recently, when it was discovered that a horse of rhyolite from 5 to 20 feet wide separated this stringer from the main ore body, that the great deposits being opened today were struck. On No. 3 level the body of ore, consisting of gold and silver-bearing lead and zinc sulphides, disseminated through a sericite talc gangue, sometimes reaching the width of 30 to 40 feet, and contained between two good but irregular walls, supplied the mill to full capacity. The mill is closed until April or May, the management having determined to develop the ore bodies during the winter months, in order to block out reserves that would supply the mill continuously with at least 150 tons of ore per day. The ore bodies have now been opened on No. 5 level, and the deposit has every indication of continuing to great depth below this. It is intended to sink the shaft to the depth of 1,000 feet and prospect the ore bodies at the successive levels below No. 5. Two classes of concentrates were made in the mill during the past season. One a zinc concentrate running 45 per cent. zinc and 8 ounces silver; the other a 79 per cent. lead concentrate carrying 9 to 10 ounces in silver. They were produced with ordinary milling devices, crusher, rolls, Chilian mills, Richards hydraulic classifiers, and Wilfley tables. The lead product was shipped to the American Smelting and Refining plant at Durango, and the zinc product to the zinc plant at Blende.

The Moose Smelting and Refining Co. is planning to erect a new plant, of which they will build a 100-ton unit at once. This smelter will be used to smelt the ore taken from the old Moose mines near Alma, in Park County, and property immediately adjoining. The original Moose mines were first discovered by Captain Daniel Plumber in 1870 and 1871, yielding between $4,000,000 and $5,000,000. J. L. Seward, the chief engineer, has left Denver for Alma to lay out the improvements and get the mines in shape.

Tungsten Ore in Arizona.—About 12 miles south by a little east of Benson, Cochise County, Ariz., in the Whetstone Mountains, an attempt has been made to mine wolframite, the tung-

state of manganese and iron, from deposits that are thought to be unlike anything heretofore described in the literature of ore deposits. The mineral occurs in a light-colored granite that is intrusive in mica schist and limestone, being found near the contact of granite and schist and in a tongue of granite 60 or 70 feet long which runs out into the schist. Most of it is in segregations in the granite similar to hornblendic and biotitic segregations in granite at many other places. The ore appears to be an original constituent of the granite. The deposit was formerly worked and a few tons of ore were taken out and shipped. It is said that as mined and hand picked the ore averaged 10 per cent. of tungstic trioxide. No work has been done for 2 or 3 years, and the deposit is of interest mainly because of the peculiar occurrence of the mineral.

Metal Mining in New Mexico in 1910.—The Mogollon or Cooney district, in the southwest corner of Socorro County, contains several gold mines, chief of which is the Ernestine mine, which is and has been for years the heaviest producer of gold in the territory. The Socorro Mines Co.'s mill, at Mogollon, was operated throughout the year; and the Helen company and the Treasure Mining and Reduction Co. were also producers in 1910. In the Magdalena district, a lead-zinc camp, in the central-western part of Socorro County, the operations of the Ozark Smelting and Refining Co., the Tri-Bullion Smelting and Development Co., the Germany Mining Co., the Mine Development Co., and the Mistletoe Mining Co. should show an increased output. The 20-stamp mill of the Rosedale Gold Mining Co., at Rosedale, which has been operated on gold ores, was burned in September.

In Grant County there was much prospecting with churn drills and some development of copper ore bodies. The Chino Copper Co., of Santa Rita, acquired in 1909 a total area of 2,800 acres, including the Santa Rita mines, and has been prospecting for 2 years with churn drills and developing the properties by underground work, as well as making an output both in crude ore shipments and concentrates from the old Santa Rita mill. The company is building a 3,000-ton mill at Hurley, 9 miles from the mine, to treat ore from the Santa Rita mines. This mill will be similar to that of the Utah Copper Co. plant at Garfield, Utah. Other mines of the company will be worked by steam shovel and the caving system. In November, 1910, the Chino company's reserves were estimated at 30,000,000 tons, assaying 2.6 per cent. copper. In the Burro Mountain district mining and shipping was carried on by the Burro Mountain Copper Co. The Chemung Copper Co. was engaged in prospecting until October, 1910. The Mangus Development Co., which was prospecting with churn drills, stopped work temporarily in October. The mines of the Savanna Copper Co., in Burro Mountain and Pinos Altos districts, were worked under lease. In Lordsburg district the principal mines were the Bonnie, Eighty-Five, and Superior.

Colfax County placers maintained an even yield for the year. Shipments from Dona Ana County were not equal to those of 1909. Lincoln County maintained its gold output from amalgamating mills at White Oaks. Luna County produced chiefly lead, from Cooks Peak and Victoria districts.

The Oro Grande Smelting Co., at Oro Grande, was idle in 1910. Some placer gold was recovered at Oro Grande during the year. The Tularosa Copper Co., at Tularosa, completed a 100-ton mill and started shipping copper concentrates in November.

The Boston-Cerrillos Mines Co., at Los Cerrillos, Santa Fe County, which produced gold, silver, copper, lead, and zinc in 1909, was reported to have suspended work in December, 1910.

There was considerable activity in Sierra County, particularly in the Black Range district, near the towns of Chloride, Fluorine, and Kingston, and in the Las Animas district, near Hillsboro.

As a great part of New Mexico ore goes to the smelter at El Paso, Tex., the enlargement of this plant, started at the close of the year, is of interest. The Chino Copper Co. has contracted to ship its concentrates to El Paso. The Ray Consolidated Copper Co., of Arizona, will also ship to this plant.

The Director of the Mint estimates a production of gold, of \$397,974 in New Mexico in 1910, against \$252,800 in 1909 and an output of 683,111 fine ounces of silver in 1910 against 324,200 fine ounces in 1909.—*U. S. Geological Survey Report.*

Mercury Minerals From Terlingua, Tex.—The Terlingua mercury field, in Brewster County, Tex., has furnished some mercury minerals that are of unusual interest to students of mineralogy or crystallography. They are kleinite, montroydite, terlinguaite, eglestonite, and calomel. The first four were but recently discovered, and the chemical character of kleinite is still somewhat uncertain, though it is known to be a mercury-ammonium compound, the first such compound recognized in nature. Terlinguaite and eglestonite are oxychlorides of mercury; montroydite is mercuric oxide; calomel is the long familiar chloride of mercury. None of these minerals is important as an ore of mercury, the metal being obtained chiefly from cinnabar, but also in the native state.

Mining in Nevada, 1910.—In Churchill County, the Nevada Hills and Fairview Eagles were consolidated. In northern Elko County the Jarbridge district boomed. This being as yet a difficult place to prospect, not much has been done. Freight rate from Twin Falls, Idaho, \$50 per ton in summer and fall only.

In Esmeralda County the gold output was increased nearly \$2,000,000, and provided the Goldfield Consolidated had not been injured by fire the increase would probably have been greater. A steady production came from Rawhide. The Rawhide Coalition and the Rawhide Queen mines were consolidated in August, 1910. Silver-lead ore was shipped from the Lucky Boy district. The Pittsburg Silverpeak Mining and Milling Co. (120 stamps) took over the properties of the Silverpeak, Valcalda Gold Mining Co. in Silverpeak district. The United States Smelting Co. purchased the Eureka & Palisade Railroad in order to put it in commission and serve its Richmond-Eureka mines at Ruby Hill. Low-grade ore was discovered at Buckhorn, 20 miles west of Alpha on the Eureka & Palisade Railroad. In Lander County, Bannock, 14 miles southwest of Battle Mountain, attracted attention. Renewed interest was taken in the vicinity of the old Dean and Cortez camps southeast of Battle Mountain. The Austin-Manhattan Consolidated Mining Co. remodeled their mill. The Bamberger-Delamar properties, in Lincoln County, were permanently closed, and mines in Pioche district were cut off from shipments by floods carrying away part of the San Pedro, Los Angeles & Salt Lake Railroad.

The mines about Silver City and Dayton were the most active in Lyon County. In the Mason or Yerington district work was continued and interest was centered in the completion of the Nevada Copper Belt Railroad from Wabuska, on the Southern Pacific, to Yerington and Mason, and in the erection of a smelter at Wabuska.

In Nye County the large mines at Tonopah probably made an increased output, owing to greater mill capacity, in 1910. Both the deep and placer mines at Round Mountain and Manhattan, north of Tonopah, maintained steady production. At Johnnie and in the Bullfrog district, the mines and mills increased their yield of gold and silver.

The mines of the famous Comstock Lode, in Storey County, continue to make an annual production of over \$800,000 from the workings above the water level. It is reported that substantial progress was made in exploring the territory east of the larger workings. This work may be the means of opening new ore deposits and of reviving this district.

Whitepine County furnished approximately two-thirds of the tonnage of crude ore mined in Nevada. The consolidation of properties, which made it possible to mine the low-grade ores of Ely, was carried still further in 1910 by the Nevada Consolidated Copper Co. absorbing the Cumberland Ely, together with that company's interest in the railroad and smelter.

THE MINING INDUSTRY OF NICARAGUA

*Written for Mines and Minerals, by T. Lane Carter**

In the opinion of a great number of Americans the only thing that Nicaragua and the other Central American republics are good for is revolutions and fevers. Nicaragua has suffered from both these curses in the past. Both of them, however, can be removed, and will in time disappear.

Resources of the Country and Conditions To Be Encountered in Operation of Mines

Mining in Nicaragua has been hampered by malarial fevers in the swampy coast regions, although at the mines situated in the mountains few have suffered from fever. The extraordinary success of the United States Sanitary Corps in banishing malarial fevers in the Panama Canal Zone proves that the same happy results can be obtained in removing malaria from Nicaragua.

Revolutions have hurt mining in this country more than fevers; not so much by stopping work during the fighting, as by frightening away outside capital. A few months ago the worst revolution the country has ever known came to an end. Peace is restored, and the outlook for mining development work in Nicaragua is now better than ever before. Estrada, the new president, fully realizes the importance of developing the mines of his country. He is very friendly to Americans and Europeans, and will welcome them to Nicaraguan mines.

Recently an American was appointed to act as financial adviser to the new government of Nicaragua. Before long a start will be made on the construction of a railroad from Bluefields, the metropolis of the east coast of Nicaragua, to the west coast of Nicaragua. This line will be of great assistance in the development of the country.

The lack of capital is one reason why mining in Nicaragua is so backward. What has been done so far was accomplished by prospectors who with a little money made the mines pay from the "grass roots." We all know how hard it is to carry on mining without capital. It is as if a man tried to "lift himself up by his boot straps." Had a strong development company, with able technical and financial management, gone into the country a dozen years ago, the output, instead of being about $1,000,000 annually, as at present, would amount to ten times that figure.

Nicaragua has been a gold producer for several centuries, and while the Spanish adventurers did not have so much success in finding gold as in Columbia and Peru, still they found much of the precious metal in Western and Central Nicaragua. The gold is found in the hilly country about 80 to 100 miles inland, but the Spaniards never ventured past the coast swamps. Until 15 or 20 years ago the Pis Pis district, Fig. 1, which is today the most promising gold region in Eastern Nicaragua, was unknown to the white man.

During the rush to California in '49 and '50, many of the fortune seekers went there by way of Nicaragua. Some of them were so attracted by the chances of gold mining in Nicaragua that they remained. In this way several gold producers commenced work, and they have been turning out gold intermittently to this day.

So far gold is the principal mineral in Nicaragua that has claimed attention; for example, of the 500 mines and prospects about 490 are gold properties. This is in striking contrast to Honduras, the republic to the north, where silver appears to be the prevailing metal. No coal or promising copper deposits have yet been found, possibly because little search has been made for them.

Fig. 1. Map of Nicaragua

Another great obstacle in the way of the mineral development of the country has been the lack of railroads. There is not a railroad in Eastern Nicaragua, so that transportation facilities are today as in the time of Columbus. All mining machinery is carried from the coast in "pit pans" or canoes, paddled up the sinuous streams by Indians. To carry freight from Prince-pulca, the port on the Atlantic, to the mines, costs from 1.9 cents to 7 cents per pound, or $38 to $140 per ton. The lower figure is only possible to mines near the river; when freight is carried overland the expense is enormous.

For instance, in the mountains about 20 to 28 miles from the village of Tunky, there is a mine developing. To take freight on oxen or mules from Tunky to the property costs more than to bring it from New Orleans to Tunky. There are no roads, and it is therefore impossible to transport material by wagons. Through the dense tropical forests the natives cut a path with their machettes. Then the mules and oxen follow in this trail until their hoofs wear deep holes in the soil, and make a Nicaraguan highway. In the dry season this trail is bad enough, but in the wet season every time the animals take a step they sink to the belly in mud. With a barrel of flour or two sacks of beans strapped to his back, the animal has a hard time. Imagine a heavy piece of machinery, like a mortar box, or a boiler, being moved over such a road; then wonder how pieces weighing 6 and 8 tons have been transported.

Heavy loads such as mortar boxes or Huntington mills, are dragged with yokes of oxen by the use of heavy block and

* Mining Engineer, Chicago, Ill.

tackle. Progress of course is slow, but it is remarkable what can be moved in this way.

While it is possible to grow sufficient food for the miners on the soil of Nicaragua, it is only attempted at established mines. With an annual rainfall of from 300 to 125 inches, and a rich soil, the growth of vegetation in Nicaragua is rapid. One of the first things that is done at a new mine property is to clear the forest so as to plant bananas and plantains. Corn, sweet potatoes, yucca and sugar cane grow well, but on account of the ants many of the vegetables and fruits raised in the United States cannot be grown. To the mine owner in Nicaragua the production of food stuffs is most important, for it reduces his freight on imported articles.

Few countries in the world have a more liberal mining law than Nicaragua. There are no "extra lateral rights," or apex laws, but the law of vertical planes, similar to the law of Mexico, and the Transvaal, simplifies matters. The taxes on claims are probably the lowest in the world, being $2 per hectare (14,184 square yards) per year. There is no difficulty in acquiring mill sites or water-power to run the mills. Every facility is offered for the acquisition of the title. It is required that the owners keep the monuments of their claims in order, and also keep the boundary lines cleared. The mines are allowed to use for fuel or timber the wood in the surrounding forests.

The employer is required to provide for the treatment of his laborers in case of sickness or accident.

Fig. 2. Huntington Mill and Plates, Mars Mine, Pis Pis District

As regards labor, Nicaragua is far better off than some mining countries, Alaska for instance. Labor can be divided nto four classes; namely, Indian, Spaniard, American, Europeans, and Jamaicans.

The transportation of freight by canoes is practically in the hands of the Indians. For generations these natives of Nicaragua have gone about in these small boats, and it is doubtful if any race living could do this work as well. Taken away from the canoes, the Indians seem lost, and at other work are not satisfactory laborers. The wages paid the Indians vary from 40 cents to 80 cents per day, the employer supplying food and shelter.

The Spaniards or Nicaraguans are the miners of the country. Properly handled they do excellent work, being more satisfactory than the peons of Mexico. Fiêstas and "booze" account for most of the time lost by these miners. While they work fairly well on "day's pay," it is found more satisfactory to put them on contract. The wage per day for a Spaniard varies from 80 cents to $1. Miners working in adits and drives on contract can earn from $2 to $4 per day. Nearly all mining in Nicaragua is done by hand, there being no air drills. On the whole the rock is not very hard and breaks well, so that from 60 feet to 80 feet per month is advanced in a drift working double shift.

The expense of feeding the laborers varies at the different mines. Where all the food is imported, and there is a long land transportation from the river, the cost to feed a man per day will amount to $1. Under favorable conditions the cost is 55 cents. Compared with the figures of the gold mines of the Transvaal where the cost per day to feed a Kaffir is 6 cents, and a Chinaman 12 cents, one realizes what an item of cost the feeding of the laborers of the Nicaraguan mines is.

No systematic study has ever been made of the geology of Nicaragua. The engineer at once notices the absence of sedimentaries, and the vast extent of rock that he can designate "porphyry." Belts of highly crystalline limestone are found here and there. In Western Nicaragua, around the lake, near the volcanoes, the evidences of vulcanism are apparent, but in the eastern section of the country, in the Pis Pis district, the strata are not disturbed, nor are there evidences of andesite or any daloidal diabase "flows." If these "sheets" ever existed, no doubt the intense erosion of the country has carried them away.

In the Pis Pis district few dikes or large faults exist, and the outcrops of the large gold veins can be followed for miles. Most of the veins are replacements, with one wall well defined, generally the foot-wall. The veins dip from 65 degrees to 85 degrees.

The "wash" from the veins is sometimes rich in gold, and on the hillsides one frequently finds large pieces of ore which have weathered from the vein.

The evidences of intense denudations are apparent on every hand. The country rock is much effected by water action, as are the veins also. In the valleys, water level is from 50 feet to 100 feet below the surface, but on the hills not much water is encountered before the level of the valley is reached. It is probable that below the zone of oxidation will be a sudden change in the veins to sulphides.

The engineer is much surprised to find such low working costs in Nicaragua. There are cases where the cost per ton is $10 and over, but as a general rule the cost per ton milled in Nicaragua is from $2 to $3. When it is remembered that no mills in the country have a capacity of more than 100 tons per day, the reader realizes what a chance there is in this country for low working costs when the tonnage per day is raised to 300 or 500 tons. Few countries can then show lower mining costs.

The reason for low working costs in Nicaragua are that the veins are generally wide; water-power or hydroelectric power is available in most cases; and mine labor is fairly cheap and efficient. These advantages counterbalance the enormous transportation charges.

In nothing is the country more backward than in metallurgy. In some cases in the Pis Pis district the extraction of the gold is only about 33 per cent. At the Siempre Viva and Lone Star mines they have cyanide plants, and the extraction is about 70 per cent. At only one mine in the Pis Pis district is there a slime plant; namely, the Bonanza mine, where they have installed the clumsy inefficient decantation slime process. At the Bonanza the extraction is around 80 per cent., when with the average ore in the Pis Pis district the extraction should be from 85 per cent. to 90 per cent.

On account of the large quantity of alumina in the ore, the slime is very flocculent, and causes much annoyance, but judging from laboratory experiments the slime problem in Nicaragua can be solved.

After the ore passes through a crusher it is pulverized by stamps or Huntington mills. In the writer's opinion the best machine for Nicaraguan gold ores is the 5-foot Huntington mill, because so much of the ore is soft and sticky that it clogs the screens and sticks to the "shoes" of the stamps. Making the stamps heavier only augments the difficulty. With these ores fine grinding is not necessary as they are porous and the sand ideal for leaching. Crushing to a 20-mesh screen an extraction of 84 per cent. on the sand has been obtained. It will be seen, therefore, that tube mills will be required in the Pis Pis

district. Crushing in cyanide solution gave good results, and it would pay in most cases to do away with amalgamation and to crush the ores in a 25-per-cent. solution of sodium cyanide.

Often one finds very hard quartz in the vein, and if pieces of this material from 1 inch to 2 inches in size get into the Huntington mill the wear and tear on the machinery is heavy. Where the amount of this hard quartz is large the ore is first crushed in a Dodge or Blake crusher, then passed through rolls and broken to pea size. With ore of this size the Huntington mill is one of the best grinders in the world. It is not an impact crusher, but an attrition pulverizer.

The strengths of the cyanide solutions used vary from .2 to .3 per cent. It is impossible to use lime for neutralizing on account of the heavy cost of transportation; for which reason caustic soda is added on top of the sand in the tank; and spread uniformly over the charge. Ordinarily it would be added to the stock solution before it is pumped on to the sand.

The cyanide secret of high extraction of the gold in a sand tank is a close separation of the slime from the sand. In some countries, the Transvaal for instance, the presence of slime is not as detrimental as in other places. In the Pis Pis district this is an almost impalpable powder and a small percentage in the sand tanks prevents the extraction by hindering leaching. For getting rid of the slime, cone separators are used. At some mills the amount of sand caught in the tanks is 55 per cent. of the pulp from the mill, but as a rule it will be found best to catch only 50 per cent. in the tanks.

With the exception of the cyanide plant of the La Luz mine, the tanks of which are steel, the mills use wooden tanks, about 20 feet in diameter and 4 feet deep. These are imported from the United States, and must be carefully tarred to prevent the depredations of ants and to keep the wood from rotting.

The ordinary wooden zinc box is used with zinc shavings, for precipitation. Occasionally there is some trouble from copper, but this difficulty is overcome by the use of chloride of mercury. On account of the flocculent material which goes into the zinc boxes and settles on the shavings, the clean up is a very dirty one. Sulphuric acid is used in the "clean up;" but the transportation of acid in Nicaragua is not only very expensive, but very dangerous as well. There have been cases where the jolting and the heat has caused the vessels holding the acid to break, with disastrous effects on man and beast.

The cyanide gold bullion sent out of the country is rather base, not over 600 fine. For years the gold output of the mines has been entrusted to the Indians in the canoes, who carry it down the rivers to the coast ports. Strange to say, there has never yet been a loss, or a "hold up." The Indians will steal flour or sugar but they would not take the gold bullion.

The outlook for prosperity in Nicaragua was never better now that the United States is paying more attention to Central America. To develop the gold-mining industry, however, will be no easy task. There are many difficulties in the way, but the gold is there and if sufficient capital is put into the country and competent men are employed to develop the mines, Nicaragua will be known as a successful gold producer.

AMERICAN TIN MINING IN BOLIVIA

The annual report of the British legation at La Paz, on Bolivian trade, shows continued activity in tin mining, which furnishes the most important article of export. Shipments during the first 6 months of 1909 amounted to $6,767,000 (American currency). Among the operating companies the following is mentioned:

Andes Tin Co., an American concern, which has spent a great deal of money on the erection of a hydroelectric power plant. Its concentration plant is now in course of erection. Patches of very rich ore are found, and there is an abundance of water from the melting glaciers above the mines on the high peaks at an altitude of 18,000 feet. The mines themselves are at an altitude of 16,000 to 17,000 feet.

THE INTERNATIONAL CYANIDE TRUST

By Consul General Robert P. Skinner

To an inquirer who wished to procure cyanide of potassium and cyanide of sodium from outside manufacturers, I am only able to report that the one concern which at one time I had reason to believe was still independent is no longer in the market, and the trust, therefore, apparently is in complete control of the business. The formerly independent house referred to wrote quite recently: "We are not in a position to offer cyanide of potassium. We do not produce this article."

It is understood that the trust has obtained such general control of gas works waste in Germany and in other countries under long-term contracts as to render it practically impossible for competing concerns to be organized, and it is believed that the few not already in the trust who might produce cyanide of potassium are paid to refrain from doing so. Indeed, according to the published report of the Deutsche Gold und Silber-Scheideanstalt for 1909, a payment was made in the year covered of $121,201 "according to special agreement."

Some years ago, before the organization of the trust, cyanide was manufactured and was being sold with at least a fair profit for from 62 to 70 marks ($14.76 to $16.66) per 220 pounds. The price has been driven up from this level to 118 to 122 marks ($28.08 to $29.04) for cyanide of potassium and 122 to 128 marks ($29.04 to $30.46) for cyanide of sodium. Both of the materials named are indispensable in modern gold and silver reduction processes, the demand being particularly great in the United States, Mexico, and South Africa. It is believed that not a pound of either material is now shipped either from the German producing factories or from the associated concerns in other countries except under conditions dictated by the trust, which produces also many other chemical products, not all of which are controlled so absolutely as cyanide.

The published report of the Deutsche Gold und Silber-Scheideanstalt, which is both a manufacturing and holding company, states that it is the owner of the stock of the three American concerns as well as of eight other companies in Europe.

For the fiscal year ended March 31, 1909, the Deutsche Gold und Silber-Scheideanstalt paid out dividends amounting to 30 per cent., and in the succeeding year there was a distribution of 33 per cent., or $785,000. These enormous dividends were made possible after paying the "tantieme," or non-producers graft, "according to special agreement," amounting to $121,201, and a further "tantieme" to the board of supervising directors of $51,032. The report contains the statement that the American companies in which the parent concern was interested were obliged to reduce the scale of their operations, and could not show as good results as in previous years. This situation was considered to be merely temporary.

The exports from Germany of cyanide potassium and sodium, for the first 10 months of 1910, and for the full calendar years 1909 and 1908 were as follows:

Countries	1910 Tons	1909 Tons	1908 Tons
United States	1,098.3	1,349.2	1,394.8
Great Britain	117.4	147.0	148.7
Russia	199.9	140.3	217.5
South Africa	1,638.0	2,600.7	2,564.0
Japan	87.2	133.7	105.6
Dutch Indies	139.8	199.1	125.3
Mexico	1,065.6	1,118.1	19.1
Australia	110.2	216.1	77.9
All others	548.6	377.8	233.7
Total exports	5,005.0	6,282.0	4,886.6
Value	$1,548,190	$2,093,210	$1,628,150

MINING IRON UNDER THE SEA

*By H. W. Buker**

Method of Driving a Submarine Slope 7000 Feet Long At Wabana Iron Mine, Newfoundland

In Conception Bay, on the eastern coast of Newfoundland, lies Bell Island, the site of the Wabana iron mines, which during 1910 supplied 1,250,000 tons of ore to Canadian, European, and United States ports. It has now been established that the iron which outcrops on the island is only the fringe of great deposits which extend far under the bay. A submarine slope has been driven nearly 7,000 feet from high-water mark by the Nova Scotia Steel and Coal Co., and has shown that the ore improves in quality and thickness as it gets deeper. It is largely because of the proximity of these deposits to the coal fields of Cape Breton that the steel industry of Eastern Canada has grown to its present proportions, and the ore reserves which have now been proved to exist under the ocean, insure the prosperity of that industry for centuries to come. It is only 40 years since accident revealed the presence of the ore deposits, and not until 1893 was any attempt made to explore or develop them. Investigation was begun at that time by the Nova Scotia Steel and Coal Co., Ltd., which resulted in the lease of the deposit and its eventual purchase.

The north side of Bell Island contains a number of seams of red hematite iron ore, running from 50 to 55 per cent. in iron, three only of which are economically important at present. These seams or beds are interstratified with the sandstone and shales of the formation, with a common dip and strike, the former being about 8 degrees.

Early Development.—The work of the company, begun in 1895, at first was only open-cut mining, the earth covering being stripped and the ore then carried by an endless-rope tramway to a pier on the south side of the island. This pier was a block set out some distance from the shore and connected with it by a suspension bridge; a trestle work was built to it later.

FIG. 1, NEWFOUNDLAND, SHOWING LOCATION OF WABANA

At first the ore was used solely to supply the blast furnace of the Nova Scotia Steel Co., at Ferrona, but in 1899 the Dominion Iron and Steel Co. commenced operations at Wabana. This company purchased from the Scotia the lowest of the three parallel beds of ore which the latter had been operating, together with all their equipment. The Scotia company at once commenced to open up the middle, or Scotia, bed of ore in order to secure an uninterrupted supply. Twelve hundred men were kept at work that season, mining from the old property and developing the new, and when the time came for them to turn the Dominion property over, the Scotia bed was ready to produce all the ore required. Tramways had been constructed, a new pier built, and the Scotia went ahead producing ore without a break in deliveries.

The Dominion company secured the upper and lower of the three beds of ore and a block of submarine areas adjoining the shore and containing about 3 square miles. The land ore beds of the two concerns overlap, one company working in some cases directly underneath the other, but in the submarine areas each company owns all the ore in its holdings.

When building their new pier, the Scotia company took advantage of a great gulch near it to construct an immense storage pocket for ore. By very little work, this break was converted into a storage receptacle of much greater capacity than the one formerly used.

The Scotia Bed.—For 2 years the Scotia bed was "stripped," but in 1902 two slopes were begun. Work was carried on rapidly, and within a year these mines had been opened up and were being worked by the room-and-pillar method. Both slopes were sunk at a considerable height above tidewater, and one was driven out on the shore above tidewater mark, forming an adit. The slopes are carried from 8 to 10 feet in height, and 17 to 20 feet wide, in two sections, a 14-foot heading, and a side slice 6 feet wide. The sketch Fig. 3 indicates the method of drilling the holes in 8-foot ore. Another line of holes is put in when the ore is 10 feet high. The center cut holes are about 7 feet deep, and the rib holes 6 feet. All holes are bottomed with a diameter of 1⅝ inches, and 1½-inch 50-per-cent. dynamite is used for blasting.

The view of the drill at work underground, Fig. 4, and the picture of open cut mining, Fig. 2, indicate the marked vertical, or rhombohedral, cleavage of the deposit. This "blocky" ground causes trouble in drilling uppers, since the hole has a constant tendency to run out of line, requiring frequent changes in the position of the drill on the column. There is just enough moisture in the ore to make the cuttings ball up and pack behind the bit. The wet or down holes, however, are fairly easy to drill. Sullivan 3-inch differential valve piston drills are used, and 50 are now in operation. The Dominion Iron and Steel Co. employs 40 Sullivan drills, including a few 3¼-inch machines.

In the Scotia vein the drills are mounted on single screw columns without arms, but in the submarine slope and in the 14-foot bed of the Dominion company, the double screw column with arm is preferred. Tripods are used in open-cut work, and in driving raises or breakthroughs underground.

Air pressure for the drills is 80 pounds at the receivers, but at the ore faces is actually about 70 pounds, and in the submarine slope, now between 7,000 and 8,000 feet in length, the pressure falls to 60 pounds at the face.

Bonus System.—A runner of average ability puts in about 75 feet of holes per 9-hour shift. Good men have run up to 100 feet or more per shift. An interesting bonus system is employed by the Scotia company, and since it was established, in 1904, the efficiency of the drilling work has been materially increased. An account of this system was furnished *Engineering Contracting* in 1906 by Mr. A. R. Chambers, and reprinted in *Mine and Quarry* for November of that year. The following is quoted to show the results of the scheme:

"Comparing the drill sheet for October, 1904, with that for August, 1905, we see that each drill averaged 58.6 feet per day in October, 1904, as compared with 69 feet in August, 1905. This is an increase of nearly 18 per cent., but, excellent as this increase is, it is not so striking as the increase in the number of tons of ore broken down by each drill. In October, 1904, each drill averaged 26.2 tons per day; while in August, 1905, each drill averaged 44.5 tons per day—an increase of nearly 70 per cent. We note also that this remarkable increase in output was not obtained at the expense of more dynamite; for, in the 1904 sheet only .64 ton of ore was broken per pound of dynamite, as compared with 1.2 tons per pound in 1905." With reference to the decreased amount of dynamite per ton of ore, Mr. Chambers says that this has been brought about by requiring the drillers to pay more attention to placing the holes as accurately as possible—to give the dynamite the best leverage; that is, cut holes to meet within about 20 inches, and not overburdening side holes. "It is," he says, "also partly due to the fact that, whereas we used to run the drift 18 feet wide, with 12 holes, we now run it about 15 feet wide with 12 holes and side-cut to 18 feet wide with a couple of extra holes. The

*Reprinted from *Mine and Quarry.*

increased number of feet now drilled per day enables us to do this in the same time."

The following notes on the drill work in 1909 may prove of interest:

No. 1 Mine

Drill days	3,458
Total feet drilled	251,521
Average feet drilled per day per drill	72.7
Output, tons (long)	149,804
Tons per foot drilled	.5900
Tons per drill per day	43.3000
Drill repairs	$547.6600
Repairs per drill per day	.1500
Repairs per foot drilled	.0021

No. 2 Mine

Drill days	4,739
Total feet drilled	323,337
Feet drilled per day	68.20
Output	207,025.60
Tons per foot drilled	.6400
Tons per drill per day	43.6000
Drill repairs	$805.6600
Repairs per drill per day	.1700
Repairs per foot drilled	.0024

Before being sent underground, all drills are tested to determine the force of the blow, so that an efficiency rating can be assigned each machine. The testing machine is the invention of Mr. A. R. Chambers of the Scotia company. It consists of a Sullivan 3-inch drill, to the piston of which the piston of the drill under test is clamped. A system of pipes and gauges shows the relative efficiency of the new drill.

All drill steel is sharpened in a drill sharpening machine whose striking parts are made up of two Sullivan drills. This and the testing machine are located in the company's very complete machine shop, near the No. 2 mine.

Fig. 2. Open Cut Mining in the Lower Bed

The Submarine Slope—About 1905 the possibilities of developing the submarine areas began to attract the attention of the Scotia company. Further additions to its underwater holdings were secured, until it owned 35 square miles of submarine areas. At the same time the Dominion company increased its submarine holdings until it held 5 square miles. The Scotia company decided to drive a pair of slopes to the submarine areas, and, an arrangement having been entered into with the Dominion company by which the slopes were to be driven through the areas of the latter corporation, work was commenced in May, 1906. It was known that at distances of about 250 and 1,000 feet, respectively, serious faults would be encountered, so it was not considered advisable to make any expenditure for plant until these should have been passed successfully. The following description of the first 4,000 feet of this slope is abstracted from a paper presented by Messrs. R. E. and A. R. Chambers before the Canadian Mining Institute, early in 1909:

"In August the first fault was encountered. Fortunately, although this dislocation amounted to an upthrow of 29 feet with only 85 feet of strata intervening between the workings and bottom of the ocean, the volume of water encountered was trifling.

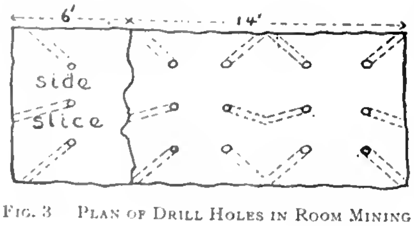


Fig. 3. Plan of Drill Holes in Room Mining

"From a study of the rock structure upon the land it was fairly evident that this upthrow would be followed by a corresponding downthrow running northeast-southwest, after driving about 800 feet approximately at right angles to the fault first encountered. It was consequently decided to continue the slope in the strata underlying the ore in order to maintain as uniform a grade as possible. The wisdom of this course was verified by encountering the ore beyond the second fault at the distance and height expected, and of greater thickness. This fault was more serious, containing 4 feet of clayey decomposed rock, and an increased flow of water. The soft ground was handled by careful spiling and the water could easily be handled by two No. 6 Cameron pumps. The quantity did not exceed 50 gallons per minute.

"Beyond this second fault no serious difficulties were encountered for a further distance of 1,800 feet, at which point a downthrow fault of 26 feet was reached. The cover had increased to a total of 293 feet, and in consequence a greater feeling of security was felt in providing against any difficulties which might occur. With the increased depth, however, a corresponding increase in firmness was found, and the clay encountered in previous faults was not met here. The slope was continued in the strata overlying the ore, but on an increased grade for a distance of 400 feet, when ore was again entered, having a normal thickness, and a normal dip of 12 per cent. The grade for the 400 feet was made 20 per cent., which by trimming will be subsequently reduced to 18 per cent. For this 400 feet the effects of the fault were noticed in the increased tenderness of the roof, necessitating timbers being placed skin to skin; and in the amount of water dripping from above. The total quantity of water was, however, in no way formidable and was easily handled.

"On entering the Scotia areas at a distance of nearly 4,000 feet from the shore the conditions were as favorable as could have been hoped, and more so in that the quantity of water handled was very small compared with what was expected.

"The advances up to December, 1910, were as follows:

	Feet
Slope commenced May 21, 1906	
Distance driven to December 31, 1906	704.60
Distance driven to December 30, 1907	1,916.10
Distance driven to December 24, 1908	3,965.00
Distance driven to December 30, 1909	5,400.40
Distance driven to December 17, 1910	6,815.62
(At a depth of 1,300 feet below the surface of the ocean)	
The month of least advance was August, 1907, in passing first fault, or	42.50
The month of greatest advance was November, 1908, or	247.81
In the 4 weeks from the 16th of November to the 14th of December, 1910, an advance was made of	270.10
In week ending January 23, 1909, an advance was made of	74.00

"This gives a record of 11¼ feet per day for a month. This gives a record of 12 feet 1 inch per day for a week. The size of excavation for this driving was 13 to 15 feet wide by 8 feet high.

"As the work proceeded it was ascertained by diamond drilling that the lower bed had increased very greatly in both thickness and richness. At the outcrop, this seam was about 11 feet thick, but it

increased gradually, and after passing the faults had more than doubled, while analysis of the ore showed that the iron content was much higher and the silicon correspondingly lower."

Method of Working.—Three 8-hour shifts are worked. When the air is cleared of smoke from the last shot, the drillers and loaders proceed to the face. The drillers and their helpers assemble the drills and columns, while the loaders throw back the ore from the face. The drills, of which there are two on this shift, are then set up 4 feet from the face, and about 20 inches from the rib or side, and each driller puts in five 12-foot center cut holes, B-1, 2, 3, 4, 5, and C-1, 2, 3, 4, 5, respectively (see Fig. 5), completing the round in about 6 hours. The holes are blown out with compressed air, when the shift is completed. After having cleared away the face for the drillers, the six loaders clear off the track, run in a trip of three cars and load up the ore thrown up the slope by the previous blast, which sometimes is 100 feet from the face. When the ore is reduced to a heap behind the drillers, one car only is used, and the loaders relieve each other, car about, three at a time. The loaders generally finish at the same time as the drillers and 40 to 50 tons are an average shift's work.

The blasters then load the holes drilled with 7 pounds of 50-per-cent. dynamite in each, tamping them securely with paper cartridges of clay prepared for the purpose. After the wires and blasting machinery are tested for strength and circuit, the round is fired. If the work has been done correctly, the center cut will be taken out as shown in the sketch, and the ore thrown well up the slope. Owing to the heavy burden on these holes they will be only about half shot out. A little high-pressure air is turned on at the face immediately after the blast, and the exhaust fans speeded up. In the course of a short while the air at the face is quite clear, and while still a little warm, work can be resumed.

FIG. 4. DRILL IN NO. 2 MINE. SHOWING ALSO THE BLOCKY NATURE OF ORE

The next shift then begins, loaders clearing face as before. The regular drillers now set their bars 4 feet back from the face and as close to the rib as possible, and drill holes A-1, 2, 3, 4, 5, and D-1, 2, 3, 4, 5, 12 feet deep. In addition to these, two spare drills are set up in the cut and drill holes E-1, 2, 3, and F-1, 2, 3, and G-1, 2, 3, H-1, 2, 3, respectively, 6 feet deep. As the extra drills do not come on the same shift each time, there are two spare drillers on each shift who, when not at this work, repair and advance ventilating pipes, swinging Sampsons, etc., and other odd jobs.

The order of blasting is now more elaborate than on the previous shift, and on the care taken in this work greatly depends the advance to be gained.

They first load and fire holes F-1, 2, 3, and G-1, 2, 3, 3 pounds to a hole. Next they fire the old bottoms of holes B-1, 2, 3, 4, 5, and C-1, 2, 3, 4, 5, with 3 to 5 pounds each, according to necessity. The third round is E-1, 2, 3, and H-1, 2, 3, 5 pounds to a hole, and finally A-1, 2, 3, 4, 5, and D-1, 2, 3, 4, 5, are fired, 5 pounds to a hole. This makes a total of about 200 pounds of 50-per-cent. dynamite to an advance of 8½ to 9 feet, according to the success of the work done.

The method of using short bars and setting up the drills on top of the ore heap was at first tried and practiced for some considerable time, but it was found that while saving a good deal of time in handling the ore, in the long run it did not pay.

The storage, thawing, and distribution of the explosives for both the mines and the submarine slope are carried on at a central station which comprises a small magazine, thawing house, with detached boiler room, and a shed for testing detonators, located at convenient distances apart.

The thawing is carried on in a small stone building, brick lined and wood faced inside, with an air space between the brick and stone, and the building is provided with a large entry for the reception and disposal of the explosive. It is heated by the ordinary hot-water system, but the radiators are closed in wooden boxes to prevent accidental contact with the explosives.

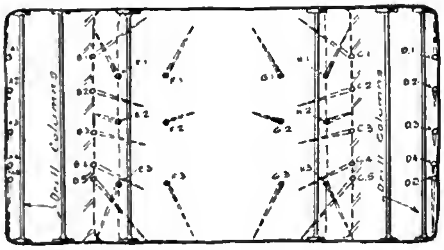
The various blasting crews, wiremen, thawers, magazine storekeeper, and detonator tester, are all under the supervision of an explosive inspector, who constantly inspects all operations connected with the use and disposal of all explosives, and who immediately reports to the underground manager any unusual occurrence, and, if possible, the cause of the same.

The Plant.—Sullivan rock drills are used in the slope. Compressed air at 80 pounds receiver pressure is supplied to the drills and pumps in both the slope and in the working mines by two cross-compound steam two-stage air compressors and one straight-line compound steam and two-stage compressor, having a total capacity of 7,200 cubic feet of free air per minute. This plant also drives small hoists underground and the fan engines.

The ventilating plant consists of one 4-foot and two 3-foot Sturtevant fans. The 4-foot fan is direct connected to a vertical air-driven fan engine, and is located near the upper part of the slopes, just at the bottom of the present workings of the

No. 2 mine, of which the slopes are a continuation. This fan is placed in the air-course, and is operated exhausting. The two 3-foot fans are placed as described below.

The intention was to sink the slope by double entry. In practice, however, it was found that the tendency was to place chief importance upon the slope farther advanced, with the result that it finally exceeded the secondary or ventilating slope by a distance of 2,900 feet. To provide for ventilating this long single slope, recourse was had to spiral riveted galvanized-iron pipe, connected with two blowers, with a suitable by-pass connection, so that they could be used either as blowers or exhausters. One of these was placed at the upper end of the pipe and the other, reinforcing it, about 1,000 feet further down. The first pipe used was 15 inches inside diameter, and was found to be too small to give entirely satisfactory results. Below the lower fan, 18-inch pipe was installed, which still continues to give a plentiful supply of air to the face of the slope. As the slope advances, new lengths of pipe are added. On account of the blasting, the pipe cannot be laid nearer than 150 feet of the face, but no inconvenience whatever is felt.



End Elevation

Dams.—To protect No. 2 mine from possible inundation, sites for mine dams were constructed at a point situated vertically below high-water mark. When approaching this point the face was narrowed to 6 feet and then gradually widened again, forming a **V**. No reduction was made in the height. The sides, top, and bottom were lined with concrete. A small pipe was placed in the roof as an outlet for air when the pressure should come on the lower side of the dam, and capacious air and water pipes were built into the floor. Timbers to fill the opening were made of 10″×10″ pine, tapering to 6″×6″ and piled at the mouth of the slope in the same position they would occupy in the dam and carefully protected from the weather.

Piping.—The air line is 8 inches in diameter as far as the workings in No. 2 mine and to the large Jeanesville pump, and 5 inches for the rest of the submarine slope. The water discharging line is 6 inches, being double from the large pump to the surface.

Testing Air Pipes.—The air pipes are periodically tested as follows: All pipe ends are carefully closed as near to the drills as possible, and a pressure gauge is connected to the section to be tested. The section is then filled from the mains, and the supply valve closed. The air in the line is then released until the gauge reads 50 pounds. The time is taken and the pressure allowed to drop by leakage until it reaches 40 pounds, when the time is again taken. The loss in cubic feet of free air will be approximately half the volume of the pipe tested. This divided by the time will give the loss per minute. As the amount of pipe in each section is not often changed this information is readily available. The conditions, with the exception of the time, will be practically constant. The pipe fitter is thus constantly familiar with the condition of his pipes. Corrections for temperature would be necessary for accurate results, but in practice are not needed.

Cars.—The cars used are of steel, of 30-cubic-feet capacity, holding from 1.7 to 1.8 tons of ore.

Tracks.—The tracks are of 28-pound rails, except near the face, where an 18-pound rail is used temporarily. The gauge is 2 feet.

Telephone System.—Telephones are installed in the slopes, connecting through a central office with all parts of the works, the office, and the houses of the manager and underground manager.

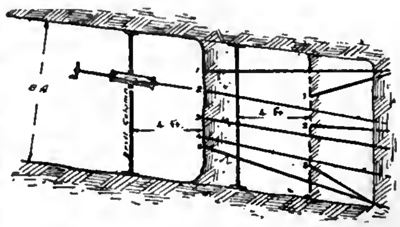
Efficient and modern electric pumps, fans, hoists, and other devices have been installed and the equipment of these mines is considered to be unsurpassed.

The writer acknowledges with thanks many courtesies and much aid in preparing this article to Mr. A. R. Chambers, manager of the Nova Scotia Steel and Coal Co., and to Mr. L. H. Johnston, manager of publicity for the company.

THE VALDEZ CREEK PLACER DISTRICT

*By Fred. H. Moffit**

Interest increased in the gold resources of the Copper and Susitna river basins and the adjacent coast region during the year 1910. Many prospectors in the vicinity of Valdez, Seward, and the head of Cook Inlet searched for lode deposits, but in the interior only placers received much attention. These interior placer districts include the Nizina River, Slate Creek, Valdez Creek, and Yentna River districts. The Valdez Creek district has attracted attention on account of the discovery of gold-bearing gravels that promise an extension of the known gold-producing area.



Side Elevation

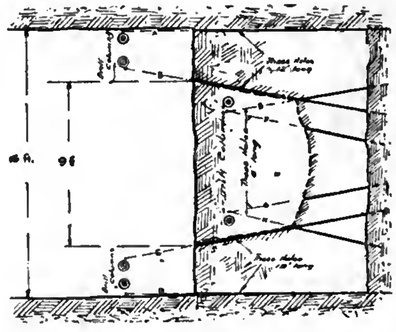


Fig. 5. Plan of Drill Holes in Submarine Slope

Valdez Creek is one of the upper tributaries of Susitna River, which flows into Cook Inlet. It is about 14 miles long and lies in the mountains flanking the Alaska Range, flowing into the Susitna from the east. Its broad valley is floored with gravels that are largely the product of glaciation. All its largest tributaries rise in the mountains south of Valdez Creek. They include Timberline Creek, White Creek, and Roosevelt Creek. Rusty Creek and Lucky Gulch are tributaries of White and Roosevelt creeks, respectively. The total gold production of the Valdez Creek district up to 1910 is approximately $260,000. The most productive part of Valdez Creek proper is in the lower 2 or 3 miles of its course and includes about 1 mile of the stream. In addition to the placers of the creek, Lucky Gulch, which is about 7 miles from the main Valdez Creek camp, produces gold, and prospecting on Rusty Creek has given good grounds for believing that that stream also will become a gold producer. During 1910 most of the mining on Valdez Creek was confined to four creek claims and to the Tammany bench claim. Mining began here in 1903, and at present the richest parts of the creek claims, so far as they are known, are nearly worked out. The Tammany bench claim, however, which is the richest property yet discovered in the region, continues to yield as it has in the past, and there is no reason to suspect that it will not do so for some time to come.

This part of Valdez Creek flows in a cañon cut in slate country rock and the overlying gravel deposits. The cañon

*Condensed from United States Geological Survey Report that will be ready for distribution about June 1.

Fig. 6. Scotia Pier and Power Plant

Fig. 7. Ore Shute to the Pier

has its greatest depth, 170 feet, in the vicinity of the so-called Monahan tunnel, which is the adit to the Tammany bench, just mentioned.

The creek claims on Valdez Creek that have received most attention are those known as Discovery claim and Nos. "1 below," "2 below," "2 above," and "3 above." No. "1 above" includes a deep cañon too narrow to permit mining. The richest of the productive claims were "Discovery claim," No. "1 below," and the lower half of No. "2 above." There is little variation in the character of the creek gravel. It ranges from 3 to 8 feet in thickness and consists of slate, schist, and granite, together with a small proportion of light colored porphyritic intrusives and dark basaltic and tuffaceous material. An important characteristic of the gravel deposits is the great proportion of large granite boulders. Such boulders are difficult to handle in the cuts and add much to the cost of mining. The gold of Valdez Creek differs considerably in appearance. Part of it is flat and smooth, but another part is rough and little worn. Much "ruby sand" and a little "black sand" is associated with it.

Fig. 8. Tramway to the Shipping Pier, Wabana Mines

AMETHYST COLORATION OF GLASS

Some time during 1910 a correspondent from Arizona sent in to us a piece of ordinary bottle glass. This glass he stated was originally white and had been colored while lying on the ground in Arizona. Old textbooks ascribed this kind of coloration to peroxidation due to manganese in the glass, although it might occur if the glass were in contact with manganese in the ground. Recent investigators have attributed the effect to the violet solar rays, that is, the radium rays.

Radium deoxidizes air and water, two of the most stable chemical combinations. J. H. Niemann has an article on "Amethyst Coloration of Crystal Glass" in the *Australian Mining Standard,* September 21, 1910. Some of his views agree with the correspondent's in Arizona, particularly that glass becomes colored in a few months in certain sections. He further states that when colored the glass darkens photo plates evenly all over; not in the speckled manner which is observed when radiographs are taken with monazite sand mixed with inactive tin grains.

The colored glass stimulates the growth of plants as powerfully as radio-active ores, owing to the decomposition of the air and the formation of probably a readily absorbed nitrogenous plant food. The glass also decomposes water. Water placed in an amethyst-colored tumbler bottom evaporates with twice the usual rapidity. Mr. Niemann has learned that glass becomes colored in the northern territory mineral bed of Australia in the Nyngan–Cobar district, New South Wales, in the Coolgardie and Kimberley districts, West Australia, and in one or two other districts which are visited regularly by frictional winds, after the said winds have traversed mineralized belts in which there is a strong probability that radioactive ores exist. To this list he may now add Arizona. By frictional wind he means winds that blow for some time in one direction, driving fine sand and radium emanation over objects on the surface of the earth.

In conclusion he states that if one has reason to suspect the presence of radio-active minerals, a sample should be taken from a few feet below the surface, since radium itself becomes liberated when it finishes its work of decomposition on surface material; also that it is only waste of time to use pitchblende as a guide when searching for the precious elements in Australia.

Sumner S. Smith, mining engineer in charge of United States mine rescue car No. 4, with headquarters at Rock Springs, Wyo., gives the following itinerary covering the movements of the car for the next few months: April will cover Scofield, Winterquarters, and Clear Creek. During May the car will visit Castle Gate, Kenilworth, and Sunnyside; and in June be at Hiawatha and Mohrland, all in Utah. The car will stop a week at each town, except at places where there is more than one mine, in which case each mine will have the benefit of a week's training. All mail intended for the car should be addressed, unless its station is known, to Rock Springs, Wyo., whence it will be forwarded

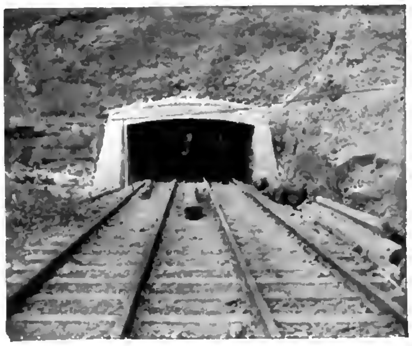


Fig. 9. Entrance to Submarine Slope

Fig 10. Head Buildings at the Slope

IRON ORES OF THE SOUTHWEST

*By C. Colcock Jones, M. E., Los Angeles**

Large Bodies of Easily Mined High Quality Ores—Smelting Methods With Oil and Electricity

It has long been known that in California, and more especially in Southern California, there were large and commercially valuable iron ore deposits, which, considered in connection with other known large deposits in the adjoining states of Nevada and Arizona, and in Lower California and other parts of Mexico, afforded a source of supply for one or more steel and iron plants large enough to supply local Pacific Coast demands—provided there were a solution to the fuel problem; and that the growth of population and other economic questions justified the necessary large investment of capital.

It is my purpose to show briefly in this paper that ample tonnage has been actually developed along the lines of the several railroads radiating from Los Angeles, which can be delivered at tidewater at a reasonable freight rate, that the enormous tonnages known to exist in Lower California are largely tributary to this port, and that as a center of several oil fields cheap fuel oil is assured. The stumbling block up to the present time has been the absence of a local supply of coal and coke and prohibitive price of the imported article; but the importance of the fuel question dwindles in the light of recent advances in iron smelting.

A word as to the geology of the ores in order to bear out certain assertions to be made hereafter. Geologically, the ores of the Mojave and Colorado desert regions and of Lower California belong to a well-recognized type of igneous contact ores, which are best described by quoting from recent bulletins of the United States Geological Survey—Mr. Charles K. Leith in his report on the iron ores of Southern Utah (Bulletin No. 338) says: "It early became apparent to me as it has been apparent to others, that the iron deposits of the West are prevailingly of a distinct and uniform type—an irregular replacement of limestone near the contact with igneous rock—a type fundamentally different from that of the important producing districts east of the Mississippi River and probably of different origin."

Also, from Bulletin 394 on Iron Ores of the United States, by C. W. Hayes: "This group of igneous contact ores is based exclusively upon its geological relations, and the deposits are widely distributed, though the most of them are located in the Rocky Mountains and Pacific States. The essential characteristics are steeply dipping lens-shaped bodies which closely follow the contact of an intrusive igneous mass and an intruded limestone. They occur partly within an igneous rock as dike-like veins, and partly within the limestone as replacements, but generally at the immediate contact. The limestone is always altered for a variable distance, sometimes several hundred feet from the contact. These ores appear to be due to ascending heated waters and vapors given off by the cooling igneous rocks. The ores include both magnetite and hematite."

Work on my own ore bodies and examinations of many other iron deposits of this region, confirms the essential characteristics so concisely stated in the foregoing extracts.

This particular form of ore occurrence has resulted in much greater surface exposures than are found in the Lake Superior iron regions, and in places where erosive action has been favorable, faces of ore are shown in intersecting gulches, several hundred feet in height and of as great or greater width.

Under such conditions the calculation of a definite tonnage is greatly simplified and the cost of mining, or rather in many instances simple quarrying, is reduced to its lowest terms.

Throughout the desert regions of California and Western Nevada and Arizona the intrusion of limestone beds by igneous rocks is one of the most common conditions, and there is scarcely a desert range of this nature that does not show some ledges of iron or float ore, and I venture the assertion that when industrial conditions and the demands justify it, systematic prospecting and mining will uncover as great bodies of ore with like tonnages as have been already exposed through the accidents of erosion.

For many years large deposits of iron ore were known to exist in San Bernardino and Riverside counties, but until the building of the Salt Lake Railway and the Tonopah & Tidewater Railway, all of them were too far from transportation to receive serious attention. These deposits are capable of being connected with the main lines of the Southern Pacific, the Santa Fe, the Salt Lake, and the Tonopah & Tidewater railroads by branch lines of from 1 to 50 miles in length.

Within the past few years a number of the known deposits have been patented and a large part of the tonnage is now owned or controlled by strong interests identified with transportation or manufacturing, which only await the successful outcome of certain experiments to actually engage in iron smelting.

These ore bodies have been purchased or taken up with an eye to that time when the economic conditions and growth of population will render the establishment of an iron or steel plant feasible, granting that the fuel question can be satisfactorily solved.

The various deposits lie within limits of from 150 to 300 miles of the port of Los Angeles. At a reasonable western freight rate in the inception of the business, say 5 mills per ton per mile for main line traffic, they come within the freight limits of 75 cents to $1.50 per ton. Add to that the cost of branch lines, mining costs and a fair profit, these ores can be landed at Los Angeles at not to exceed $3.50 to $4 per ton, and I have had offers for like ores from Lower California, Mexico, at this same price on dock at Los Angeles.

When it is realized that these ores run from 4 to 10 per cent. higher than the standard ore of Lake Superior in metallic content, that is from 60 to 66 per cent. iron, and carry practically no moisture to pay freight on, the low price of $3.50 to $4 per ton at San Pedro counteracts in a measure the higher price of fuel.

The two principal owners of iron ore in Southern California are: First, the Union Oil Co. through its subsidiary iron and steel manufacturing branch—the California Industrial Co.; and, second, strong interests said to be allied with the Southern Pacific Railroad, operating under the name of the Iron Chief Co.

The former of these companies has for many years been systematically acquiring iron properties in the southern part of California and in Lower California, Mexico, until its aggregate proven tonnage now amounts, I am informed, to 300,000,000 tons, about one-third of which is in California and two-thirds in Lower California, Mexico; and for several years past this company has been experimenting with fuel oil and gaseous fuel for smelting, preparing for the time when conditions shall reach a stage to render the smelting of iron profitable in Southern California.

The Eagle Mountain mines in Riverside County, 45 miles northeast of Mecca, a station on the Southern Pacific Railroad, 141 miles east of Los Angeles, contain probably the largest known deposit of iron ore in California, with the possible exception of the Minaret claims in Madera County. In 1908 and 1909 these deposits which had originally been patented by the Colorado Fuel and Iron Co. and several individuals, were acquired by the Iron Chief Mining Co. The deposit conforms to the type heretofore described, and is physically so situated as to definitely show 30,000,000 tons of proven ore capable of being cheaply mined, the average analysis of which is 64 per cent. iron, with phosphorus within the Bessemer limit.

There are a number of smaller deposits known in Riverside County on which the tonnage is a matter of conjecture, but with the establishment of an iron plant at tidewater it is just these smaller deposits held by individuals that would go far

* Read before the American Mining Congress, September, 1910.

toward developing a tonnage and opening deposits not now known, providing the usual economic policy were pursued of buying ores from various sources for a furnace mixture.

The only other deposit in California approximating in size the Eagle Mountain deposit is that of the Minarets in the high sierra of Madera County. At present this is almost inaccessible, but a branch railroad of 80 miles would make it equally available to San Francisco Bay or Southern California.

On the Santa Fe Railway, the California Industrial Co. owns large deposits 14 miles south of Newberry and 162 miles east of Los Angeles, and there are other known deposits north of Barstow.

On the Salt Lake Railroad, the Iron Chief interests have acquired the mines 1 mile north of Scott's siding, 190 miles east of Los Angeles, primarily as soft ore for mixture with the Eagle Mountain ores. A large amount of development work has been done, and an actual tonnage of 10,000,000 tons blocked out, with, conservatively estimated, three times that quantity as the total resources of the property.

Within a few miles of this deposit are other large iron outcrops.

On this same railroad 9 miles south of Kelso and 236 miles east of Los Angeles, in the Providence Mountains, I have proved up a body of 5,000,000 tons of soft hematite ore, Bessemer quality, capable of being quarried and loaded with steam shovels. This occurrence is one illustration of the large exposed outcrops so prevalent with these Pacific Coast ores. There is a quarrying face of absolutely clean ore of a 64-per-cent. grade, 350 feet wide and 250 feet high extending for a length of 800 feet.

On the Tonopah & Tidewater Railway 230 miles east of Los Angeles and 12 miles west of Silver Lake station the Colorado Fuel and Iron Co. owns one of the most extensive deposits in the state showing over 13,000,000 tons of exposed ore capable of being quarried and loaded with steam shovels.

There are tributary to all the railroads mentioned various other deposits of known merit, but enough has been said to indicate that, contrary to certain popular notions, the iron ores of Southern California are not controlled by any one set of men, and as soon as conditions are ripe and the capital can be secured a smelting industry will be established.

After carefully considering all the data based on my own observations and upon the figures supplied me relating to the two larger interests and by others, my conclusion is that, conservatively estimated, there are 200,000,000 tons of available high-grade iron ore, and double that amount of probable ore, in Southern California; and of this tonnage about three-fifths is owned and controlled by the combined Iron Chief and Union Oil interests, the other two-fifths being divided among individual holders.

In addition to this at the lowest estimate, an equal amount can be counted on from Lower California, Mexico, which will naturally seek an outlet to the northward, and the ownership of which by various individuals and corporations precludes more than any other factor the monoply of the iron ores of this district by interests adverse to their utilization, on account of cheap water transportation.

Several years ago an eminent Swedish geologist undertook to classify the world's supply of iron ore and his general deduction as to America was so pessimistic and out of keeping with the known facts that a more accurate estimate was undertaken by Mr. C. W. Hayes, of the United States Geological Survey in 1909, in which he credits the Pacific slope, combining Washington, Oregon, and California, with total available supplies of iron ore, 68,950,000 tons, and not available 23,905,000 tons. In the course of time it is to be expected that the Survey will get the actual facts and California will receive credit for something like the conservative estimate I have given.

Up to within very recently the smelting of iron ore has depended entirely on an available supply of cheap coke, the lack of which has prevented the development of the smelting industry on the Pacific Coast. But within the past year the Noble Electric Steel Co. at San Francisco has solved at its plant in Shasta County more perfectly than had previously been done in the world, the problem of making commercial pig iron with the electric furnace, and the process is a success.

In the case of this plant the close association of the iron ore bodies, the water-power for the production of electricity, and forests for the manufacture of charcoal, have, perhaps, furnished ideal conditions; but so far as Southern California is concerned the enormous recent development of fuel oil renders it practicable, when consideration is taken of the valuable by-product in the form of coke—both for use in the electric plant and the manufacture of electrodes—to effect an equalization between the cost of production of electricity by water-power with costly transmission lines, and from a compact plant producing the necessary power by the use of fuel oil either by steam or through the gas engine.

Los Angeles also has the added advantage that if from the development of large volumes of electricity in the building of the aqueduct there is any surplus for sale it can be more profitably utilized for the benefit of the community in general in such an industry as iron smelting than otherwise, providing it can be sold at a price low enough to warrant its use.

My own investigations have been along the line of a combination method, making a preliminary reduction of the ore by means of fuel oil or gaseous fuel and finishing up the operation in the electric furnace. The development of such a process promises more for the utilization of California ores than any other, for it is problematic if the coal or coke from the Washington or Alaska fields could be delivered at a sufficiently cheap price to warrant their use in smelting operations in Southern California.

As the matter now stands there is ample tonnage to justify the establishment of a plant at Los Angeles of a size in the beginning commensurate with the local demand; and for the moderate amount of solid fuel needed a high-grade coke product can be made by Professor Lowe's method from the refuse now going to waste in gas plants or that would be produced in making electricity on a large scale through gas engines as a motive power.

Pig iron is worth today on the Pacific Coast from $23 to $25 per ton, and we have the anomaly of importations of pig iron from China in competition with iron from the eastern United States.

The most ambitious project on the coast today is the plant of the Western Steel Corporation, now building on Puget Sound near Seattle. This company, in order to obtain a certain amount of pig iron from China, also contracts to use a large amount of Chinese iron ore. This, to owners of iron ore in Southern California, is conservation with a vengeance, for with any kind of interest on the part of the railroads in making extensions and furnishing dock facilities this tonnage could be shipped from Southern California on a sufficient margin of profit. The preferable plan would be, of course, to supply our own needs in pig iron first, utilizing our ores at home. There is at present a market for 100 tons of pig iron per day in this vicinity, and a trade in steel castings from electric steel could be built up to large and profitable proportions, without going into the manufacture of the heavier shapes and rails in the beginning.

Using fuel oil for the production of power, pig iron should be produced here electrically at from $16 to $18 per ton, and I am in hope that in the very near future our capitalists can be so thoroughly shown and convinced of profit in the business that the needed capital will be forthcoming.

In conclusion, a few words on the question of conservation. It has been suggested that among our other mineral products iron ore needs conserving for future generations. This I take to be the most extreme of all ideas in connection with this question. If there be any one product of the earth which, when

extracted from its ore becomes an absolute asset in the world in a finished state, it is that of iron.

The proper policy to pursue, particularly with regard to iron, as far as we Americans are concerned, is to manufacture all we can, as fast as we can, and as long as we can; supplying all our needs and the needs of the rest of the world so far as possible. This accomplishes not only the object of putting this metal in permanent form to be used and reused in commerce, with but a small fraction of a loss annually, but should such an unlooked-for thing as the exhaustion of our own iron deposits take place, we are even then in the best position in having all of the equipment for cheaply manufacturing iron ores brought from all other parts of the world.

I doubt if many men in this audience know of or remember one of the first acts that was done in good faith by the United States Government for the benefit of posterity. I speak of the time when our clipper ships and ships of the line, all wooden-bottomed vessels, were the pride of the nation and the envy of the world. In order that there might be no shortage for naval purposes, the Government set aside, removed from entry and reserved for future needs, large forests of the finest live-oak known, and stored up in this form and in the navy yards the pick of the oak lumber; in order to preserve the supremacy of our navy and merchant marine.

It seems almost laughable in view of the progress of the world, and no doubt many things now advocated in the name of conservation will seem as absurd to future generations as this live-oak conservation.

TRADE NOTICES

The Cananea Consolidated Copper Co., Cananea, Sonora, Mex., has just placed an order with the Westinghouse Electric and Mfg. Co., Pittsburg, Pa., for nine motors to be operated on three-phase, 60-cycle, 440-volt circuits. The order also included three single-phase transformers, and three three-phase transformers.

The Bureau of Steam Engineering, United States Navy, has recently decided to use the "Beard-Deputy-Marsaut" safety lamp, fitted with gas indicator, for use in the coal bunkers of their battleships and also around oil storage tanks. The American Safety Lamp and Mine Supply Co. has recently received a large order from the United States Navy for the equipment of the Atlantic and the Pacific fleets with this lamp.

Link-Belt Co. has announced a reduction in the price of original Ewart link-belt.

For many years, engineers have experienced difficulty in proper maintenance of brick settings of furnaces of all types. Ordinary fireclay has been commonly used as a bonding material, and as this fuses at a comparatively low temperature, the bond between the bricks is rapidly destroyed and cracks are opened up between the bricks, through which the gases of combustion enter, eventually weakening the brick and causing the walls to collapse. Another feature which is objectionable is that surfaces of refractory bricks become porous, the clinkers from the coals attach to them, and in removing these with tools, the bricks are broken. After several years research work on refractory cements to overcome these troubles, the H. W. Johns-Manville Co., New York. are now offering a line of cements called J-M Refractory Cements for furnace setting of various types, cupolas, lining brass furnaces, assayer's crucibles, oil burning, tilting and rotary furnaces, and for patching and facing bricks in place in the fire-zone under various conditions. These cements are rated to resist temperatures as high as 3,000° F. They also have produced a coating for walls, known as J-M Brickline Cement, which prevents clinkers from adhering and seals the pores of the brick. Realizing that the conditions under which these cements are used are variable, they have been made capable of modification to meet conditions as they come up, and the Johns-Manville Co. invite the trade to place before them any conditions that are troublesome and they will make a careful investigation and offer suggestions which will tend to overcome the difficulty.

The Scranton Acetylene Lamp Co. has discontinued its sales agency, formerly conducted by Francis H. Coffin & Co., and will hereafter conduct all departments of its fast growing business at the home office of the company, 144 Belmont Terrace, Scranton, Pa.

In the February and March advertisements of the Crawford & McCrimmon Co., through the accidental omission of a period and a capital letter, the sentence: "We protect the iron body with a hardened lead composition cast in." was run together with one following it, which should have been, "The wearing parts such as cylinder lining, rod seats, etc., the best quality of Phosphor Bronze under the sun." Of course it would have been evident to any one having experience with pumps, that the lead linings were only for the parts especially exposed to the action of acid waters, while the wearing parts were of bronze.

NEW INVENTIONS

Complete specifications and drawings for any of the following patents can be obtained from the COMMISSIONER OF PATENTS, WASHINGTON, D. C., at the following rates.

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
An entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED FEBRUARY 7 TO FEBRUARY 28, 1911, INCLUSIVE

No. 983,283. Coal and slate separator, Clarence Walter Honabach, Edwardsville, Pa.

No. 983,584. Coal elevator, Max C. Volk, Chicago, Ill.

No. 983,735. Slime filter, Elvin H. Lutz, Hay Springs, Neb., and Edward J. Sheda, Denver, Colo.

No. 983,363. Pneumatic mining machine, Martin Hardsocg, Ottumwa, Iowa.

No. 983,621. Automatic ore feeder, Charles T. Hutchinson, Oakland, Cal.

No. 983,616. Ore jig, Henry Foust, Galena, Kans.

No. 983,453. Method of treating ores by the blast-furnace process, Fredrik Adolf Kjellin, Stockholm, Sweden.

No. 984,406. Soft-coal crusher, John H. Wiestner, Philadelphia, Pa.

No. 984,498. Method of treating corrosive gaseous fumes or smoke, Clarence B. Sprague, Salt Lake City, Utah.

No. 983,881. Magnetic separator, Alvin Dings and Myron Dings, Milwaukee, Wis.

No. 984,355. Safety device for mines, Peter Darabos, Oakdale, Pa.

No. 984,222. Ore classifier, Frank G. Janney, Salt Lake City, Utah.

No. 984,090. Treatment of gold-bearing antimony ores, John Jones and Horace Sandford Bohm, Shandon Hill, Mount Morgan, Queensland, Australia.

No. 984,171. Rock drill, Daniel Shaw Waugh, Denver, Colo.

No. 984,326. Apparatus for disposal of waste rock from concentrating mills, Henry R. Wahl, Elvins, Mo.

No. 985,053. Apparatus for distilling shale and other bituminous substances, James Noad, East Ham, England.

No. 984,866. Aero ore concentrator and placer machine, Earl H. Tate, Los Angeles, Cal.

No. 984,567. Table adjustment for ore concentrators, Henry F. Jurs, Denver, Colo.

No. 984,633. Process of and apparatus for concentrating metalliferous ores, Henry E. Wood, Denver, Colo.

No. 985,266. Leveling apparatus for horizontal coke-ovens, Friedrich Aus der Mark, Sterkrade, Germany.

No. 985,660. Miner's cap and lamp, Lewis W. Cogswell, Taylorville, Ill.

No. 985,611. Machine for separating solutions from crushed ores, Edward P. Lynch, Los Angeles, Cal.

No. 985,447. Apparatus for separating precious metals from associated materials, Frederick H. Prentiss, San Francisco, Cal.

Mines and Minerals

Vol. XXXI—No. 10. SCRANTON, PA.—MAY, 1911—DENVER, COLO. Price, 25 Cents

PRODUCTION OF IRON ORE IN SPAIN

*Written for Mines and Minerals, by Harry A. McBride**

Extent of the Deposits. Analyses of the Different Ores. Methods of Mining and Handling

Although Spain is a great producer of iron ore, the industry seems to be stationary, the output in 1909 being about the same as in 1899, while the other leading producers have been increasing their output by rapid strides.

Table 1 shows the production of iron ore in the principal countries in 1909 and the exports therefrom during the same year.

The failure of Spain to keep pace with the other nations in iron-ore production is due largely to the growing scarcity of ore in the Biscayan iron zone, from which more than one-third of Spain's entire production comes, and to the non-exploitation of mines in more recently discovered ore districts, owing in many cases to the want of transportation facilities.

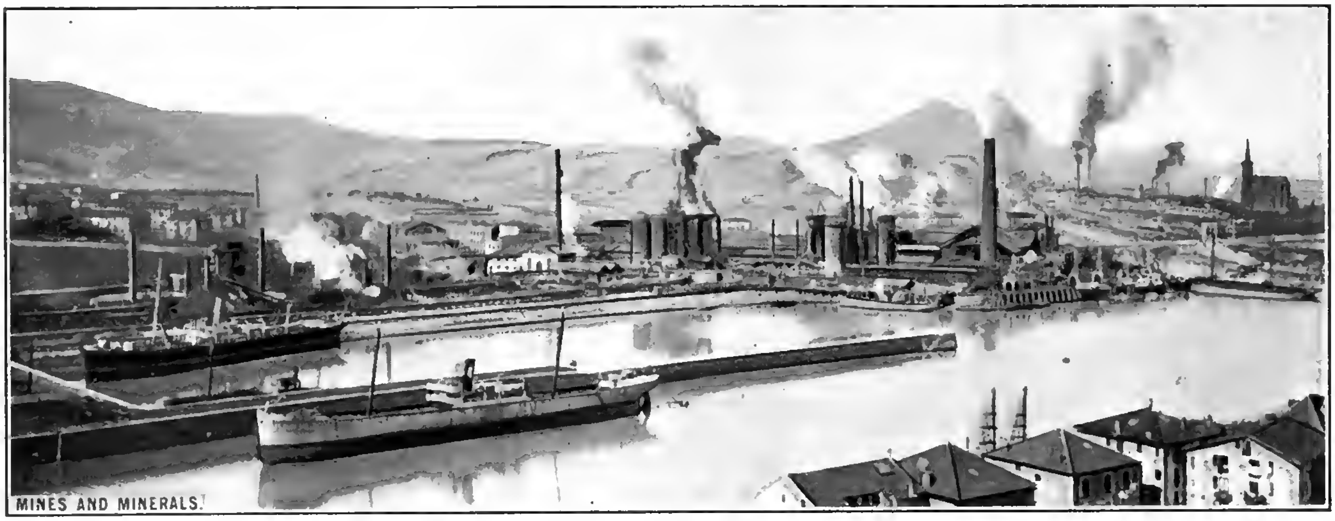


Fig. 1. Los Altos Hornos Steel Smelting Works, Bilbao, Spain

Table 1. Iron-Ore Production, 1909

Countries	Production Tons	Exports Tons
United States	53,882,415	401,567
Germany	27,500,000	4,250,000
Great Britain	15,450,000	
France	12,325,000	3,907,000
Spain	9,384,634	8,544,634
Russia	6,550,000	750,000
Sweden and Norway	4,093,725	3,387,943
Newfoundland	1,110,000	1,100,000
Total	130,295,774	22,341,144

In 1909, in the Province of Biscay, 3,871,927 tons of ore were mined, of which 3,297,414 tons were exported, 44,138 tons were consumed in home ports, and 530,375 tons by local iron and steel manufactories. The provinces of Guipuzcoa and Navarre export yearly 20,000 to 30,000 tons of ore by rail to France, through the station of Irun, and about 75,000 tons by water to England and Germany from the port of Pasajes. From the Province of Santander, via Santander and Castro-Uridales, 800,000 tons are exported annually, the greater part of which contains slightly more phosphorus than the Biscayan and Guipuzcoan ores, which are from .02 to .07 per cent. phosphoric.

Farther to the west, along the northern coast, are the ports of Rivadesella, San Esteban, Ribadeo, Vivero, and Corunna, whose exports amount annually to 400,000 tons of ore, containing from .75 to 1.50 per cent. of phosphorus. Practically all the ore from southern Spain, 3,500,000 tons, is non-phosphoric, and thus adaptable for the manufacture of Bessemer steel.

The mines most recently opened are located in the Province of Teruel, where a railroad 127 miles in length was completed in 1907 to afford transportation for ore from the mines at Ojosnegros to the port of Sagunto. The production of these mines in the year 1909 was over 400,000 tons, which has greatly increased during the present year. The extraction will, it is thought, reach 1,000,000 tons in 1911.

It is to these new mines that Spain must look if she would retain her present place as an iron-ore producer. It is hoped that recently discovered fields in the provinces of Lugo, Oviedo, Leon, Teruel, and Seville will soon be opened up and, through them, within 4 or 5 years, a new annual supply of 1,500,000 to 2,000,000 tons will be obtained; but it is also calculated that during this time the present output will diminish in about the same ratio, so that the total production of the country in 5 years' time will be very little greater than at present. Judging from the reserve supply, as yet untouched in many provinces, the Spanish output will be able to retain its present place for many years.

The following statement, based upon recently published reports of eminent engineers, shows the known deposits of iron ore in the Iberian Peninsula, in tons: Biscay, 70,000,000; Guipuzcoa, and Navarre, 10,000,000; Santander, 25,000,000; Oviedo, 55,000,000; Lugo, 65,000,000; Leon, 150,000,000; Estremadura, Segovia, etc., 10,000,000; Huelva, not counting sulphurous deposits, 10,000,000; Seville, 35,000,000; Malaga and Granada, 25,000,000; Almeria, 25,000,000; Murcia, 15,000,000; Teruel and Guadalajara, 135,000,000; Ciudad Real, 10,000,000;

*Consular Agent, Bilbao, Spain.

Catalonia, 10,000,000; Aragon, 40,000,000; Logroño, Burgos, Soria, etc., 20,000,000; deposits in Portugal, 15,000,000; total deposits in Spain, 710,000,000; deposits in Portugal, 15,000,000; grand total in the Iberian Peninsula, 725,000,000 tons.

If the present rate of export of Biscayan ore continues, it is calculated that the reserve supply will become exhausted in about 18 years, but as the exports tend to decrease the supply will probably last at least 30 years. Earnest efforts are now being made to bring the people of this province to realize that in a few years their greatest industry will have ceased to exist, and to induce the establishment of manufacturing plants and other business, in order that the now important city of Bilbao may better withstand the impending blow to its commerce.

Bilbao, the capital of Biscay, is the chief center of exportation. Over 3,000,000 tons are shipped annually from this port to England and Germany via Rotterdam. The other noteworthy ports of shipment are Santander, Seville, and Almeria. Many cargoes of ore are now shipped to Philadelphia from Sagunto. This movement will approach 200,000 tons in 1910. annually in the kingdom, by far the greater part is unsuited for smelting furnaces.

Bilbao is today the center of the smelting industry in the country. There are two large establishments under the same management located here. Their seven large furnaces at Los Altos Hornos, shown in Fig. 1, are of the latest types and the rolling mills are equipped with modern machinery. The consumption of iron ore in Spain in 1909 is estimated at 840,000 tons, of which 530,375 tons were consumed in the Bilbao works. The output of these latter works in 1909 was as follows, in tons: Pig iron, 255,627; Bessemer steel, 148,345; Siemens steel, 60,225; steel plates, etc., 149,866; tin, 8,015; coke, 215,627.

Products of these works are shipped to all parts of Europe, America, Asia, and Africa, and recently large contracts for steel rails were filled for Turkey and Finland. At present they are manufacturing material for the new Spanish battle ships, now building at Ferrol. Some of the ore for these battle ships came from the mines of Galdames, shown in Fig. 3, which are in the province of Biscay.

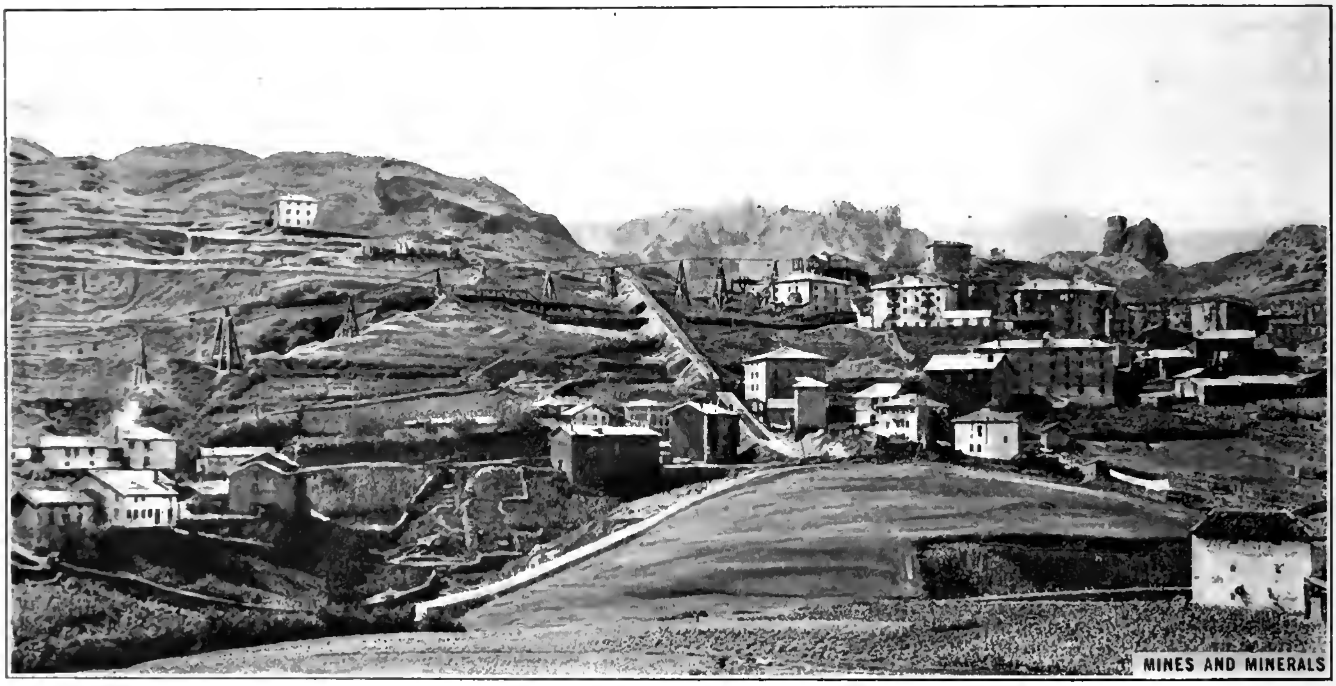


Fig. 2. Triano Mines, Near Bilbao, Spain

The average price per ton for Biscayan ores f. o. b. Bilbao, since 1906, with their average percentages of iron, is given in Table 2.

Table 2

Description and Year	Calcined Superior Grade	Calcined Second Grade	"Rubio" Superior Grade	"Rubio" Medium Grade	"Rubio" Inferior Grade
Per cent. of iron in crude ore	56	52	50	48	46
Per cent. of iron in dried ore	57	53	55	52	50
Prices:					
1906	$3.76	$3.27	$3.52	$3.15	$2.42-$2.67
1907	4.24	3.76	4.00	3.40	2.91- 3.15
1908	3.15	2.67	2.79	2.42	2.18- 2.30
1909	3.76	3.40	3.64	3.40	2.67- 3.15
1910	3.88	3.52	3.76	3.52	2.79- 3.27

In comparison with the position held as a producer and exporter of iron ore, Spain takes very low rank as a manufacturer of iron and steel products. This is largely due to the excessive cost of fuel, most of which has to be imported from England. Spanish coal from the Asturian mines is used to some extent, but of the 4,000,000 tons which is produced

Spain has several districts that are unusually rich in iron ores of high quality. Up to the present time, Bilbao has turned out the most ore; and it is estimated that the total quantity of unworked ore in the several districts amounts to 710,000,000 tons. The ores of the south of Spain are virtually untouched, although, so far as can be ascertained, they are of at least as high quality, and probably can be worked fully as cheaply, as those of the north. Between Malaga and Cartagena there are some large and easily worked deposits of high-class ores, a few of them running up to as much as 65 per cent. iron, and at least one or two of them within 20 miles of the coast. The ores of the south of Spain are well adapted for the Bessemer and open-hearth processes and it is a general characteristic of these ores that they contain considerably more manganese than those found in the north, reaching up to as much as $4\frac{1}{2}$ per cent., and averaging in some districts $3\frac{1}{2}$ to 4 per cent.; while they exist both as hematite and as magnetites, the former occasionally running more or less into spathic ore. It has been found that ores of this class are admirably adapted for mixing with other high-class ores, such as the hematites of West Cumberland and Lake Superior.

The rocks associated with the Bilbao iron ore belong to the Cretaceous formation. Blue limestone and shales are found above it, and limestone and schistose grit below it. The latter is recognized as the floor of all the deposits. The ores are classed as follows: Rubio, a brown hematite or hydrated ferric oxide ($Fe_2O_3+H_2O$), which forms the great bulk of all now produced; campanil, a red hematite or ferric oxide (Fe_2O_3), containing a somewhat less quantity of water in combination; vena dulce, also a hydrated ferric oxide (Fe_2O_3), found in veins, of a soft and porous nature, and more or less intimately mixed with both of the above kinds; spathic ore, siderite or ferrous carbonate ($FeCO_3$). The upper portion of the deposits generally consists of rubio. It is full of cavities, some of which contain earth and clay; and, consequently, it requires more careful selection than the other kinds. It has always a honey-combed appearance. Brown hematite and spathic ore are sometimes found mixed in broad bands, locally called *pedrisco*.

Rubio contains on an average about 50 per cent. metallic iron, as delivered in cargoes, and campanil a little less. Down to a certain depth in the mines the quality seems to improve. Reduced metallic yield generally arises from want of care in selection, or from wet weather at the mines, when the mineral is so coated with muddy water that it is difficult to detect impurities. In Table 3 are average analyses of the four kinds of Bilbao ore.

In examining these analyses the following points seem noteworthy. Rubio and campanil do not differ much in richness; but with rubio the associate minerals consist more largely of silica with combined water; and with campanil, of lime and carbon dioxide. Vena is richer in iron and manganese, and freer from silica than rubio and campanil. In lime and carbon dioxide it is on a par with rubio, and in freedom from combined water it is more than equal to campanil. It is, therefore, the richest and purest of the three kinds. Spathic ore has less iron, and more silica, combined water, carbon dioxide, and sulphur, than any of the others named. If calcined, however, it loses 25 per cent. of its original weight, and then becomes nearly as rich and pure as rubio.

The mining consists of the removal of ore by open-cut mining, in steps 30 to 60 feet high, and of a considerable length. In some cases a tunnel at the lowest level of the ore is driven in as far as the working face. A winze and mill holes are made down to the tunnel and through these the ore is shot and loaded into cars in the tunnel. The quarry levels are worked by this milling system down to and even below the tunnel, but in the latter case a shell of unworked mineral is retained around it, to protect the rolling stock from the results of blasting, etc. It is customary to drill deep holes into the working faces with jumpers, longer and larger ones being employed until a depth of nearly 30 feet is reached. A small charge of dynamite is then inserted and fired by a fuse to bull the hole into a chamber, after which is introduced a larger quantity of dynamite, and the explosion of this brings down a considerable quantity of ore. Quantities of from 2,000 to 3,000 tons are then brought down by a single blast. Vena ore can be worked out with a pick. After a blast the fallen masses of ore are attacked by men and boys with hammers, wedges, and bars, and broken into pieces that can be handled; impurities are here separated, and taken to the dump pile. The cost of quarrying and assorting varies according to circumstances. In a few campanil mines it is as low as 25 cents per ton; but in rubio mines, where more care is necessary, it may reach 50 cents. Of this from 5 to 10 cents per ton is for explosives and tools, and the remainder for labor.

Fig. 3. Iron Mines at Galdames, Province of Biscay, Spain

Two systems of aerial tramways are used, the Hodgson and Bleichert. In the Hodgson system an endless steel wire rope is made to travel by means of an engine fixed at the lower end. The buckets are hooked on to the rope at intervals, and move with it, passing over the pulleys as they come to them. The full buckets travel in one direction, which is generally down hill, and the empty buckets in the opposite direction on the return rope. At either terminus they are switched on to an outer rail, to be loaded or tipped, then run around on two rollers attached to the hanger, and finally entered on the return rope. At each angle for changing the direction of the route they are switched by hand. Stations must not be more than 2 miles apart and there may be several endless ropes on each set of towers, but two is the usual number. The quantity of ore conveyed in this way is about 2,000 tons per rope per week of 72 hours. Each bucket holds 400 pounds and one passes every 26 seconds, which is equivalent to 28 tons per hour. The place where the hanger grips the wire rope is furnished with

a seating of rubber, which acts as a spring, and prevents damaging the rope. The system will not do where the inclination exceeds 1 in 4, as then the hangers are liable to slip upon the rope in wet weather. To obviate this difficulty, the Bleichert system was introduced. In this the main rope is stationary,

Table 3. Average Analyses of Bilbao Iron Ores

Constituents	Rubio. Brown Hematite Per Cent.	Campanil. Red Hematite Per Cent.	Vena. Purple Red Hematite Per Cent.	Spathic. Ferrous Carbonate or Siderite Per Cent.
Ferric oxide, Fe_2O_3	80.71	78.426	85.71	
Ferrous oxide, FeO				
Manganese monoxide 77½ per cent. MnO	.92	1.303		
Manganic oxide 69½ per cent Mn_2O_3			3.50	
Alumina, Al_2O_3	1.22	1.130	.95	
Lime, CaO	.31	3.550	.35	
Magnesia, MgO	.10	.180	.16	
Silica, SiO_2	7.50	5.480	4.70	11.68
Carbon dioxide, CO_2	None	4.500	.20	
Sulphur, S	Traces	Traces	Traces	1.09
Phosphoric acid, P_2O_5	.03	.020	.02	
Combined water, H_2O	9.00	4.600	4.15	
	99.79	98.189	99.74	
Loss by calcination				25.00
Moisture	11.00	11.000	14.00	
Metallic iron, dry, Fe	56.50	54.900	60.00	
Iron in damp ore	50.29	48.870	51.60	40.00
Iron in calcined ore				53.33

the buckets traveling on it, as on a rail, and being hauled by a traction rope; when the gradient exceeds 1 in 4 the system becomes self-acting. The relative costs of construction are: Hodgson system, about $10,000 per mile, single line; Bleichert system, about $20,000 per mile; but the latter system is capable of conveying nearly one-half more than the former per annum. The working cost is about the same in each case, from 15 to 25 cents per ton per mile. In both systems a powerful brake is required to regulate the speed, and apparatus to keep the rope in uniform tension. There are a number of gravity planes which operate at a cost of from 6 to 8 cents per ton.

MINING LAWS IN SANTO DOMINGO

Written for Mines and Minerals, by C. A. Haussler

There are Americans who appear to be under the impression that the mining regulations of the Republic of Santo Domingo are similar to those of the United States. They are quite different and should be thoroughly understood by any one proposing to embark upon the mining enterprises in that island.

The Dominican Government is divided into three sections: legislative, executive, and judicial. Some Americans have supposed and assumed that special legislation is necessary whenever a new concession is granted in Santo Domingo, and that all mining rights emanate from the congress, that is, from the legislative power. This is not correct, for while congress can make the laws it is the function of the executive branch to enforce them.

Dominican laws are based upon the Code Napoleon, and under this system mining rights constitute property quite apart from land or surface rights or the actual ownership of the land. It is the duty of the President and his ministers to enforce the laws affecting all persons, the mining-concession holder as well as the land owner, in accordance with the specific mining laws that existed at the time the concession was granted. This law, therefore, is in fact a contract between the two parties aforesaid, made for them by act of congress, and the government is not a party thereto.

All existing mining concessions in Santo Domingo are those granted under the laws of 1876 and 1904, the latter having displaced the former, which in turn was abrogated this year, but not yet succeeded by a new law.

Concessions made under either of these two systems hold good respecting the laws in force at the date of the grant; thus a concession in 1876 is governed today and for all time, by the law of 1876; a concession granted in 1904 must be likewise subject to the law of that date. This of course assumes that the concessionnaire has in all cases complied with the law and paid his taxes, thus keeping the concession alive.

As might be expected of people absolutely ignorant of mining, all the Dominican mining laws, up to the present, have been defective in numerous respects, some clauses being unreasonably favorable to the concessionnaire or the land owner, while others are a serious detriment to both.

According to the law of 1876, any one could apply for a mining concession anywhere and even upon anybody's land, providing the land owner was duly notified. Within three months from the date of such notification, the landlord could claim his so-called "preference rights" which would enable him to take possession of the concession if he reemburse the concessionnaire for all expenses incurred in prospecting and developing the property. Under the law of 1876, a government tax of 2 per cent. was imposed upon the gross mineral product of the property, no other import being demanded, hence by this law the only effective forfeitive clause was that of non-production. Immense tracts of land were tied up in this way; the concessionnaire simply conducting one small productive operation, at any one spot upon the concession, no matter how large the concession may have been. It appears probable, however, that most of these old concessions have now been forfeited, since no mining whatever is going on in the island.

According to the law of 1904, the land owner in claiming his "preference rights" is entitled to 66.6 per cent. of the production of the concession, but he must assume charge of the operation and supply the concessionnaire with 33.3 per cent. of the net profit of his work. In all the concessions so far granted under this law, as far as known, none of the land owners could afford to claim their preference rights, and, consequently, the concessionnaries retain their full possession.

In case the applicant should be a share owner in the peculiar undivided properties known as "communeros," the other coowners are not entitled to any participation, no matter how small a share the concessionnaire may hold in the communero.

Under the law of 1904 there is no government tax on the product, hence the exploitation of the concession is in law no concern to the government as long as the holders pay the yearly tax of $10 per 100 tareas (64 cents per acre) when for precious metals, and $5 per 100 tareas (32 cents per acre) for concessions taken for all other metals and mineral products.

The non-payment of this tax is the only cause for forfeiture.

It is well to note in this connection that land is not taxed in any manner whatever in Santo Domingo.

A rich island like Santo Domingo, wherein there is good cause to expect considerable mineral wealth, should be provided with a code of good and just mining laws similar to those now successfully in operation in Mexico.

The Dominican Government stands in urgent need of competent advice in framing the proposed new mineral code, and such information can only be obtained from experienced mining men familiar with similar laws in different parts of the world.

The United States laws would never do in Santo Domingo, nor would the Mexican and South African (Transvaal) mining laws; unless modified to suit local conditions.

The Code Napoleon considers in a broad way that the mineral in the earth belongs to the state, while our common English law regards the owner of the land as possessing everything from the heavens above, to the depths beneath, within his surface boundaries. It is perhaps a debatable question which of these two fundamental principles is best adapted for the development of a new country.

MINES OF MISSOULA COUNTY, MONTANA

*Written for Mines and Minerals, by J. P. Rowe**

Placers on Nine-Mile and Cedar Creeks. Developments on Veins in Other Districts

Missoula County, in the western part of Montana, is bounded on the north by Sanders County, east by Powell County, south by Granite and Ravalli counties, and west by the high precipitous Bitter Root Mountains and the southeastern end of the Coeur d'Alene Mountains. The summit of these mountains is the dividing line between Idaho and Montana. Missoula County has an area of about 5,000 square miles, and the Missoula River flows northwesterly through it. The Missoula River, whose name changes to Hell Gate, Missoula, and Clarks Fork of the Columbia, depending upon its locality in the county, has many minor tributaries. The entire area of the county is made up of mountains and intermontane valleys, and in all save the valleys conditions are such that ore-bearing veins might be found. There are at present six mining districts where ores are found in various geological formations.

Mountain making in this territory was probably all accomplished from the close of the Laramie epoch to or through the Miocene, when dynamic disturbances fractured and fissured the rocks and injections of igneous material formed laccoliths, bosses, stocks, sills, batholiths, and dikes.

The intermontane valleys during the Neocene epoch contained lakes, and in several localities lake-formed beds from deposits of volcanic ash furnish excellent specimens of the Neocene flora. The rocks in the mining districts are chiefly granites and quartzites, except at the northwestern end of the country or the eastern Coeur d'Alenes, where they are largely siliceous shales.

The mining in Missoula County is confined chiefly to gold placer, gold quartz, copper, and lead-silver minerals, besides some lignite deposits found in the Neocene lake beds that are being worked in a desultory way.

It is not possible in a necessarily restricted article of this kind to give a detailed description of each mining district in Missoula County; however, this article is sufficiently broad to enable the reader to form a fairly good idea of six ore-mining districts of the county and the conditions of mining in the placer and lignite beds.

Of the several placer properties in Missoula County, the ones now being worked are the Nine-Mile placers, the Windfall placers, and the Kansas City Commercial Co.'s dredging property on Cedar Creek.

Fig. 1. Flume, Cedar Creek, Montana

The Nine-Mile Creek placers have been worked since 1874. These placers are about 12 miles from Huson, a station on both the Northern Pacific and Chicago, Milwaukee & Puget Sound Railroads. Several placers of more or less note have been worked in the past on this creek and some of them are still paying. The small creeks leading into Nine-Mile Creek furnish the best placer grounds. From a general examination of the Kennedy Creek placer it appears that this company has considerable good ground, plenty of water, and a remunerative dredging proposition.

The Windfall Placer Mining Co., operating on Windfall Creek, in the extreme western portion of Missoula County, has one of the most extensive placer deposits in the county. The Windfall company owns 300 acres of virgin placer ground that extends 2 or 3 miles along the creek, besides 80 acres of timber land. The placer, which is 12 miles from the railroad, varies in width from 100 to 500 feet and has an average depth of about 35 feet. This ground averages about 15 cents per cubic yard. The company expects to clean up nearly $100,000 during the coming season. It is estimated that the ground tested will yield over $5,000,000, and it will be worked by hydraulicking and bed-rock flumes. The company has an estimated water supply of 1,500 miners' inches during the working season of about 6 months.

Fig. 2. Dredge on Cedar Creek

The Cedar Creek placers, discovered in 1869, have been worked almost continuously since. In the early days this creek boasted a population of 10,000. The LaCasse Brothers, until recently, owned and operated the ground near the head of Cedar Creek, about 15 miles from Iron Mountain. At present, however, the only operating company on Cedar Creek is the Kansas City Commercial Co. This concern owns 570 acres of placer and operates a 100,000-cubic yard per month dredge, which, costing about $100,000, is capable of digging 55 feet below the surface. The ground is said to yield 25 cents per cubic yard in gold valued at $19.75 per ounce The dredge shown in Fig. 2 is run by electricity generated at the company's power plant 3 miles down Cedar Creek from the dredge. The amount of current generated at the switchboards of this plant is 2,300 volts, but this suffers a line loss of nearly 10 per cent. before it reaches the boat. The coal burned in the boilers of the power house is Montana subbituminous coal that costs $7 per ton laid down at the company's bunkers. The company plans to construct a ditch for 2 or 3 miles, and by using a large Pelton water wheel, which has already been installed, generate the electricity by water-power, thus saving the price of fuel and labor of stokers, which in a year is no small sum.

Ten of the 15 miles from Iron Mountain, or to the Amador mine, is traveled by means of a large gasoline speeder over the Amador Mining Co.'s old railroad track, but from this place to the camp a wagon road has been made.

The dredge is operated 10 months in the year, with three shifts of men, each working 8 hours. About 30 men are employed, with wages for the boatmen running from $3 to $5 per day.

The Clinton mining district is in the eastern portion of the county in rather a rough country, 2½ miles from the town of Clinton. Transportation facilities are the best, as the main

* Professor of Geology, University of Montana.

lines of the Northern Pacific, and the Chicago, Milwaukee & Puget Sound Railroads pass through this town. The haul from the district is all down grade and the ore can be transported to Clinton from the mines for $1 a ton.

The mountains are rather high and mining by tunnels is possible to a depth of 800 to 1,000 feet for the most of the properties. The country rock, largely hornblende granite, is the

Fig. 3. Upper Tunnel and Ore Bins, Iron Mt. Tunnel Co.

western portion of a granitic batholith, which extends east for several miles to the Copper Cliff and the Garnet mining districts of Granite County. This granitic mass was pushed through the sedimentary shales, sandstones, and limestones, and in localities especially near the contact, metamorphism has changed limestones to marbles, sandstones to quartzites, and shales to slates. The age of the batholith and later vein formation is probably Post-Laramie. There are a few dikes of porphyry, some of which cut the veins, and a few small faults are found. For the most part the veins of the Clinton mining district vary in width from 2 to 24 feet. There are several nearly parallel veins with a general northeast and southwest strike that dip about 60 degrees to the west. The ore is composed largely of the copper minerals malachite, azurite, and some native copper in the oxidized zone, chalcocite and covellite, with some bornite and chalcopyrite in the zone of secondary enrichment; and in the primary zone, chalcopyrite with some bornite. The gangue is quartz with small amounts of pyrite, siderite, and some barite as accessory minerals. Several groups of claims have by consistent development work been proven of value, the best known and best developed being the Cape Nome, the Hidden Treasure, the Aladdin, and the Triangle. Although the discovery of the district was in 1878, active development did not start until about 6 years ago.

The Cape Nome property, in Missoula, comprises three patented claims. There are two veins which extend through this property. One of them, known as the main vein, is from 2 to 8 feet wide, and the other, called the "blind lead," about 70 feet west, has a width of nearly 8 feet. This company has done considerable development work and has gone to a depth of over 500 feet. There are two main tunnels; the upper one is driven on the vein, while the lower one is a cross-cut. A double-compartment shaft was sunk 500 feet, and two levels were driven on the vein from this shaft, the 300-foot level and the 500-foot level. The former level has developed the vein nearly 700 feet, and the latter over 900 feet. The presence of an ore shoot whose exact size is unknown, as the levels have never been driven through it, has been proven. From the 300-foot level to the surface and for a distance of many feet on the vein, the ore shoot is ready for stoping. There are about 4,800 feet of tunneling, shaft sinking, etc., done on this property. The surface improvements are first class for a new property, and consist of boarding and bunk houses, shaft house, and blacksmith shop. The equipment consists of two 120-horsepower boilers, a 40-horsepower hoister, sinking and stationary pumps, besides the necessary mining tools and supplies. The locality is ideal, there being water and wood for all ordinary mining and domestic purposes.

Smelter returns from carload shipments of mine-run ore gave 5 to 6 per cent. in copper, and 7 to 12 ounces in silver.

The Hidden Treasure property consists of eight patented claims and has been developed by means of an upper and a lower tunnel. There is a vein about 20 feet wide between porphyry and granite that extends through the group and is parallel to the Cape Nome vein. It is southwest of the Cape Nome property and can be worked to a depth of 700 or 800 feet by the tunnel method. The ore is chiefly chalcopyrite, carrying a few ounces of silver. In the upper tunnel a fairly good ore shoot has been exposed, yielding 8.5 per cent. copper and 4.68 per cent. silver, when the samples were taken across the vein. The lower tunnel is now in 1,400 feet and a wide vein is exposed carrying fairly good values in copper and silver. This property has proved its worth as a copper mine and the near future should see it a large producer. There is plenty of wood and water on the property for all mining purposes.

The Aladdin group comprises three claims, and has running through it the Cape Nome vein. It lies directly northeast of the Cape Nome property. There are several shallow shafts and surface tunnels on the property besides the 1,800 feet of drifting done at the 500-foot level in 1907 and 1908 through the Cape Nome shaft by the Speculator Mining Co., of Butte, Mont. This is a promising property and will doubtless some day develop into a good mine.

The Triangle property comprises 11 claims; the two most developed being the Triangle and the Grass Widow. There are two nearly parallel veins extending through the Triangle claims, and one vein that extends through the Grass Widow. The country rock is granite, and the ore is largely in parallel stringers of chalcopyrite. The main Triangle vein has been drifted upon for a distance of 540 feet and has a maximum cover of about 400 feet. The Grass Widow has been developed through the Triangle tunnel. The veins of the two claims being parallel, a cross-cut tunnel driven from the main tunnel on the Triangle claim cuts the Grass Widow vein at a greater depth, with less expense than any other method. Besides the eleven

Fig. 4. Mill, Iron Mt. Tunnel Co.

claims, this company has a lease on the upper workings of the Hidden Treasure claim and owns 120 acres of the East Clinton town site, besides a mill site at Clinton. The mine is being rapidly developed, and the company expects to build a 100-ton concentrator at Clinton early in the spring.

There are several partly developed groups of claims in the Clinton district, but the ones mentioned are the most important, and give an idea of the ore deposits and mining conditions

of the district. It is probable that this will be an important copper-mining district in the future.

The Lo Lo mining district is not well known, as no shipping properties have been developed. There are several east-west veins on both sides of Lo Lo Creek, some gold bearing and some copper. The country rock is largely quartzite, but silicious shales are also present. While there is no igneous rock near the most promising veins, however, to the south and west the mountains are composed almost entirely of granite and gneiss.

On the south side of Lo Lo Creek, about 7 miles from the town of Lo Lo, there are true fissure veins, striking almost east and west, and dipping to the south. The vein that has been most developed assayed about $8 per ton in gold 150 feet below the surface. It is 4½ to 5 feet wide with quartz, iron pyrite, and iron oxide as the gangue minerals. Assays were made from material taken from the outcrop on one of the three claims examined and from another outcrop not in the property, but from one of the veins that passes through the property, and both assays gave about $60 per ton in gold. Not much is being done at present in this district owing to insufficient capital to do proper development work. Transportation is not to be a serious problem from any part of this district.

The Lothrop mining district is situated southwest of Lothrop, a small town of the Coeur d'Alene branch of the Northern Pacific Railroad, and on the main line of the Chicago, Milwaukee & Puget Sound Railroad. The ore deposits are of copper minerals. The district is not well known and but two properties are at present being worked. The country is part of a great anticline of quartzites and jasper slates. There are several dikes of porphry on both sides of the best deposits, and the country is quite badly faulted. The deposits in the Lothrop district for the most part are bedded, generally quite wide—from 10 to 60 feet. The ore is largely chalcopyrite in stringers that branch in every direction within the mineralized zone. The bed appears to have been fractured and the fissures formed channels for the waters carrying the ore in solution to circulate. There are, in places, small openings filled with quartz druses, but generally the fractures are filled with either chalcopyrite or quartz. Near the town of Lothrop a small vein about 2 feet thick cuts the country rock. It contains considerable barite, besides the quartz gangue, and yields a small amount of gold.

Some of the thin seams in the impregnated zone of stratified rock yield more than 10 per cent. copper, and are identical with the Snowstorm copper ore of Idaho. Beautiful specimens of chalcopyrite in quartzite are sometimes found; however, the best of these strata are but from 1 to 2 inches thick and are surrounded by barren or very lean strata. The impregnated strata, north of the West Fork or Petty Creek, dip north at an angle of from 10 to 45 degrees; south of the West Fork the veins dip in a southerly direction. Most of the properties of this district could be worked for several hundred feet by tunnels. At one place on the Hailstorm property a depth of 2,000 feet from the outcrop could be gotten. There is plenty of wood and water for all mining and milling purposes.

There are many properties around Lothrop that are being gradually developed, but the Hailstorm and Coppersmith properties have taken the lead. The district has never been a shipping locality but several of the properties seem to warrant thorough development.

Before describing the Iron Mountain and Carter districts it will be interesting to know that on the south and west sides of the railroad which follows the Missoula River to St. Regis, and then up the St. Regis River, the mining properties other than gold are copper, and on the opposite side the deposits other than gold are primarily lead. This condition begins in the neighborhood of Lothrop and continues to the Amador copper property, near Iron Mountain, and to the north and west it prevails in the Monitor and Bullion mines. Opposite the railroad, across the river, near Iron Mountain, is the Iron Mountain lead-silver mine; the Mountain Gem mining property; the Glen Metals; O. R. & N.; the lead-silver properties at Carter; and still farther northwest on the lead-silver side of the river is the Saltese district with the Tarbox, Ben Hur, Last Chance, Bryan, and other lead-silver properties. This country has not been studied sufficiently to even attempt an explanation of the occurrence. It is true, however, that the lead-silver deposits are found largely in the eastern portion of the Coeur d'Alene Mountains, and the copper deposits are found largely in the Bitter Root Mountains.

The Iron Mountain and Carter districts have been worked for some time, and a considerable amount of pay ore has been taken from the deposits in these localities.

The rocks probably belong to the pre-Cambrian period and are largely silicious slates. The bedding planes of the slates have a southerly dip and strike northwest and southeast. The veins are found generally following the dip and strike of the bedding planes of the country rock, and appear to have been formed in crushed or faulted country rock. There are several parallel veins that strike across the country in these districts and their continuity for several miles to the Saltese district, seems certain. The writer visited several of the mining properties in these districts and traced the veins on their strike for several miles. The width of the zone of vein formation is from ½ to 1½ miles, and the length is 30 miles. The ore for the most part is argentiferous galena, with varying amounts of zinc. The mines at present in operation, beginning with the Iron Mountain Tunnel Co., near Superior, and continuing west, are the Mountain Gem, the Glen Metals, the King and Queen, the O. R. & N., all containing silver-lead veins; and the Carter Mining and Milling Co.'s gold property. The Iron Mountain Tunnel Co., has been rather a large producer and is now mining and concentrating about 100 tons per day.

The Iron Mountain Tunnel Co.'s property, shown in Fig. 3, is located 3 miles from the Chicago, Milwaukee & Puget Sound Railroad, and 4 miles from the Northern Pacific Railroad. A good wagon road, all down grade, connects the mine and mill with the railroads. This mine has been developed by means of several tunnels and in the 80's some very rich lead carbonates were taken out, running as high as 600 ounces per ton in silver. At present the company is operating the mine through the new 5,600-foot tunnel and taking out the ore below the tunnel level, which is 1,600 feet below the surface. The only ore shoot that has been so far worked is 500 feet wide, and it has been stoped to the 1,750-foot level. The vein at the 1,850-foot level is 22 feet wide, with bunches and kidneys of ore throughout. There is at present a stope 100 feet high the entire width of the ore shoot. The three-compartment shaft is down 250 feet below the tunnel level. The ore shipped last June gave the following smelter returns: Lead, 23.2 per cent.; zinc, 12.5 per cent.; silver, 43.5 ounces per ton.

The company now has a modern 100-ton concentrating mill, shown in Fig. 4, and only concentrates are now shipped to the smelter.

If this property is representative it promises well for the future of the district.

There has been considerable ore shipped from the active development work now in progress on the Glen Metals, O. R. & N., King and Queen, and other properties in the neighborhood of Carter.

The veins of the Saltese district are undoubtedly a continuation of those occurring at the Iron Mountain mine and those at Carter. The chief difference between the veins of the Carter and Saltese districts is that the latter carry a large amount of arsenopyrite and antimony, intimately associated with the galena. The amount of zinc present in the Saltese veins is smaller than at the districts farther east.

In the Saltese district the vein extends with a general strike northwest and southeast, and a southerly dip through the chief

properties. The outcrop and tunnels show that the deposit continues through the Tarbox, Ben Hur, Last Chance, and Bryan properties, all of which have been worked for carbonate ores. There are other lead-silver properties in this district, but at the present time none are shipping ore.

The copper properties at the town of Saltese could probably be said to belong to this district. The principal copper mine, named the Monitor, shipped several thousand dollars worth of ore, and until last summer's forest fires, its future looked bright. The Monitor has a shaft 700 feet deep with several levels and many hundred feet of drifts. The surface equipment, consisting of shaft house, air compressor, engines and hoisting machinery, several bunk and boarding houses, and 5,000 cords of wood, were completely destroyed by the great forest fires that swept through this section of Montana in 1910. Other copper properties that have been more or less developed are the Copper Dome mine, the Optimo, the Keystone, the Bald Mountain, the Jumbo, and the Boston-Colby. None of these, however, except the Monitor have been commercial producers.

The other mining interests of Missoula County consist of lignite deposits that are quite thick in some localities and cover large areas. They were formed during the Neocene epoch and are found in almost every intermountain valley of western Montana or in the old Neocene lake beds. The deposits best known are those near Missoula, those near De Smet, and those up Nine-Mile Creek. Part of the deposits near Missoula are owned by the Hell Gate Coal Co., and the Standard Lime and Brick Co., both of Missoula. The former company has been operating for some time, and sells the fuel to the people of the city at from $4 to $5 per ton delivered. The seam of the Hell Gate Coal Co. is about 7 feet thick and intercalated somewhat with bone, dips 15 degrees or 20 degrees to the north, and has a general easterly and westerly strike. According to drill holes reported to have been made several years ago, a section through the Missoula coal beds gave 52 feet of clean lignite in going 290 feet below the surface. One seam is reported to have measured between 17 and 20 feet thick. The Missoula lignite gives the following proximate analysis: Moisture, 9.31 per cent.; volatile and combustible, 41.17 per cent.; fixed carbon, 39.60 per cent.; ash, 9.92 per cent. This analysis was made from a selected piece and does not represent the amount of ash that would be found in the run-of-mine product. The fuel ratio is .96 and the caloric value is about 7,500 British thermal units, or 4,167 calories. This lignite makes a fairly good domestic fuel, but slacks too badly on being exposed to the air to be of value as a shipping fuel. It is a good quality of lignite for making producer gas, and recent experiments also show it to produce an illuminating gas.

ALUMINUM BRONZE

Aluminum alloys with copper in all proportions and produces homogeneous mixtures. When the aluminum goes beyond 10 per cent. the bronze gets brittle, and a 12-per-cent. mixture is hard enough for hard dies. For sand casting the 10-per-cent alloy is generally used, as it is hard and tough and has the properties of a strong metal. The great difficulty in casting aluminum-bronze is its oxidation when melted. It must be poured without agitation, to avoid dross. Dross formed in melting may be skimmed off, but that which is formed while pouring enters the casting. Another difficulty is the shrinkage. By the use of sufficiently large risers, and the avoidance of sharp corners in the casting, and an ample gate, this shrinkage may be overcome. The toughness of aluminum bronze is greater than that of any metal except steel. It works hot better than copper, as it is softer at a red heat and it is not black-short like copper and zinc alloys.—*Brass World.*

GENERAL ORE MINING NOTES

F. Augustus' Forty-Seven Varieties Luck.—It is a newspaper report that F. A. Heinze, ex-copper magnate, has procured gold properties from which $25,000,000 is to be gleaned this coming year. These properties are adjoining the Dome mines in the Porcupine gold fields, Ontario, Can. A number of Nome, Alaska, men are said to be in on the ground floor in the new diggings, and the following is what one says: "One prospector has stripped the vein for a distance of 50 feet and polished it in places, so that gold is visible all along. His trench is 3 feet deep and he asks $200,000 cash for it as it lies. A party of Alaskans offered the owner of this claim $5,000 a shot for all ore that could be blown out with two sticks of dynamite, but was refused. The field is situated 155 miles from Cobalt. The Toronto & Northern Railroad has already completed its construction for 100 miles, and 55 stage coaches are carrying passengers between the end of the railroad and the new Eldorado." According to reports they play "try your luck" as follows: Instead of three shots for a nickel and every time you hit the baby you get a cigar, one pays $5,000 a shot and gets all the ore the shot lifts. The report gives no information as to who originated the game, though it states the Nome men are glad to have Heinze in the new camp with them.

New American Tunnel Record.—Unless something turns up to beat it, which seems improbable, a new American tunnel-driving record has been established at the Laramie-Poudre irrigation tunnel by the force under Contractor James A. McIlwee. During the month of January, the east heading, using water Leyner drills, was advanced a total of 609 feet through solid granite, the progress for the last 4 days of the month averaging 28 feet each. This exceeds any record made by the Los Angeles aqueduct and has been beaten only by the Simplon tunnel.

First "Process" of 1911.—To Dr. O. B. Dawson, of El Paso, Tex., belongs the honor of "inventing" the first process devised this year. It is so extremely simple that it is to be wondered that it has not been "discovered" before. After crushing, some 5 or 6 tons of the ore are charged without fluxes into a rotary furnace and heated by oil or gas or any old kind of fuel. The furnace is then "closed against the air" and 20 pounds of sulphur per ton of ore are added. After rotating for 20 minutes, the product, which is in the form of fine metallic shot, is discharged into a water "laundry (sic)." "All the mineral in the ore has been reduced to metal and nothing has been fused. The charge is ground more finely and passed onto a concentrating table," where all the metals are presumably caught. Among the many advantages claimed for this ingenious process are that "minerals may be reduced to metals at a cost of about $1 to $1.20 per ton," that even this cost will be more than made up because "there can be no loss in sliming as all mineral is reduced to metal," and no high temperatures will be required "as no fusion of the rock takes place," and consequently the heat necessary is only that required to reduce the mineral to metal, "which is low by reason of the exclusion of the air." Thanks to Doctor Dawson, America has come into her own and the metallurgists of the Fatherland had better look to their laurels. This reads like another Thermos-bottle furnace.

Closing of the Tombstone Consolidated.—The attempt on the part of the Development Co. of America to unwater the Consolidated mines at Tombstone, Ariz., has been abandoned permanently on account of the excessive cost of the project. This will prove a setback for the camp, the loss of a pay roll estimated to have been as high as $2,000 a day being a serious matter.

Missouri Zinc Area Widening.—During the year 1910 two new zinc-ore camps in the Joplin district were put on a paying basis; namely, the North Fork camp, at Neck City, and the Thom's Station camp, north of Joplin. The ore at Neck City

is of high grade and is in soft ground, while at Thom's Station the ore is partly free and partly disseminated.

Zinc Smelting Companies Mining Ores.—Following the policy pursued by the American Zinc Lead Smelting Co., the Granby Mining and Smelting Co., the American Metal Co., through its subsidiary the St. Paul Mining Co., and the Empire Zinc Co., a subsidiary of the New Jersey Zinc Co., began the work of prospecting and mining on their own account in 1910. This makes four of the largest companies smelting zinc ore to enter the Joplin field for the production of part of their ore supply.

Leadville Zinc Ore.—An adjustment satisfactory to the shippers has been reached at Leadville by the buyers and producers of zinc ore. The decision of the smelters to make some changes in their schedules that would reduce the profit, especially on oxidized zinc ore, was followed by a temporary suspension of work by a number of the larger producers. Now, however, the smelters have agreed to stick to the old rates, and work has been resumed with a prospect of an early increase in production.

Portland Gold Mining Co. Meeting.—The annual meeting of the stockholders of the Portland Gold Mining Co. was held at Cheyenne, Wyo., in February. The most interesting feature in the year's work was the opening of the new Portland mill that treats about 350 tons daily, and the beginning of drainage from the company's mine through the Roosevelt tunnel. The treasurer's report shows that during the year just closed 67,515.20 net tons of ore of a value of $1,241,168.30 were mined. Receipts from all sources were $2,345,210.06, out of which expenses of $1,970,360.08, and dividends of $240,000 were paid. Including ore on hand, cash in bank, etc., the total assets at the close of business were $5,402,898.48.

Cost of Mining in Cripple Creek, Colo.—A comparative statement of the cost of mining operations, contained in the annual report of the Elkton Consolidated Mining and Milling Co., of Cripple Creek, Colo., gives the cost of breaking ore (labor only) in the properties of the company on Raven Hill, at 56.8 cents per mine car, and 81 cents per ton, while the cost of breaking waste rock is 92½ cents, or $1.32 per ton. The total cost of ore shipped (including underground labor and power) was $2.44 per ton. The average wages per shift of the entire mine, including the salaries of manager, superintendent, and office force, was $3.70. The Elkton company has paid $2,704,460.47, exclusive of the dividend of $37,500 paid February 25. The dividend number was 101.

Dead and Don't Know It.—The Las Vegas *Age*, Nevada, says: "James Patterson, shot and reported killed by the half-crazy Indian Queho, is much incensed over his death, and that Patterson is oiling up his old musket preparatory to going on a hunt for the man who killed him. Patterson says such reports are damaging to the mining interests of the Southwest, both on the Arizona and Nevada sides of the Colorado River."

Calumet & Hecla Consolidation.—The opposition to the consolidation of the Calumet & Hecla subsidary companies must have been disappointed, as all the companies voted for the merger. The consolidated corporation is to be known as the Calumet & Hecla Mining Co., $10,000,000 capital stock, par value of the share $25. The valuation of the several properties entering into the consolidation is agreed to as follows:

Calumet & Hecla	$4,270,850
Seneca Mining Co.	150,000
Ahmeek	1,000,000
Allouez	425,000
Centennial Copper	125,000
Laurium	100,000
Osceola Consolidated	1,201,875
Tamarack	330,000
La Salle	151,475
Superior	450,000
Total	$8,214,200

The property and assets of the several companies are taken in full payment of 328,568 shares of stock.

American-Mexican El Tigre Mine.—W. L. Rynerson, who is working some of the El Tigre, an American property in Sonora, Mex., says the 250 insurrectos under Colonel Blanco, who captured the camp, were well behaved and did not molest any American person or thing. They purchased merchandise from the El Tigre Mining Co.'s store. The Mexicans, however, had to help the cause without delay or there was confiscation.

Drainage Law for Colorado.—Among the bills presented to the Colorado Legislature, is one of interest to the metal-mining industry of the state. This bill authorizes the creation of drainage districts in the various mining camps similar to public-improvement districts in the various cities. The funds needed for driving a drainage tunnel will be raised by a special levy on all mining properties to be benefited, and what remains after collection by the regular politicians will be paid into the hands of a board of supervisors, which will be in charge of the drainage tunnel work. Possibly as much as 30 per cent. of the money raised in this way would go into the tunnel. A similar bill failed to pass 4 years ago. The Roosevelt tunnel was then started and now the complaint is made that many prominent companies in Cripple Creek are draining through it which have never contributed to its construction. If a mine owner is content to go without a drainage tunnel, while another must have one, why should he pay for its construction? If, however, a tunnel is constructed that unwaters his mine he should donate at least part of his former cost of pumping.

Cow-Bell Hoisting.—At a large Michigan iron mine, where Kimberly-style skips are used, the skips run with practically no noise and the skip tenders have to watch carefully to keep from dumping ore into the shaft, thinking that the skip is in position. One night the top men removed a bell from a cow which had wandered near and placed it on the skip. This bell would ring when the skip started and stopped, as well as when it was in motion, and was found to be so convenient that soon all the skips and cages upon the property were equipped with cow bells. Where the bells came from has not been inquired into by the management, but they would gladly supply any bells which would be needed to keep the skips and cages equipped, as it has been found to be more convenient and safer than when not so equipped.—E. S. Dickinson.

Cripple Creek Drainage Tunnels.—At the beginning of March the discharge from the Roosevelt tunnel, Cripple Creek, was at the rate of about 9,000 gallons per minute. Contrary to expectations the drainage seems to be derived pretty generally from the camp and not entirely from the mines immediately on the line of the tunnel. It was originally supposed that the north and south dike traversing the district would have to be crossed before the mines on the east side could drain into the tunnel, but from the action of the water in some of the mines on that side it would appear that the dike is broken by a fault which allows the water to pass it. It is estimated that the life of the Cripple Creek mines to the level of the Roosevelt tunnel will be about 8 years. In order to prevent suspension of work when this depth is reached, the directors of the Cripple Creek Deep-Drainage Tunnel have authorized its mining engineer, Mr. T. R. Countryman, to begin preliminary surveys for a new tunnel which will tap the workings 500 feet below the level of the present Roosevelt tunnel. It is said that work will be begun as soon as possible on the new enterprise.

The American Mining Congress is actively agitating the adoption of workmen's compensation laws by the various coal mining states with the object of providing a fund by a small tax on coal production, to furnish indemnity for the victims of mine disasters and a pension for aged mine workers. Recently a committee of the Mining Congress drew up a draft for a law of this character, which is now being submitted to the legislatures of many mining states for their consideration. The Mining Congress is also working for more efficient inspection and regulation of coal and metal mines with a view to decreasing the loss of life in mine disasters; the standardization of electric practice in mines, the general revision of the mineral land laws, etc.

REPORT ON THE POVERTY GULCH MINE

*By Charles W. Henderson**

Geology, History, Description of the Ores, Cost of Labor, Freight, Treatment, Power, and Supplies

Part I. Preliminary.—The object of this report is to describe in detail the operations of a mine through the various stages of prospecting, developing, and mining, with maps, plates, and specifications for plant and buildings, followed by subsequent assumptions regarding profit or loss that might ensue from marketing of a certain number of tons of ore per day.

Conditions surrounding the report are that the property consists of four overlapping claims located in Section 18, T. 15 S., R. 69 W., in Cripple Creek district, Colorado. All claims are 1,500 feet long by 300 feet wide. The ore deposits consist of mineralized sheeted zones in breccia and connected with two dikes. The character of the ore and the mineralogy are the same as that in the Portland mine nearby. The values occur in three shoots A, B, and C, having the following dimensions:

Shoot	Length	Width Feet	Average Width Feet	Values	Average Value
A............	500	0–12	3	0–$150	$15.00
B............	700	0–10	3	0–$150	$13.00
C............	300	0–40	16	0–$150	$20.00

The ore in shoot C is formed by solutions from two dikes, one basic and the other phonolite, and the ore deposit follows down on the intersection of these dikes which dip 60 degrees

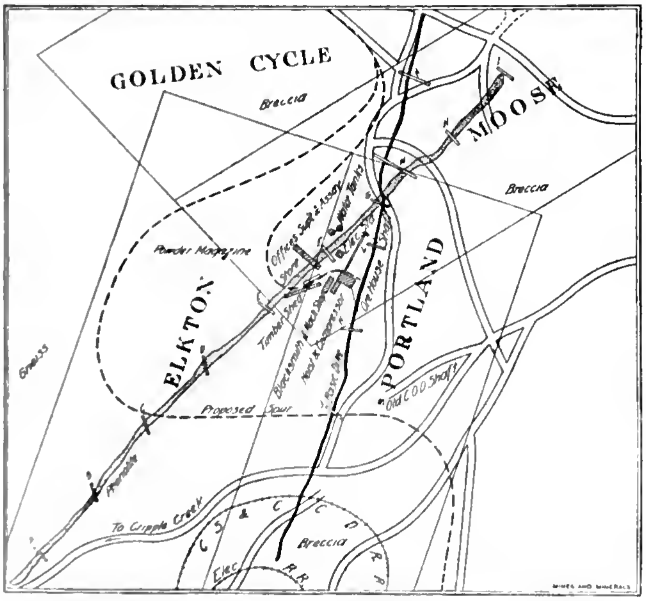


FIG. 1. MAP SHOWING LOCATION OF MINE

to the west and 50 degrees to the east, as shown. In order to plan the developing and prospecting of the property so that at the end of 3 years the mine will be producing 200 tons for 24 hours, it would be necessary to design a surface plant capable of handling this tonnage from a depth of 100 feet. At the end of the development period there should be a reserve of 18 months ore supply blocked out. At the end of the development period, the mine, it is assumed, will produce 780 gallons of water per minute, and therefore a pumping plant necessary to handle the flow must be designed. Assume that 54 per cent. of the water is intercepted at the 600-foot level and the rest is pumped from the bottom.

The mining district in which the property is located is in a group of hills some 7 to 12 miles southwest of Pike's Peak in the western part of El Paso County, Colo. It is 30 miles west of Colorado Springs and 75 miles south of Denver, near the center of the territory embraced by the Pike's Peak sheet of the Geological Atlas of the United States, issued in 1894. The general elevation of the district is from 9,000 to 10,000 feet above sea level. It is situated between Beaver and Oil Creeks, two tributaries of the Arkansas River. The former on the east has several tributaries which rise on the southern face of Pike's Peak, while Oil Creek, on the west, drains a large surface of the plateau west of Pike's Peak. The creek from which the town and mining district derive their name is one of the numerous branches of Oil Creek which has cut deep gorges near the rocks on the way to the main stream. The area covered is 6 miles in length and 5½ miles in width, in high rolling grass-covered ground between the two main ranges of the Rocky Mountains in Colorado.

The climate of the region is dry and the vegetation correspondingly scanty. There are three railroads entering Cripple Creek. One, the Colorado-Midland, enters from the north, the Colorado Springs & Cripple Creek District Railroad enters on the east, and the Florence & Cripple Creek Railroad, from Florence on the Denver & Rio Grande Railroad, enters from the south. These railroads furnish easy access to Colorado City, where are the custom chlorination mills of the Golden Cycle, the United States Reduction and Refining Co., and Portland Gold Mining Co.; to Pueblo or Denver, where are smelters of the American Smelting and Refining Co.; to Salida, where an independent company, the Ohio & Colorado Co., has located its smelter.

In addition, the district is traversed by electric passenger lines belonging to and connected by switches to the Colorado Springs & Cripple Creek District Railroad, called the Short Line, by which any mine in the camp can be reached at the expenditure of from 10 to 25 cents, and with no more walking than is necessary in any suburb of a large city.

The accompanying sketch map shows the situation as regards railroad and smelter points, and the United States Geological Survey Topographical map shows the steam and electric railroads in the district.

History of the District.—The Pike's Peak gold excitement in 1859 and the collapse of that boom occurred many years before gold was known in Cripple Creek. In 1874 H. T. Wood, who had been on the United States Geological and Geographical Survey under Hayden, from observations made several years before while on the survey, formed a party and started out to make investigations in the region now known as Cripple Creek. He found some float, and in August, 1874, formed the Mount Pisgah Mining Co. The usual rush was made to the region. A number of shafts and tunnels were opened, but the rich ore bodies discovered later by the more fortunate prospectors of the present Cripple Creek were not encountered. From the old openings found it is now evident that the unsuccessful explorers of 1874 were in the very heart of the Cripple Creek district.

The second mining excitement was in 1884 and was started by the alleged discovery of fabulously rich gold deposits west of Mount Pisgah. During this rush about 5,000 people were soon on the ground. They found that the prospect of rich gold had been salted and the originator of the scheme fled to escape being lynched, after which the district was rapidly deserted.

The first to discover valuable gold deposits in the Cripple Creek district was Robert Womack, who owned a ranch in that section, and who, led on by his frequent discoveries of frag-

* EDITOR'S NOTE.—The Poverty Gulch mine is the subject of a Mining Thesis by Charles W. Henderson. It appealed to the Editor to such an extent that he asked for and was granted permission to publish it. To those who would learn how to make a mine report it is recommended; to those interested in economical and scientific mining it is recommended; to those who do not know it all, it will meet with approval. Owing to the length of the Thesis it is necessary to divide it into sections, each of which is a complete article replete with useful information.

ments of rich ore, continued his search for the source of these pieces until, in the fall of 1890, he found ore in place and located the El Paso claim on which Gold King mine, in Poverty Gulch, is situated. He took some specimens of ore to Colorado Springs, and, on the strength of these in the winter of 1891, E. C. Frisbee and E. M. De La Vergne went into the region, making their headquarters at the Broken Bow Ranch of George W. Carr, near the site of the present town of Cripple Creek, and prospected with success. They were the first to really open and develop valuable mines in the district. The first paying mine was the Gold King, in Poverty Gulch, which was opened by Frisbee on the El Paso claim, and from which ore was shipped in 1891. Other discoveries followed, and shortly after the opening of the Gold King, De La Vergne opened the Raven mine. Thus successfully started, the region began to develop. In the spring of 1892 the rush began, and for the third time people streamed in. More discoveries were made and the region rapidly developed. The record, however, of the two previous attempts to find gold made many people skeptical as to the real value of the discoveries. The developments in 1892 and 1893 were so encouraging that in the latter part of 1893 and 1894 capital turned toward the district, and in the fall of 1894 there were over 100 mines, more or less developed, of which nearly all are strong mines today.

The production of the district from 1891 to 1894, inclusive, has been estimated by the mint of the United States at Denver at $5,543,967, and by various local authorities at over $7,000,000. The district has produced to the end of 1909 in gold $203,451,312, and 1,002,940 ounces of silver.

In all the central part of the area, where the hills are made up of volcanic material, there are but rare outcrops to aid the geologist in outlining the various formations. The slopes are usually smooth, covered with a scanty growth of grass or of aspens, or with a mantle of slide rock, effectually concealing the contacts of formations.

The group of hills containing the mines of the Cripple Creek district are almost entirely made up of volcanic rocks, and the result of investigations proves that there is here a small vent, in and about which the sequence of phenomena has been remarkably complete and typical of a true volcano. The principal material is fragmental breccia, or tuff, resulting from a succession of explosive outbursts. Massive rocks occur, representing in part bodies that have been largely destroyed in succeeding explosive periods of the volcano, and in part intrusive bodies of various sizes and shapes belonging to later eruptions.

The formation through which the volcano burst, and upon which its superficial materials lie, is the granite-gneiss complex of the Colorado Range. Pike's Peak and the southern end of the Colorado Range are composed mainly of massive reddish granite, both coarse and fine grained. Southwest from the peak and toward Cripple Creek, bands of gneiss are occasionally met with, and the evidence that the gneisses are formed from the granite on zones of shearing is locally very distinct.

Extending from Cripple Creek to the Platte cañon on the north boundary of the Pike's Peak sheet, is a line along which the granite and derived gneiss contain numerous large and small included fragments of quartzite, quartz-fibrolite schists, or quartz mica schists, and other similar rocks. The igneous rocks of the Cripple Creek district are found resting upon or penetrating this complex of granite, gneiss, and included schist fragments, which are here micaceous or fibrolite quartz schist and much decomposed.

The basic dikes are usually inconspicuous on the surface, but are exposed in prospects and in the mine workings.

The Pre-Cambrian crystalline complex which forms the general plateau of the region was perforated by a volcanic explosion. The deep pit thus formed was filled with a breccia composed of fragments of phonolite and allied rocks, and of granite, gneiss, and schist. The walls of the pit are steep and rather irregular. They exhibit local bench-like flattening, and in some places overhang the breccia. While the breccia may in a few places rest upon an uneven surface of erosion, it occupies in the main a steep-walled chasm extending to unknown depth, and constitutes a typical volcanic neck. The breccia is cut by intrusive bodies of syenite and trachyphonolite, these two rocks showing frequent gradations from one to the other. The intruded bodies are usually of very irregular shape and have been peripherally shattered to such a degree that their contacts with the breccia are very obscure. These rocks generally form stock-like masses of thick, irregular sheets. The breccia is also cut by numerous dikes of phonolite, and this rock forms intrusive sheets and masses of considerable size in the prevolcanic crystalline rocks.

Ore Deposits.—The ores of Cripple Creek district are almost exclusively gold bearing. A little silver generally occurs in most of them, and in a few cases its quantity is sufficient to be of importance. The chlorination and cyanide mills do not pay for the silver in the ore. No other metals occur in any of the ores in quantities of commercial value. The ore consists usually of country rock, more or less impregnated with and replaced by quartz and other minerals, among which the most abundant are fluorite, opaline silica, and kaolin, with iron pyrites and other iron minerals, manganese oxides, and more rarely small quantities of galena, cerussite, anglesite, malachite, acanthite, tetrahedrite, stibnite, sphalerite, calaverite, native gold, oxidized tellurium minerals, gypsum, calcite, and numerous other minerals in still smaller quantities. The stibnite, the sphalerite, and the copper minerals, including tetrahedrite, are very rare.

The ores differ from those of many gold districts in often, but not always, consisting simply of country rock, either eruptive materials, or granite containing more or less secondary quartz and associated minerals, while in most gold districts, and in fact, in parts of the Cripple Creek district, the ore consists of well defined bodies of these materials. The characteristic ore of the district is an intimately mixed mass of quartz and purple fluorite, prominent from its purple color. The gold occurs in the ore as free gold, as tellurides of gold, and possibly as auriferous iron pyrites. In the veins the free gold usually predominates in the upper parts of the ore bodies, but within a depth of 100 feet, and in some cases much less, the telluride of gold associated with more or less iron pyrite usually appears, and gradually takes the place of free gold.

The average tenor of gold in the ores is about $30, or 1½ ounces per ton, and at various mines ranges from 1 ounce to 3 or 4 ounces. Ore at less than $12 per ton is rarely mined. Small amounts of ore up to 2,500 ounces gold have been mined. The gold in the Cripple Creek district is found in vein deposits, which generally occur in fissures in the country rock, which usually represents slight faulting, while more rarely they occur in other positions. The veins intersect all rocks in their course, and have been formed mostly by replacement along the fissures and not by the filling of open gaps.

The ore bodies were in almost all instances casually related to fissures. They comprise lodes or veins, and irregular replacement bodies usually in granite. These two types are not sharply distinct. All the ore bodies are characterized by the narrowness of the fissures and by the comparatively small volume of material deposited on these fissures.

Productive ore bodies occur in all the rocks of the district, with a possible exception of the schist. They are most abundant in breccia and in granite. Many lodes follow phonolite or basic dikes.

The structure most characteristic of the Cripple Creek lodes is the sheeted zone which occurs in various degrees of regularity, and in widths ranging from a few inches to over 100 feet. The fissures are usually very narrow, although there are a few notable exceptions to this rule. Evidence of tangential movement or faulting along the fissures is rare.

The fissures were probably formed about the same time

as the intrusion of the basic dikes, and represent a late phase of volcanic activity. They were probably opened under a relatively light load by local compressed stresses due to a slight subsidence of the solidified breccia and associated intrusive rocks forming the volcanic neck.

COST DATA

Labor.—The wage scale per 8-hour day is general throughout the district.

Hoisting engineer	$4.50
Blacksmith	4.50
Machine runners	4.00
Time keepers	4.00
Shift bosses	4.00
Watchman	4.00
Ore bosses	4.00
Timbermen	3.50
Machine helpers	3.50
Firemen	3.50
Trammers and muckers	3.00
Ore sorters	3.00

Freight Charges on Supplies.—Rates from Denver to Cripple Creek are:

	Per 100 Pounds
First class	$.80
Second class	.75
Third class	.70
Fourth class	.65
Fifth class (or carload lots)	.55

Below are some of the classifications as regards mining machinery:

First class: Steam or electric hoists (when less than carload); drill sharpeners; compressors.

Second class: Blacksmith blowers; ore buckets; blacksmith tools; blacksmith forges.

Third class: Ore cars; ore sacks; air receivers L. C. L.

Fourth class: Wire cable; drill steel; bar iron; sheet iron; pipe fittings; sledges and hammers; steel shafting; bolts and nuts, screws, washers, rails, fastenings, track bolts, etc.

Commodity rate ($.55 per 100 pounds), pipe.

Treatment and Freight Charges.—(1) Custom chlorination or roast-cyanide mills at Colorado City, based on payment of $20 per ounce for gold content.

	Freight and Mill Treatment Per Ton
Up to ½ ounce ore	$4.00
From ½ ounce to ¾ ounce ore	5.00
From ¾ ounce to 1 ounce ore	5.50
From 1 ounce to 1½ ounces ore	6.00
From 1½ ounces to 2 ounces ore	7.00
From 2 ounces to 3 ounces ore	8.00
From 3 ounces to 5 ounces ore	8.50

(2) Smelter rates, f. o. b., Denver, Pueblo, or Leadville.

	Per Ton Treatment Plus Freight
From 5 ounces to 7½ ounces ore	$5.50
From 7½ ounces to 10 ounces ore	6.50
From 10 ounces to 12½ ounces ore	7.50

Lumber.—The average price for lumber is $20 per 1,000 (board measure). More specifically, lumber for mine timbers is sold as follows:

Delivered	*Each*
8-inch stulls	$ .70
10-inch stulls	1.25
12-inch stulls	1.50
14-inch stulls	2.00
16-inch stulls	2.50
18-inch stulls (this size is used only in very wide stopes)	3.00
6-inch lagging	.55

The stulls are from 15 to 20 feet long. Purchaser pays the freight, which is about 15 per cent., and then deducts same.

Coal.—Steaming coal, $6.50; delivered, $6.75 per ton; blacksmith coal, $20.50; delivered, $20.75 per ton; lignite, about $4.

Electric Power.—Standard power rates (Central Colorado Power Co.):

	Per H. P. Year
First 100 horsepower	$39.00
Next 400 horsepower	27.00
Fixed Charge	
Next 500 horsepower	21.00
Over 1,000 horsepower	12.00
Energy or Consumption Charge	*Cents Per Kilowatt Hour*
First 40,000 kilowatt hours	1.3
Balance	.5

Schedule (December, 1908). Pueblo & Suburban Traction and Light Co., water power plants, Victor, Colo. Motors, 1 to 100 horsepower, alternating current. Single-phase 110 to 230-volts, direct current 500 volts.

A Fixed Charge of $1.50 per Horsepower per Month	*Per Cent.*
Plus 1½ cents per kilowatt hour, with a load factor of	10
Plus 1¼ cents per kilowatt hour, with a load factor of	20
Plus 1$\frac{1}{10}$ cents per kilowatt hour, with a load factor of	30
Plus 1 cent per kilowatt hour, with a load factor of	40
Plus 1 cent per kilowatt hour, with a load factor of	50
Plus $\frac{9}{10}$ cent per kilowatt hour, with a load factor of	75
Plus $\frac{8}{10}$ cent per kilowatt hour, with a load factor of	100

Other Supplies.—Cost to lessees or to small operators where large quantities are not bought:

Candles, per box	$ 3.50
Powder (40 per cent.), per 100 pounds	15.50
Fuse, per 100 feet	.65
Caps (per 100), per box	.85
Coal, per ton	6.50
Machine steel, per pound	.09
Making up drills (.20 for a drill, .10 for a shank), per drill	.30
Sharpening machine steel, per bit	.10
Assays (custom) three for	1.00

Water.—Water for domestic purposes of the miners is supplied by regularly organized water companies, which obtain the water from the melting snow of Pike's Peak. Water for the different mills of the camp, and for washing the ores at the mines, is supplied by the pumping operations in the lower levels of the deep mines.

Acquirement of Property.—It is assumed that the claims in question were acquired by location by the members of the operating company.

(*To be continued*)

DR. R. W. RAYMOND'S RESIGNATION

At a joint meeting of the Board of Directors and of the Council of the American Institute of Mining Engineers, Dr. R. W. Raymond tendered his resignation as secretary. He has been relieved from the many executive and administrative duties which have been increasing steadily, and which with the suggested extensions of the activity of the Institute must further multiply.

Doctor Raymond has been appointed Secretary Emeritus of the Council, the Institute thus retaining his services for special editorial and other duties for which he is so well qualified by training and experience. Doctor Raymond, one of the founders of the Institute in 1871, was elected vice president and became president in that year, following David Thomas, who resigned because his advanced years did not enable him to perform the duties of his office to his own satisfaction. He was annually reelected president in 1872, 1873, and 1874. In 1883 he was appointed secretary to succeed Dr. T. M. Drown, who had resigned. He was elected secretary in February, 1884, and has been annually reelected since that time.

Doctor Joseph Struthers, who has been connected with the Institute as assistant secretary for 8 years, and as editor for 5 of these years, has been elected Secretary of the Board of Directors and has been appointed Secretary of the Council. Doctor Struthers was editor of the *Mining Industry* for the years 1900–1903 (Vols. 8, 9, 10 and 11), and prior to that time was for 15 years on the teaching staff of the Department of Metallurgy of the School of Mines, Columbia University. He has been Field Assistant of the United States Geological Survey, in charge of the preparation of numerous reports for the Mineral Resources, and Special Agent of the United States Census, in a like capacity. In connection with the work of the Institute, he is a member and treasurer of the Board of the United Engineering Society, Chairman of its House Committee, and Secretary of the Library Conference Committee, which has general supervision of the libraries of the engineering societies in the United Engineering Society Building, 29 West 39th Street, New York City.

DRY PLACER MINING MACHINES

Written for Mines and Minerals, by E. B. Wilson

Knowing that there are placer deposits in the arid lands of the southwestern United States and the north and northwestern Mexican states, men have been working for years to devise machines that would treat the material successfully. One of the early difficulties was the necessity of having water for the steam power needed to excavate and treat sufficient quantities of ground to make the undertaking a commercial success. Some of the machines gotten up were elaborate; for instance, the Bennett dry-placer machine, in which a number of Hartford, Conn., people were heavily invested. Probably this machine would have worked successfully on a rich placer where there was plenty of water, but it never could work anywhere else to advantage. This is mentioned to emphasize that the phrase "dry-placer machine" does not necessarily make a dry-placer machine.

Different Methods That Have Been Used for Concentrating With Little or No Water

A fairly good machine that has some merit is known as Wood's dry-placer machine, Fig. 1. It was worked in Nevada on rich ground about as follows: The dirt was screened, as in Fig. 2, through one-quarter mesh sand screen, then placed in the hopper of a Wood machine. The upper part of the machine consists of a frame covered with heavy cloth across which are wooden riffles. This frame and cloth form the upper part of a bellows, and being set on an angle the puffs of wind cause the lighter particles to travel down the incline, while the riffles hold back the heavier particles composed of gold and black sand. When the riffles are full the frame is lifted out and the material caught dumped into a tub. After sufficient concentrate has accumulated in the tub, it is put over the machine and this second treatment holds the gold while the black sand becomes the tailing. The former is sacked and shipped, but the latter is panned in water before sacking. Fig. 3 shows an old dry placer once worked by Mexicans by tossing the dirt in the air

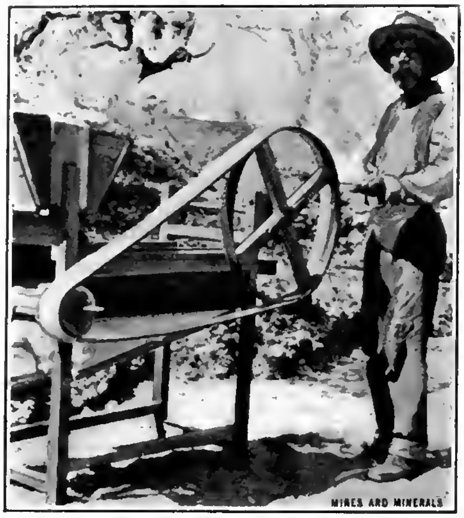


FIG. 1. WOODS DRY PLACER MACHINE

to allow the wind to blow away the dust from the heavier particles that fell on a blanket. The old placers discovered in 1828 by Mexicans at the mouth of Cunningham cañon, in Santa Fe County, near the town of Dolores, interested Thomas A. Edison some 20 years ago. He therefore constructed an extensive plant to recover the gold by means of static electricity, but this failed because the requisite dry gravels could not be obtained. About 1900 Mr. Edison again undertook to overcome the difficulties connected with dry-placer mining by means of the machine shown in Fig. 4. The device was described in the *New Zealand Mines Record* in 1904 as follows: "The revolving roller *b* discharges the gravel from the hopper *a* upon a shelf *c*, from which it falls into the air-blast created by the centrifugal fan *d* discharging its air through the screens *e* and *f*. The parting-board *g* divides the heavier portion of the gravel—the gold and iron or black sand—which falls into the

FIG. 2. SCREENING PREPARATORY TO DRY WASHING

chute *h* from the lighter portion falling into the tailing-chute *i*. The lattice *k-k* is simply to prevent eddy currents of air going down the chutes *h* and *i*. The end *m* is open. By a suitable adjustment of the speed of the fan, the position of the parting-board *g*, and the rate of feed of the gravel, a concentrate was obtained, the details of which are given below:

"Total gravel excavated, 33.8 cubic yards; weight, 137,464 pounds; total gravel treated in mill, 38,896 pounds. This was divided among the five sizes as follows: A, 8,911 pounds; B, 7,022 pounds; C, 8,040 pounds; D, 8,075 pounds; E, 6,848 pounds. The results are shown in the following table:

Size	Amount Treated Pounds	Weight of Gold in Concentrate Milligrams	Weight of Gold in Tailing Milligrams	Percentage of Gold Saved
A.........	8,911	513.2	147.40	77.7
B.........	7,022	1,737.5	66.65	96.3
C.........	8,040	3,478.0	75.70	97.8
D.........	8,075	3,667.0	340.46	91.5
E.........	6,848	2,542.5	26.41	99.0
Total....	38,896	11,938.2	656.62	95.0

"The concentrate is further treated in order to reduce its bulk and increase its value, by passing it over an Edison magnetic separator, which removes the magnetic black sand. Black sand is found almost without exception in all placer deposits, and being the heaviest constituent of the gravel, with the exception of the gold, it usually forms the bulk of the concentrate."

The conditions for the success of a dry-placer machine are: The operating must be cheap; it must have a large capacity; it must save a high percentage of gold; it must save all sizes of gold, from a few thousandths of an inch to the size of peas or larger. The necessary requirements for air separation by means of a blower appear to be: A free fall of the gravel through a horizontal current of air; a blast free from all puffs; a blast of perfectly uniform velocity throughout its entire cross-section; all particles of gravel and gold must enter the blast at a uniformly low velocity; and, the material treated at any one time must not vary in size beyond certain limits.

Machines of the semidry type are designed to work alluvials where water is insufficient for dredging. They are constructed on platforms provided with trucks to move on tracks under their

Fig. 3. Old Dry Placer in Altar Gold Field

own power. A dipper, or endless bucket chain digs and raises the gravel at one end of the platform. At the other end a tailing stacker disposes of the tailing. The machine is, in these particulars, similar to an ordinary dredge save that the buckets are lighter, this being possible because all work is done in plain sight and extra strains may be avoided; also, because bed-rock need not be dug into, as it may be more economically cleaned up by hand.

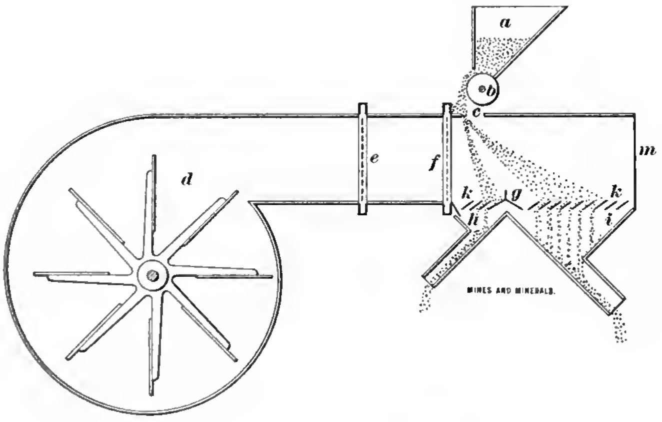


Fig. 4. Edison Dry Placer Machine

Separation is effected by the lifting force of water, driven by propeller blades. In this process it is necessary that a rising column of water shall lift all the sand and leave behind all the gold, however fine. It has been found by experiment on a commercial scale, that gravel, screened to $\frac{1}{4}$ inch, may be cleanly separated from its gold content. Therefore, all dirt is delivered by the bucket chain to a grizzly and the undersize to a rotary screen with $\frac{1}{4}$-inch mesh. The gravel is brushed by steel brushes over the grizzly for nine-tenths of the circumference. This arrangement is unnecessary, because the particles tumbling over each other clean themselves. If this is done under water, separation of coarse and fine gravel is very complete. The coarser material is carried to the tailing stacker after passing over a device wherein are saved any nuggets too large to pass the screen.*

The finer material is fed by gravity into a tank 7 feet in diameter and 5 feet high, containing water. In the middle of this tank stands an upright tube 2 feet in diameter. This does not reach to the bottom of the tank, so that water and sand have free ingress. Within the tube is an upright shaft, to which are fixed two sets of propeller blades.

When these propellers are made to revolve at the speed required, the water within the tube is forced rapidly upward. Its place is taken by water from the tank. The result is a hydraulic elevator fed constantly by water in the tank.† The force is so graduated as to raise sand and gravel, but not the gold, which remains at or near the bottom of the column, according to its fineness. Surrounding the top of the column is a launder into which the uprising water flows outward in all directions, and, losing force, drops the gravel and sand which it has lifted and which is automatically conducted to the tailing stacker. The very fine sand and mud in suspension do no harm if the water be kept sufficiently cleared by the addition of fresh water from time to time. Clayey gravels, and most of them are, require more water for treatment than clean gravels.

The machine described worked near Newhall, Cal., and handled gravel at the rate of 20 cubic yards per hour. Power was supplied by a 9-horsepower Fairbanks-Morse gasoline motor, and water was delivered by a 2-horsepower motor, pumping 700 gallons per hour from a 90-foot well through 1,200 feet of 2-inch pipe.

Often the impression prevails that there is no water in the dry-placer districts, a surmise which is correct, so far as surface water goes; however, if a well is driven in a "coulee" the chances for obtaining water are excellent, although the water may not be fit to drink. With a small quantity of water considerable work may be done where only the heavy sand is to be washed. The excavator and washing machine, shown in Fig. 5, is intended to work where there is sufficient water for the excavator boiler and the engine connected with the concentrating machinery. If, however, there is sufficient water, the sluice boxes to the rear of the concentrator can make use of it to advantage.

The machine was photographed near Quartzite, Ariz., where there is little water available. The excavator consists of a railroad crane having a 75-foot arm to which is attached a scraper bucket. This radius permits a large territory to be excavated before the machine needs moving. As seen in the illustration, the bed frame of the excavator rests on rollers, these on suitable rails and cross-ties so that it may be moved as desired. The gold-bearing dirt is about 8 feet deep before bed rock is reached, and it is said that the first three yards delivered to the hopper produced $1.23 in gold.

The bucket is arranged both to dig into and scrape up the earth, even close to bed rock, hence little hand labor is needed

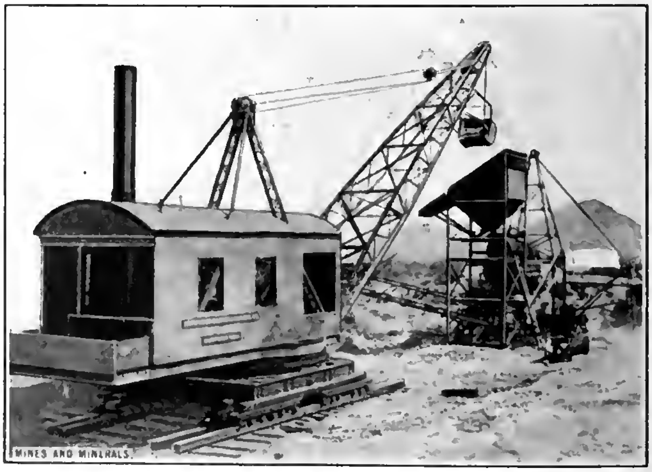


Fig. 5. Excavator and Washing Machine

in cleaning up the gold on bed rock. It has a capacity of $1\frac{1}{2}$ cubic yards, which makes it possible to deliver a large quantity of dirt to the concentrating hopper during the course of the day.

*Gold Dredging, page 146.

† Those acquainted with the agitation of gold slimes in cyaniding will notice a similarity between this apparatus and some agitators.

To the rear of the excavator is seen the concentrator mounted on the steel frame. At the top is a steel hopper containing a grizzly that terminates in a chute for the disposal of the coarse rock that passes over the bars. Below the hopper is a rotary screen which sizes and dry-washes the dirt that passed through the grizzly. The coarse material from this screen passes to the traveling-belt tailing stacker shown to the left of the tower, and the fine material that goes through the screen is washed through the sluice extending from the rear of the tower.

Where a dry placer is worked there is usually clay baked to the gravel, and to separate this there should be some kind of scouring action, such as is given by screening. When the gravel is almost cemented with clay the Quenner dry pulverizer, described in the November, 1910, issue of MINES AND MINERALS, will be found useful, as it is run by gas or oil engine power and pulverizes the material sufficiently to liberate any gold. The introduction of gas and oil engines will aid materially in working dry placer deposits, and as the conditions for recovery are better understood from experience it is probable that much gold will be won; however, a good many schemers will work out plans for separating the unsuspecting avaricious from their money by using the phrase "dry-placer machine."

HAND JIG FOR SMALL MINES

*By J. M. Calderwood, M. I. M. M.**

For the coarse concentration of ores containing lead, zinc, copper, tin, and iron, jigging machinery is universally employed. Although a great variety of designs are in use, they are practically alike in principle, whereby the separation of the mineral from the gangue takes place on a screen or bed of material, the ore particles arranging themselves into layers according to their specific gravity. This is effected by an up-and-down current of water in the jig box, the pulsations being imparted by a vertically operated plunger. All jigs are dependent for good work on approximately sized material, based on the character of the ore, and a suitable adjustment of speed and stroke of plunger for the class of material worked. Mechanically operated jigs make clean concentrates when treating properly sized material, and, in addition, a middling product which is generally treated by regrinding machinery and concentrating tables.

In districts where water is scarce and the scale of operations does not permit of a large expenditure, it is possible to produce small quantities of ore which would yield a shipping product, if suitable appliances could be installed. For this purpose a jig operated by hand has proved its usefulness, especially in the treatment of lead, zinc, and copper ores. With sulphide ores, good results can be obtained, provided the ore is fine enough and several sizes carefully screened for treatment. At the Messina copper mine, in the Northern Transvaal, a number of hand jigs were installed when output was commenced in 1906, and, although a large concentration plant has since been erected, these small jigs are still in use for working up jig middlings, retreating Wilfley table product, cleaning concentrate, etc.

Until the concentration plant was built the method of preparing the copper ore for shipment was as follows: The ore from the mine was trammed to the sorting tables, cleaned by a water spray, and the larger pieces of copper ore handsorted and bagged for shipment. The residue from the tables was then wheeled to the screens where several sizes were made and delivered to the hand jigs arranged in rows. Fig. 1 gives sufficient detail to show the construction of these jigs, which were cheaply and easily made, generally from bush timber and old packing cases. As will be seen from the sketch, the ore box, having a screen bottom, is moved up and down in the water tank by means of the lever, which is given a quick down stroke with somewhat slower recovery, and just sufficient play or loose motion to give the required movement. The work is soon learned and carried out in an expert manner by young native boys, and even the smallest piccanins employed become quite skilful in turning out clean concentrate and separating the tailing. The box is filled by shovel, and a few minutes jigging is sufficient to allow the heavier mineral to settle on the screen and the tailing to collect on top. The top layer of tailing is first removed by means of a tin scraper, and then the concentrate. The concentrate is not removed each time the tailing is scraped off, but allowed to accumulate until in the judgment of the operator there is sufficient accumulation of clean product, which is then dried and bagged for shipment. The tank or hutch is kept full of water and the fine material and slime settles in the bottom and is emptied as often as necessary by removing the screen. The quantity of clean water required is small, and is chiefly to make up for the loss by splashing and for refilling the tanks. The fines can be retreated in another jig having a smaller mesh screen if of sufficient value. The mesh of the jig screen depends upon the size of the screened material treated, and the proportion of fines desired. For coarse ore, $\frac{1}{2}$-inch aperture iron wire screen cloth is used, and for finer material $\frac{1}{4}$-inch to $\frac{1}{16}$-inch screen cloth, although for close work even finer screens may be used. The capacity of the hand jig depends upon the class of material treated and the grade of ore. From $\frac{1}{2}$ ton to 1 ton of material

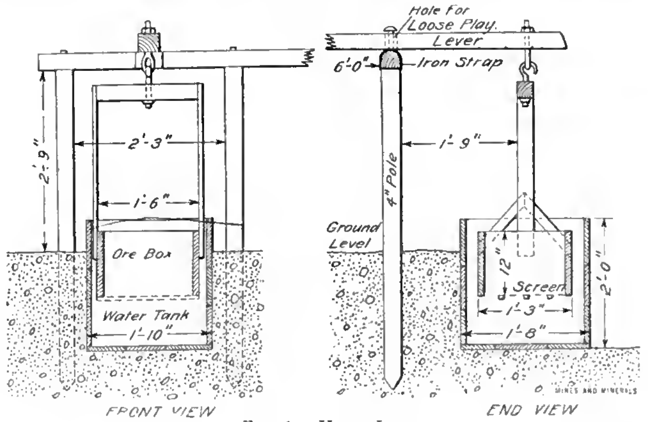


FIG. 1. HAND JIG

per day can be handled by each jig, according to the grade treated and the experience of the operator.

The results obtained compare very favorably with the word done by mechanical jigs, as regards the value of the concentrate and tailing produced. The initial outlay is small and the cost of operating a battery of hand jigs is merely a question of native labor, repairs to screens, etc., as constant white supervision is not necessary with a good gang of trained piccanins. The operation of the jigs may appear to be tedious, and it certainly is when compared with the work done by the large mechanically operated jigs, whereby clean concentrates are automatically turned out and the tailing passes off without handling, but their importance and usefulness has been recognized and employed on many occasions where conditions have been feasible for their adoption.

A paper by F. W. Hinrischen relates to investigations as to the cause of, and means of prevention of, accidents arising from the poisonous and explosive gases evolved from ferrosilicon. These gases are chiefly phosphoretted hydrogen derived from the calcium phosphide. Some samples also contain arseniuretted hydrogen. The most dangerous grades are those containing from 30 to 70 per cent. of silicon. This article is well worth careful consideration by manufacturers, carriers and users of electric ferrosilicon. F. W. Hinrischen. (Mitt. Kgl. Materialprufungsamt, xxviii, 283.)

* Reprinted from *South African Mining Journal.*

THE SOUTH UTAH MINE AND MILL

Written for Mines and Minerals, by Leroy Palmer

Description of the Methods Employed in Mining, Milling, and Handling the Ore

The South Utah mine, better known as the Cactus or the Newhouse, has had a checkered career. It was first exploited by a French company which worked it to the 100-foot level and built a small concentrator. It proved unprofitable, failed to meet the bond requirements and was finally sold under the hammer. It then fell into the hands of Samuel Newhouse, of Salt Lake, and his associates, who organized the Newhouse Mines and Smelters. Extensive equipment was installed, including a 600-ton concentrator, and it entered its second epoch. While copper was at its highest price it paid two dividends aggregating $600,000, but with the slump soon found itself unable to meet the interest on the bonds, and it went through a reorganization process from which it emerged as the South Utah Mines and Smelters. While the reorganization was going on the mill was remodeled and the mining system materially altered, so that when the property was started under its third management on September 1, 1910, the operative methods were as different from those employed by the second management as these were from the first.

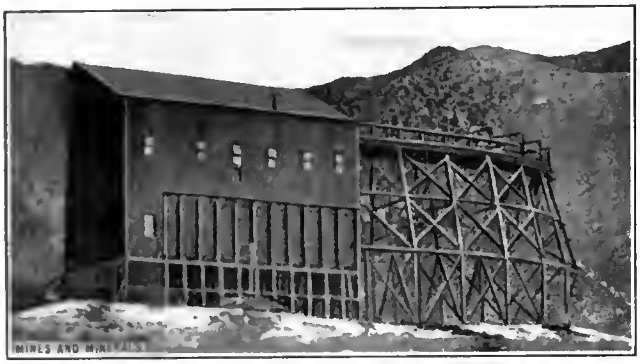


Fig. 1. Trestle and Ore Bin, South Utah Mine

The location is on the west side of the San Francisco Mountains 1½ miles from the town of Newhouse, and 4½ miles northwest of Frisco, the site of the Horn Silver mine. The San Francisco range is divided into three massifs, the southern of which, known as the Grampian Hills, consists mainly of blue lime. The Horn Silver fault runs along the base of these hills. In the saddle between the southern and the central massif is seen a fault which branches from the Horn Silver fault and shows monzonite on the north and blue lime on the south side. The central massif extends to the north to Copper Gulch, which cuts across the range in a northwesterly direction. The higher portion of this massif is composed of limestones overlain by quartzite, a body of monzonite being intruded into the limestones which have been extensively altered, and which show on the contact garnet and the lime silicate minerals often found accompanying contact metamorphism. An extensive outcrop of smaller rock, but somewhat more of a dioritic nature, is found toward the north of the central massif and was evidently part of the same intrusion; it is in this rock that the deposits of the South Utah are found. On the northern of the three massifs a fault exposes a face of rock which gives some idea of the sedimentaries, an upper layer of red quartzite about 1,000 feet thick, several hundred feet of white quartzite, and beneath, limestone of undetermined thickness. A detailed study of the district has never been made and no definite conclusions have been drawn as to the age of the rocks, but the intrusion of monzonite occurred after the sedimentary depositions.

The ore occurs in the monzonite in a zone of fracture and brecciation of irregular width, which follows the general direction of Copper Gulch, averaging N 30° W. The dip is practically vertical, being only slightly to the northeast. An interesting fact about the discovery is that the only surface indications were a slight copper stain, but sulphides were found in a winze when it had been sunk but a few feet below the surface. This is supposed to be on account of the disintegrated nature of the ore zone allowing erosion to take place as fast as oxidation and thus keeping the unoxidized portion near the surface. The width of the zone of faulting is from 15 to 30 feet, but the ore deposits are in some places much wider. There are no well defined walls, but instead rather indefinite boundaries between the mineralized and barren portion. There are some evidences of cross-faulting, and in one place is a long slip on one side of which the ore is very hard and firm and on the other comparatively soft and crumbly.

Rounded boulders of country rock have been found in the upper 100 feet of the vein, and this was at first supposed to indicate that the deposit filled what was once an open fissure and that these boulders had fallen into it. Later belief is that they are fragments from the walls that have been rounded by the action of the circulating waters. The solutions ascended through this shattered zone and in the cracks and tiny openings deposited pyrite and chalcopyrite, for the most part of very low grade, but in some portions there was a natural concentration which resulted in the deposits of high-grade chalcopyrite which has been shipped in small quantities from time to time. In some portions erosion did not keep pace with oxidation and here were found small amounts of native copper and carbonate, but in the main the sulphides were found from the surface.

The French company sunk a single-compartment shaft to the 100-foot level and worked through that, the Newhouse company sunk this shaft to the 600-foot level and drove a tunnel 6,600 feet long to it at this point. The shaft caved above the 100-foot level, so the hoisting machinery was moved inside the mine to the 200-foot level and operated from there. In the later years of the Newhouse company an incline was sunk from the 600-foot to the 900-foot and levels driven every 100 feet. As a water supply, Wah Wah springs were purchased, the water from them collected in a 36,000-gallon reservoir and carried in 8 miles of spiral-riveted pipe to a 300,000-gallon reservoir above the town.

Under previous management a system of square sets was used, and later a large tonnage was mined by stripping the surface with a steam shovel and working by the milling system, two large raises being driven for this purpose. The square-set timbering, when installed in one system, was not sufficient in itself to hold the wide vein, so pillars of ore of a thickness averaging 35 feet were left across the entire width. This left a considerable tonnage of ore which the present management desired to recover. To do this, short drifts, usually not over 10 feet long, were run into the pillars all around them at intervals of 20 to 60 feet, as the ground would stand. The pillar was then undermined and came down of its own weight, the ore running out of the short drifts. The mouth of each drift is over a "finger chute," so called because the chutes from one pillar run together and thus bear a rough resemblance to the outspread fingers of a hand. These chutes lead frequently by way of an old stope, to two large chutes installed by the previous management to the 600-foot or haulage level. In this way all of the ore in the old pillars is drawn off to the haulage level by gravity. The work is contracted at the rate of 10 cents per ton and one man handles 30 to 40 tons per shift. As the ore is by nature blocky and breaks in large pieces, the mouth of each finger chute is protected by a timber grizzly with 8-inch openings. The larger pieces are sledged or plugged with a small air hammer drill and shot.

In case the pillar does not break readily when it is undermined by the drifts, one of them is continued well into it, with

the uppers pointed up, and continued until the pillar shows signs of weakening, when it is left to break of its own weight.

Between the two "glory holes" which were used when the ore was being taken out by the milling system there were two large bodies of ore which it was desired to recover. The 250-foot level was chosen to work from, as it presented the most firm formation in the upper portion of the mine. Short cross-cuts were driven into the ore bodies from all sides, and from the ends of these, raises inclined toward each other were driven. As soon as the softer ore was reached the pillar commenced to work and the ore was run over grizzlies and into chutes as described.

Below the 600-foot level only a little square setting had been done, and a system of back stoping was introduced. The ground is blocked off into stopes and pillars, the size being varied to suit the nature of the formation, an average width being 10 to 12 feet for the pillars and 20 to 30 feet for the stopes. The length of the stope is the full width of the deposit, which may be anything from 100 to 200 feet. The pillars and stopes are staggered on the different levels; that is, a pillar put over a stope on the next lower level and vice versa. A 5′×7′ drift is driven the full length of the pillar, and at intervals of about 30 feet, also suited to the nature of the ground, inclined chutes are driven into the portion to be stoped, these chutes varying in length from 5 to 10 feet. From one of these chutes a short drift is driven to the center of the proposed stope, then turned at right angles and carried to both walls, thus extending the entire length of the stope through the center. This drift is then widened the full width of the stope and the first slice in the extraction of ore has been made. A raise is put up to the next level from each stope and the air pipes carried down them. As the stope fills, these pipes are taken off from the bottom so that no pipe is lost, and the raise provides a means of exit. When a slice has been carried across a stope, a sufficient amount of ore is drawn off through the chutes to allow the men to work the roof by standing on the remainder; uppers are put in with stoping drills and the roof blasted down. Only sufficient ore is drawn off to allow of convenient working space between the remainder and the roof, and the process is repeated, the stope being always nearly full of the broken ore. At present all of this is allowed to remain in the stopes, but as the ground above is worked out it is to be drawn off, the numerous chutes permitting it to be taken out gradually so as to allow a uniform settling of the ground above. To go into one of these stopes, 200 feet long, 20 feet wide, and nearly 100 feet high without a stick of timber gives one the first impression that such a system of mining must be precarious, but the formation is firm, giving a good wall and roof, and accidents are exceedingly rare. The ore that is drawn from the chutes is hand trammed to the incline and dumped to a 150-ton ore pocket from which it is loaded to 1½-ton skips, hoisted to the 600-foot level, and dumped to the cars. By this system of stoping 12½ tons of ore are taken out per shift to each miner and mucker, and 5½ tons to each man engaged in mine work, including office force, engine crew, etc. A little water is encountered in the lower levels of the mine, but not an amount which makes pumping a serious problem, a small electric sinking pump raising it to the 700-foot level where a 3½″×6″ outside center-packed duplex Worthington electric pump raises it to the haulage level.

The ordinary working drifts are made 5 ft. × 7 ft. They are driven by putting in a round of from 9 to 20 holes (some of the ground is very hard) with a piston drill. Both the single and double machines are used, the average progress being 3½ feet with 2¼-inch machine, and a foot more with a 3-inch. Hammer stopers are used with which one man can break 15 to 20 tons to a round, not bad work for the hard ground that must be drilled. Very few timbers are used in the drifts, only an occasional set of 10 in. × 10 in. being necessary. The stopes have no timbering and all required in the chutes are two 10 in. × 10 in. × 9 ft. long, with a 7-foot cap and two 6 in. × 10 in. × 4 ft. for braces at the mouth. The vertical shaft is driven in ore with one compartment 4 ft. × 4 ft. The hoist was operated by compressed air, but when it was moved to the 200-foot level a 52-horsepower variable speed 440-volt induction motor was geared to it. The incline is driven in the vein with a pitch of 50 degrees and with 8-foot pillars of ore on each side. It has one 5′×5′ compartment with a manway of the same size and is timbered with ten 10″×10″ sticks with 2-inch lagging. The hoist is similar to that used at the vertical shaft, and the skip has a capacity of 1½ tons.

The tunnel is 6,600 feet long and 6 ft. × 8 ft. in cross-section. It is driven through a lime formation which requires little timbering and a roomy station is cut at the shaft. Near the shaft it runs through the ore for a short distance, but not so far but that it can be shifted to the lime with small expense when the time comes for the extraction of this ore. The haulage road is laid with a 36-inch track over which the cars are hauled in trains of 14 by a Baldwin-Westinghouse locomotive. The gates at the main chutes underground are operated by air.

The coarse-crushing department is located at the mine, Fig. 1. Above the bins, which have a capacity of 600 tons, is a dumping cradle, shown in Fig. 2, capable of holding seven cars. The cars are run into this cradle in which they are held closely by means of angle irons which fit over the upper edge of the car body. The cradle is connected to an air-operated 7″×12″ duplex hoisting engine by means of a ⅝-inch cable which makes

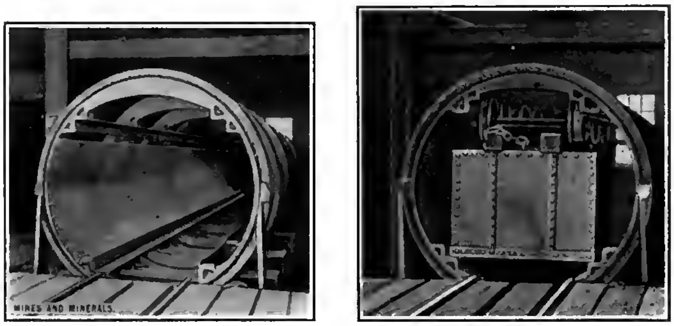


Fig. 2. Rotary Dumping Cradle

two turns around the cylinder and is attached to the drum of the hoist. By this the cradle is revolved and the seven cars dumped at one operation. Fourteen of the 4-ton cars can be dumped in 5 minutes, making all allowance for switching. The bin is fitted with a 2-inch grizzly of railroad iron from which the coarse ore is fed by hand-operated gates over 2-inch punched grizzlies to three 20″×10″ Blake crushers making 235 revolutions per minute and discharged back to the bin beneath the grizzly, from which it is unloaded through sliding gates in the flat bottom to the railroad cars.

The crushers are driven by a 75-horsepower induction motor which receives its current direct from the transmission line at 2,000 volts. There are also two generator units in which a 100-horsepower, 2,000-volt induction motor is direct-connected to a 75-kilowatt, 250-volt, direct-current generator for the haulage system, and in Copper Gulch, near the old shaft, are three 75-kilowatt Westinghouse transformers, stepping down from 2,000 volts to 440 volts for use in the mine. A manway has been kept open in the old shaft and the wires lead down it.

The mill, shown in Fig. 3, is a mile and a half from the mine, on the outskirts of the town, situated on comparatively level ground. The topography of the site is not all that might be desired, as it makes it necessary to elevate the ore more than had greater slope been obtainable. Such a site could have been obtained by building at the mine, which would also have dispensed with the railroad haul; but opposed to this was the fact that at the present site the water can be delivered by gravity, whereas, if the mill and town had been situated at the mine

there would have been a continuous expenditure of about 175 horsepower in order to raise mill water against a head of 600 feet. The saving in continuous pumping compared with hauling the ore down grade outweighed the disadvantages of the lower site on which the designer decided to build.

The ore is loaded at the mine to 50-ton cars with side dump and hauled in trains of four to the mill. This is approached by a trestle 750 feet long, the last 75 feet of steel. The ore is delivered at the top of the mill to the bin built of steel with a capacity of 1,200 tons and with bottom sloping toward both sides from the middle to deliver ore to both sections of the mill. This capacity is not sufficient to provide any adequate reserve supply of ore, so that an accident which shuts the mine down for any length of time must necessarily put the mill out of commission.

The mill is built in two units, and while these differ in some minor points, the general scheme is the same, so that only one will be described. The present capacity of a unit is 400 tons, but plans are being considered for modifications, which it is hoped will increase this to 500 tons. Each bin is equipped with four plunger feeders, two of which are operated at a time to feed the ore to a 24-inch rubber belt conveyer. This conveyer is fitted with an ingenious device in the form of a powerful magnet which is suspended just above it to pick out pieces of steel, such as hammerheads, moyles, drills, etc., which have gotten into the ore, and prevent their going into the rolls.

The conveyer discharges to elevator No. 1, which is run dry, has an 18-inch belt, a lift of 70 feet, and a speed of 460 feet

Fig. 3. South Utah Mill

per minute. It raises the ore to the top of the mill and discharges it to a 42″×60″×60″ conical trommel with 15-millimeter punched openings. The oversize of this trommel goes to a 24″×24″ chute made of 2″×8″ timbers laid crib fashion, and discharges to the coarse dry rolls on the main floor of the mill. These are 15 in. × 36 in., make 68 revolutions per minute, and are set to ½ inch. The rolls discharge to No. 1 elevator and the ore goes again to the trommel. A small Gates crusher with a 2-inch punched grizzly has been installed between the trommel and the rolls, the oversize of the grizzly going to the crusher which discharges to the rolls. But at present the ore is soft and the second crusher is not in use. The location of the rolls is not the best that can be chosen, as all oversize is dropped clear to the main floor and raised to the top of the mill a second time. Plans are being made to place these rolls directly below the trommel and discharge to the elevator at that point so that the lift of the return may be materially lessened.

The undersize of the first trommel goes to a double conical trommel the same size as the first, which is run wet. The inner screen has 9-millimeter punched holes and the outer 3-millimeter punched. The oversize of the 9-millimeter goes to the fine rolls, which are duplicates of the coarse, but run wet, with a speed of 93 revolutions per minute, and are set to ¼ of an inch. These discharge to elevator No. 2, which is a duplicate of No. 1, but run wet. The elevator dumps to a conical trommel with 3-millimeter perforations whose oversize goes back to the fine rolls and undersize to the double-conical trommel. The oversize of the 3-millimeter screen on the double trommel goes to a double bull jig with 1½-inch stroke, making 130 revolutions per minute. This jig makes a finished concentrate through a side-discharge gate, a coarse middling over the tail-board, which is sent to elevator No. 2, and a fine middling through the hutch, which is sent to elevator No. 3. The undersize of the double trommel goes through the screen line.

Here are two sets of two, and one set of four, trommels, each trommel 42 in. × 72 in., making 10 revolutions per minute. Each set is driven independently of the others by its own belt and each trommel is controlled by a clutch. All have punched steel screens, the finest being slotted. The first set has 2-millimeter openings; the second 1½ millimeters, and the third ¾ millimeter. The oversize of the first set of trommels feeds three three-compartment and three two-compartment Hartz jigs; the oversize of the second set feeds four two-compartment jigs, and the oversize of the third set (four trommels), feeds four three-compartment jigs. The stroke varies from 1½ inch on the bull jigs to ½ inch on the finest, and the speed from 130 to 180 revolutions per minute. All of the fine jigs make a tailing rejection, the three-compartment jigs make concentrate on the first and second compartments, and middling on the third, and the two-compartment jigs make concentrate on the first and middling on the second compartment. All work entirely through the hutches.

The jig middling discharges to a very simple but satisfactory dewatering device consisting of a trough set at an angle of 10 degrees with the horizontal, in which is a 16-inch belt, with scrapers bent on each end. This discharges to elevator No. 3, which has an 18-inch belt, a lift of 40 feet, and a speed of 550 feet per minute. It discharges to four 42″×72″ trommels with 1-millimeter slotted screens, the oversize of which goes to a slope-bottom bin, and the undersize is divided among the ¾-millimeter trommels in each screen line. The four 1-millimeter trommels and the slope-bottom bin are used for both units.

The bin feeds two 6-foot Walker-Huntington mills with 1-millimeter screens, making 72 revolutions per minute. A third Huntington is installed as an auxiliary. The Huntington discharges to the No. 3 elevator on its side of the mill, which raises the product to the 1-millimeter trommels.

Table System.—The undersize of the four ¾-millimeter trommels goes to a distributing tub which feeds four three-compartment classifiers which are in general principle similar to spitzkasten. There are 20 No. 3 Wilfley tables in two rows, making 240 strokes per minute, and two classifiers feed each row. The first compartments of each two classifiers originally fed two three-compartment jigs, but this was found unsatisfactory and they were cut out. Considering one pair of classifiers and one row of tables, the second compartment goes to a distributing tub which feeds four tables and the third compartment to a similar tub which feeds three tables. The overflow of each classifier goes to a settling tub, the spigot discharge of the two feeding three tables and the overflow going to eight Callow tanks. The spigot discharge of the Callow tanks is carried to the lower floor and the overflow is used as table water. The first seven tables make concentrate, middling, tailing, and slime, and the last three concentrate, middling, and slime.

The concentrate from the jigs and the first five tables in each row is run through the floor directly to 1-ton cars which are drained and hand trammed to the railroad cars outside. The concentrate of the last five tables in each row discharges to boxes from which it is shoveled to small cars and trammed to the railway cars.

Lower Floor.—On the lower floor are six No. 5 Wilfley tables with elevators and three rows of five Johnston suspended vanners. Each Wilfley treats the middling of two of the tables on the upper floor, making concentrate, tailing, and middling, the latter being retreated on the same table.

The vanners make 120 strokes per minute. The middling of the seventh and eighth tables in each row above goes to a

distributor which feeds three vanners, and the middling of the ninth and tenth tables goes to a distributor which feeds the other two vanners in the same row. Each of the other two rows of five vanners is fed from a distributor which receives the settled pulp from the eight Callow tanks which accompany each row of tables. The table concentrate goes to individual boxes at the tail of each table and the vanner concentrate by a launder to one box. All concentrate is shoveled out and trammed to the railway cars.

All slime is collected in two 3,000-gallon tanks from which it is removed by a 6-inch centrifugal pump. Two pumps are installed with common intake and discharge, and one is held in reserve. The pump discharges to a 3,500-gallon settling tank with cone bottom, the settlings going to the four $\frac{3}{4}$-millimeter trommels on the side of the mill nearest the tank and the overflow to five sets of 4,000-gallon tanks. Each set consists of three tanks, the middle one being set $\frac{1}{2}$ foot above the others. The middle tank of each set receives the slime and subjects it to a partial settling, the overflow going to the tank on each side, in which it is further settled. The overflow of the outside tanks is clear water which is laundered to a 16th tank and used on the jigs. This tank also receives the water from the compressor jackets.

The tanks are tapped through goosenecks from the bottom, and the thickened slime divided between the two units, mingling with the overflow of the settling boxes below the classifiers, and thus going to tanks to be further settled and treated on the vanners.

Skimmers are set in all of the tailing launders and as much water as possible taken off, only an amount sufficient to carry off the tailing being left. This water goes to a sump in each section which also receives the drainage from the concentrate cars and a 4-inch centrifugal pump raises it to the slime tank on the lower floor. A duplicate pump is installed in each unit and held in reserve.

Three-phase, 60-cycle, 440-volt Westinghouse induction motors are used throughout. The mill is very flexible, each group of machines being under the control of a clutch, and each individual machine being subjected to control by a clutch or a loose pulley.

Following is the motor equipment for the entire mill:

Machines	Number Motors	H. P. Each	Total H. P.
4 rolls, 28 jigs, 6 elevators	2	20	340
2 conveyers, 24 trommels	2	150	
52 Wilfleys, 30 vanners	2 4	20 5	60
2 Huntington mills, 2 dewatering belts	1	75	75
1 6-inch slime pump	1	40	40
2 4-pound return water pumps	2	30	60
Total horsepower			575

The total 575 horsepower divided by 800 equals .72 horsepower per ton of ore treated per day. This is a very low figure, especially considering the elevating of water and ore that is necessary.

Hydroelectric power is purchased from the Beaver River Light and Power Co., and received over a transmission line 60 miles long at 44,000 volts. It is delivered to the basement of the power house where there are four 400-kilowatt Wagner transformers, one of which is held in reserve, stepping down to 440 volts. The current for the mine is stepped up from 440 to 2,200 volts in two 125-kilowatt Westinghouse transformers, a third being installed in the set as a reserve. The electrical equipment at the mine has been noted. The transformer room is equipped with three 44,000-volt multi-gap lightning arresters and three choke coils.

The power house was originally equipped with Babcock & Wilcox water-tube boilers, Westinghouse-Parsons steam turbo-generators, and steam-driven air compressor. None of these is in use since the connection with the electric power except one Ingersoll-Sergeant two-stage, piston-inlet compressor with a capacity of 2,000 cubic feet of free air per minute. The valves have been taken out of the steam cylinder of this and the flywheel belted to a 300-horsepower Westinghouse induction motor. The air is taken to the mine in a 6-inch line.

Owing to slight changes being made in the mill from time to time and that pending the final adjustments the lower grades of ore are being worked, a statement of the cost of production of copper is not advisable at the present time. The mining system seems to be a desirable one for the ground encountered, as it is cheap and safe, accidents being of rare occurrence. Mining costs are in the neighborhood of 90 cents per ton on a daily output of 800 tons. Miners are paid \$3 to \$3.25, and muckers \$2.50 to \$2.75. No timbermen are employed. The mill provides for a thorough working of the ore, but the equipment has been so modified since it was first installed that it does not suit the building and is the cause of some inconvenience. Mill men are paid \$3 to \$3.25 per day, and milling costs about 50 cents per ton.

During the month of January the mill treated 22,090 tons of ore, a little more than 700 tons per day. Concentration was at the rate of 10 tons into 1, 2,091 tons of concentrate being shipped. The extraction approximates 70 per cent. and the value of the concentrate 10 per cent. copper with 1 to $1\frac{1}{4}$ ounces silver, and \$1.25 to \$1.50 gold. During December and January the production was 380,000 pounds per month, but this will be increased as the better ore is treated. The figures given above show the very low grade of ore being milled.

FIG. 4. CACTUS CLUB

A considerable dump of tailing has accumulated and as the winds are high in Wah Wah valley they have caused a natural concentration of the surface so that for a few inches in depth an average of several samplings shows considerably better than 1 per cent. copper. A 24-inch haulage road has been laid to the dump and 1-ton cars can be hauled to the mill by means of a small hoist. In this manner 100 tons of ore already crushed can be delivered per day.

Newhouse was the first "company town" built in Utah, and was made necessary by the fact that the mine is situated at the edge of a desert basin which has no other interests. The town contains 42 cement-finished three- and four-room houses, 30 two-room houses, two boarding houses, cement store building, club house, and opera house. Employes of the company are eligible to membership in the Cactus Club, Fig. 4, upon payment of an initiation fee of 50 cents and monthly dues of 25 cents. This gives the privileges of the reading room, piano player, billard room, and bar. Out of its profits the club has built one of the boarding houses and the opera house. A free dance is given every week and other entertainments from time to time. It has done much to make the place, which is in an out-of-the-way spot, attractive to those who are called upon to live there.

The future of the company seems to depend on its being able to handle a considerable quantity of a reasonable grade of

ore. The mining and milling can be done cheaply, but on the ore so far treated the most economical management could hardly more than break even. So far only the lower grade has been treated, as it has been desired to hold off the better until the mill is in shape to do its best work. The entire force in mine, mill, and office for the present output is 225 men, which compares with 400 under the method of square-set mining. If the mill will stand it, 1,000 tons can be treated per day with only a small increase in the force, and it would seem that a good profit should be made on 2-per-cent. ore.

The writer wishes to acknowledge the courtesy of E. H. Lundquist, general manager, Rollin Madison, mine foreman, and Joseph Bruner, mill foreman, as well as all other employes from whom information was sought.

CANADIAN MINING INSTITUTE MEETING

The thirteenth annual meeting of the Canadian Mining Institute, held at Chateau Frontenac, Quebec, March 1, 2, and 3, was an intellectual function, with an elaborately staged and strictly adhered to program. Probably all members have received or read perfunctory accounts of this meeting; therefore these, our impressions, are indited to those unfortunates who missed connections. We missed connections for Quebec at Montreal, and then a strange thing happened, unheard of in the states; namely, the station agent had our ticket transferred from the Grand Trunk to the Intercolonial, so that Quebec was reached in daylight. Expecting to cross from Levis to Quebec in a sleigh or ice boat we were disappointed to find less ice in the St. Lawrence than in the Hudson at Albany. During the day we were present at a dozen or so wedding parties, probably missed as many more, and were coming to the conclusion that Quebec was the most marriageable place on earth, when some one said it was the last day for weddings.

The people of Quebec, anticipating our arrival, held their first "Mardi Gras" on Tuesday evening; at least this was the impression, for most all the townspeople were in the parade, and we were the observers.

On the morning of Ash Wednesday the members were welcomed to Quebec by the authorities, requested to ride on the street cars free, and to become guests at the Garrison Club; all of which courtesies were accepted with pleasure, although it is feared the members lacked time to fully appreciate them. After listening to President Adams's pleasing address, the opening session adjourned until evening.

Wednesday afternoon the members went to Montmorency Falls in a snow storm and found the falls frozen inactive; this, however did not reflect on the excursion committee's ability to pack us in a three-story lean-to and have us lifted 250 feet perpendicular at an angle ± 100°. Arriving at the top, the committee introduced us to Foxen and Polar bears, but not to the rosy cheeked damsels who were snowshoeing. Then, thinking some in the party were not keeping Lent, ushered us into an excellently appointed club house, where they warmed the cockles of our hearts. To return to Quebec it was necessary to take that three-story house with us, and this was worse than going down a 1,200-foot shaft in 12 seconds, on account of our anxiety for the safety of the ladies in case the rope broke. The following day was devoted to business, and the night to funny business.

Professor Kemp gave an illustrated lecture on glacier climbing. It is not exaggeration to state, after seeing his illustrations, that with "suckers" on his feet we believe he could outclimb Cook and beat a fly walking on the ceiling. The man who believes photographs are truthful reproductions should listen to Professor Kemp's lecture on agriculture and land agents. While his remarks were true of some land agents, his illustrations were preposterous. After some singing, Mr. Smith attempted to recite "Hetty McEwen," but either he forgot his lines or he is too emotional for such staging. Doctor Day took his place and sang 20 verses of "I Gave Her Kisses One, Kisses One, So I Kept a Kissing On." Just before Mr. Miller was sued for breach of promise some one had the hardihood to sing "Landlord." It was quite sad to hear the lawyer for the Crown malign, and the jury fine, Mr. Miller; but if any one thinks all the grafting is done in the United States he should have heard Justice Penhale settle the dispute over the fine. The breach of promise did not get a thing except a glass of beer every time she fainted, which was quite often, unless it was a revengeful thrill at the way the culprit was abused. Friday, in addition to the regular program, Doctor Douglas gave an illustrated lecture on earthquakes, and G. C. Rice an illustrated description of the Pittsburg Testing Station. The evening was devoted to the banquet, which was not a Lenten festival. Colonel Penhale presided with rare judgment and theoretical precision. We say theoretical, for while he rang the bell on time, he could not stop the acceleration the speaker had acquired in the interval. Among the songs at the banquet was "Drill Ye Terriers Drill," "Are Ye's Ready," "Then Let Her Go," "Fire," "Bang," "All Over." This song occupies the same important place in ore mining that "Down in a Coal Mine" does in coal mining. In the rendition of either of these classics, the listener, if possessed with an ear for harmony, can frequently detect a load of "pathos."

While it was understood that some Yankees would attend this meeting, nevertheless, as a comparative stranger who had never tested the sociability of Canuck engineers, there was a certain anxiety, which proved needless, for the good fellowship so universal among real mining engineers in the States sat on the brows of the Canadians like a halo.

"Oh, we're all frank-and-twenty
 When spring is in the air;
And we've faith and hope a-plenty
 And we've life and love to spare.
And it's birds of a feather
 When engineers get together."

On Saturday morning Messrs. Constant and Garrison, while leading a personally conducted exploration party through old Quebec, discovered the "North Pole."

The statesmen present were James Douglas, Frank L. Nason, G. C. Rice, Walter R. Ingalls, F. L. Garrison, A. R. Ledoux, David T. Day, H. C. Constant, Heinrich Ries, C. H. Richardson, J. F. Kemp, Eugene B. Wilson. There were others whose names unfortunately have escaped the writer.

WORLD'S PRODUCTION OF ZINC IN 1910

M. M. Rudolf Wolff, Kreuger & Co. give the world's production of zinc for the last 3 years as follows:

	1910 Tons	1909 Tons	1908 Tons
Belgium	169,998	164,511	162,189
Silesia	138,100	140,625	138,092
United Kingdom of Great Britian	86,262	79,152	72,874
France	50,586	49,237	48,701
Roumania	60,876	58,799	53,586
Holland	20,661	19,233	16,983
Poland	12,500	12,926	13,941
Austria	13,200	13,146	11,889
Spain	6,440	6,011	6,263
Italy			75
Total Europe	558,623	543,646	524,593
Australia	600		1,169
United States	238,770	239,478	183,040
Total production	797,993	783,124	708,802
Average price in London	23-0-0	22-3-0	20-3-6

DEVELOPMENTS IN CYANIDE PRACTICE

Written for Mines and Minerals, by Percy E. Barbour

Mechanical Improvements Permitting the Application of the Process to Different Kinds of Ores

Chemically the cyanide process has changed little since the Forrests and MacArthur completed their elaborate and comprehensive experiments. It is true that much importance attaches to the discussion of the varied chemical formulas and practices, but the strength of the cyanide solution to be employed is still a variable, due to the ore, local conditions, and the personal equation; the comparative merits of potassium cyanide and sodium cyanide, the importance of and the relative amounts of lime and acetate of lead to be used, are all the subjects of more or less discussion. But mechanically the developments have been so great that it may be said that, were it otherwise, the cyanide process would today be only applicable to a few especially favorably adaptable ores.

The treatment of slime is the most important matter in the cyanide process today. A mill built 10 years ago at a cost of nearly $500,000 was a total failure on account of the large amount of slime produced, which could not be treated economically. Five years later the slime problem seemed so simple, in view of then recent developments, that the filter press was applied as the solution of the problem. Today, 5 years later still, there is only one wholly filter-press plant of any size in the country, and it is unlikely that it would be duplicated if an increased capacity were desired, although the working costs are satisfactory.

While previously it was the cyanide millman's main object in life to make the minimum amount of slime, the best practice today is to slime the entire product. Slime is generally considered the product which will pass a 200-mesh screen. No one machine has yet been devised which will economically reduce the ore, as it comes from the mine, to slime in one operation.

Eliminating the discussion of the preliminary crushers, the point where the slime first appears as a quantity to be dealt with is taken up in this article. This will be after crushing to

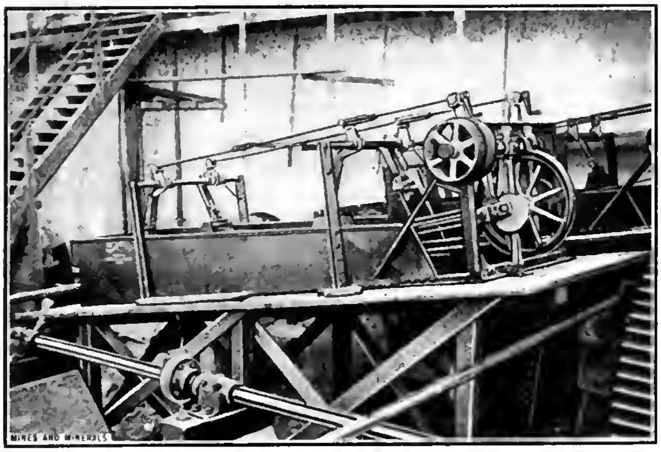


Fig. 1. Dorr Classifier

a 16- or 20-mesh screen, because when crushed this fine, almost every ore will make considerable slime. Various modifications of the hydraulic cone classifier have been used at this stage of the treatment to eliminate that material which has already attained the desired fineness. The latest machine for doing this, and one which is being very widely adopted, is the Dorr classifier, described in June, 1908, issue of Mines and Minerals. The machine shown in Fig. 1, can be set so that the discharge will fall into the feed-hopper of a tube mill, thus the amount of mill head required for hydraulic classification is reduced to a minimum. The machine requires no addition of clear water, as do the cone classifiers, thereby effecting a large saving in the water consumption. It makes a clean sand and a slime at one operation, is cheap, consumes little power, and requires no attendance. At the 600-ton mill of the Goldfield Consolidated there are 6 double-rake classifiers which discharge into the

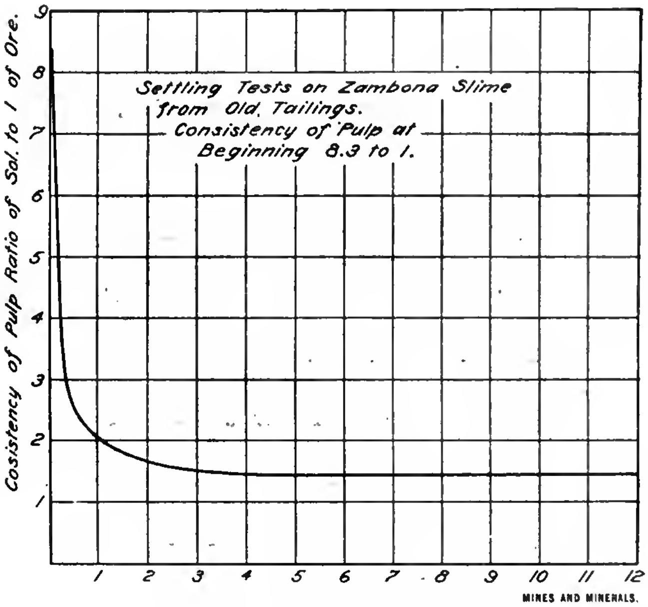


Fig. 2. Settling Tests on Zambona Slime

tube mill feed-hopper. Each machine handles 65 to 70 tons of battery sand per day, 16-mesh and under, and delivers to the tube mills a sand free from slime and containing the proper quantity of moisture (30 per cent.) for efficient tube-mill work.

Tube milling presents one of the most interesting, most anomalous, most complicated, and perhaps least understood of any of the mechanics of the cyanide process. It is by no means mysterious; but the matter of testing and comparing various types of tube mills involves such complications and often such dissimilarity of conditions that the tube-mill field is one of the broadest and most interesting for the technical investigator. A tube mill consists of a cylindrical steel shell from 4 feet to 6 feet in diameter lined with some hard tough lining. The mill usually revolves on hollow trunnions, one of which forms the inlet and the other the outlet for the pulp product. Danish flint pebbles are charged into the mill, and these falling and rolling on each other and on the tube-mill lining as the mill revolves, crush the particles of ore in the thick pulp in which the pebbles work. The efficiency of the tube mill is the measure of the slime produced per horsepower consumed; but this imposes the most unfair relative conditions for judging the comparative merits of two tube mills in different localities. For example, tube mills at the Black Mountain mill in Sonora, Mexico, treat an ore which is soft and crushes almost like volcanic ash, while the Goldfield mills treat a hard quartzose andesite. A comparison of daily tonnage through these different mills is obviously both unfair and misleading, as one ore is soft and the other is hard, the one requiring a short mill and the other a long mill. Until some better method of classifying conditions and comparing results is devised, the comparison of types of tube mills except when operated on the same ores at the same time will always be open to discussion.

The size and proportions of tube mills are for the same reasons disputative subjects. With an ore which slimes easily, the necessary fine grinding may be accomplished in the first 10 or 12 feet of the mill; with a harder, tougher ore, a 22-foot mill may be required in order to give more pebbles sufficient

time to work on the sand. The proper diameter of the mill is likewise a question. One of the most prominent manufacturers of tube mills says: "The small-diameter mills are very inefficient." As the results of a series of tests on mills of three different diameters, running side by side on the same ore and under the same conditions, an engineer of standing reports "the efficiency of a tube mill is inversely as its diameter." The

Fig. 3. Pachuca Tanks

speed of a tube mill is fixed within narrow limits. If run too slowly the pebbles simply slide on the lining, causing excessive wear and giving poor grinding effect. If the speed is too high the pebbles are held against the shell by centrifugal force and no grinding is accomplished.

An extensive series of tests* conducted at El Oro, on three tube mills of the same make, but of different diameters and different lengths, led to the following conclusions:

"The efficiency increases proportionately to the amount of pebbles contained in the mill."

"The efficiency increases with the coarseness of the sand fed to the mill."

"The efficiency decreases proportionately to the rate of feed."

In a paper† on the "Economics of Tube Milling" are given results of working tests on tube mills 5 feet diameter by 23 feet long, made at the Colorado Springs plant of the United States Reduction and Refining Co. These tests demonstrated that the greatest grinding efficiency was obtained with these mills when the pebble charge was 19,000 pounds. With a pebble charge of 25,000 pounds the amount of fine grinding was increased, but at the expense of a disproportionate increase in the horsepower required. With 19,000 pounds of pebbles the charge filled the tube to within a few inches above the center line. The percentage of solution to ore in the tube-mill pulp was varied, and it was found that both the amount of fines made and the amount of power required was affected. It was demonstrated that when the amount of solution in the pulp reached about 35 per cent. the best and most efficient work was done. This is in accord with other good tube-mill practice and it is on account of this that the Dorr mechanical rake classifier is such a satisfactory machine to dewater and classify the pulp before it is fed to the tube mill.

The Hardinge conical mill is a new type of grinding mill, a competitor of the tube mill, which is attracting attention. This mill consists of a short cylindrical section with conical sections of different lengths at each end. The cylindrical section is of considerably larger diameter than the tube mill which it is designed to replace. The most important claims for this conical mill are that "it has three times the capacity for the same power, requires one-fifth the pebbles, and has less wear on the pebbles and lining." Unfortunately this mill has not yet been tried extensively enough to provide accurate data, especially as to fine grinding, to thoroughly establish these claims. The most unique claim made for this mill is that it classifies its pulp and its pebbles, so that the larger pebbles work on the coarse sands and the smaller pebbles work on the fine sands. This is claimed to be a most efficient natural selection. (But the idea of natural phenomena in this machine working so beneficently in the right direction makes one stop and ponder over the almost human cussedness of some of the other machinery about the mill.) That the larger pebbles and the coarser sands tend to remain in that portion of the mill with the larger cross-section can be seen from a glass working model of the mill. It is undoubtedly true that a plain cylindrical tube mill would not give this sorting action when working dry, but it is also true that in a tube mill there is a certain amount of similar classification due to the action of the current of pulp through the tube. This is evidenced by the small worn pebbles which are found at the discharge end. There seems room for a reasonable difference of opinion as to the relative merits of having the finer sands worked on by smaller pebbles. But for grinding where 100-mesh is sufficiently fine the Hardinge mill has shown some astonishingly good results. It has not been tried out sufficiently for sliming to give the desired amount of data, and 100-mesh is not fine enough for the subsequent slime treatment.

In the Colorado tests referred to, the mills were started with a charge of only 3,000 pounds of pebbles and the required horsepower was 18.8. The power increased in almost exact proportion to the amount of the pebble charge, until the latter had reached 23,000 pounds, which filled the mill much fuller than even poor judgment would dictate. It would seem natural that the power consumed should increase thus with the added pebble load revolved, and this would seem to apply to the conical as well as to the tube mill. So that in this respect they would seem to be on the same footing. With these Colorado tests showing an increasing efficiency with the addition

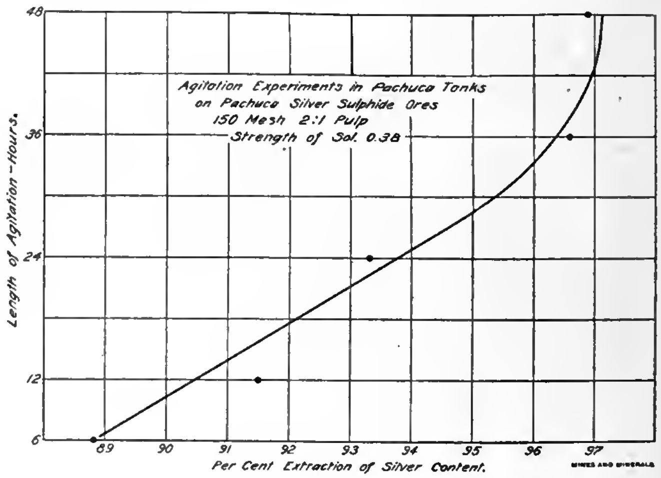


Fig. 4

of pebbles, from the initial charge of 3,000 pounds to a charge of 19,000 pounds, it would appear that the smaller charge of the conical mill must have an efficiency per pound very much in excess of the tube mill in order to give a greater relative efficiency, and very much greater than might naturally be expected.

The product from the tube mill is generally not of a wholly uniform grade, especially if the machine is being pushed a little for tonnage. The result is that a portion of the product needs

* "Fine Grinding of Ore by Tube Mills and Cyaniding at El Oro, Mex. Trans. A. I. M. E., Vol XXXVII.

† Mines and Minerals, June, 1908

to be reground. Therefore a classification is necessary. If the pulp from the grinding mill is as thick as it should be for the best grinding effect, it emerges too thick for classification without the addition of further water. Therefore this product is classified, usually in hydraulic cone classifiers, the slime going direct to the slime-treating department and the sand being returned to the tube mill for regrinding. If the head is not too great the sand is most easily and economically elevated to the tube-mill feed by means of the Frenier spiral pump, made of light sheet steel with sheet-steel discs forming the sides of the pump wheel. This wheel revolving in a hopper to which the coarse pulp is supplied, scoops up the pulp and forces it around through the spiral channel toward the center. When it reaches the center it emerges through the hollow trunnion, then through a flexible joint and thence to the delivery pipe and to the point of discharge. While this pump is very satisfactory for handling fine pulp, it has its limitations. The head against which it will work varies from 14 to 20 feet, depending upon the material handled and upon the altitude of the locality. Heavy coarse pulp is not a material to which this pump is adapted.

During the classification of the pulp subsequent to the tube milling the addition of classifying water has made the pulp too thin for cyaniding. The overflow from the classifiers also contains too much water, these two overflows are therefore brought together and are dewatered. The old style conical-bottom settling tank with its filling, settling, decanting, discharging, and flushing, and then a repetition of the cycle, is fast being relegated to abandoned practice. A continuous pulp thickener is displacing the old intermittent method. Present practice decrees that the pulp when submitted to the action of the cyanide solution shall have a consistency of about 1 : 1½ or 1 : 2. The Dorr continuous slime thickener has been developed for this particular service, and was fully described in September, 1909, Mines and Minerals. The capacity of this type of thickener depends upon the nature and density of the slime feed, and the amount of moisture desired in the thickened discharge. A tank 35 feet diameter by 12 feet deep, fed with as much thin slime as the tank would take and give a clear overflow, discharged 285 tons of thickened slime per day, containing 39 per cent., or 111 tons, of dry solids. At the Liberty Bell mine, in Colorado, a tank 33 feet diameter by 10 feet deep, fed with a pulp containing 120 tons of solids to 660 tons of solution, discharged at the bottom a thickened slime containing 120 tons of solids to 288 tons of solution, or a 1 : 4.2 pulp. The vertical shaft with the plows made 4.8 revolutions per hour and required .14 horsepower. Three tanks equipped with the thickening mechanism had the capacity of five of the same tanks used in the old intermittent method and made a considerable saving in the amount of labor required. At the Goldfield mill, thickeners were installed to handle the pulp from the agitators which contained 2½ tons of solution to 1 ton of dry slime and they discharged a thickened slime containing only 35 per cent. moisture. This is an example of the successful working of the machine over a wide range of conditions, delivering a pulp containing from 35 per cent. to 61 per cent. moisture. Except in the case of an extremely argillaceous slime, a long settling period is not necessary, and the old intermittent method ties up capital in equipment and in product without adequate return. The curve, Fig. 2, shows the results of a 24-hour settling test on a slightly argillaceous slime from ores at the Zambona mine in Mexico. During the first 2¼ hours readings were taken every 15 minutes, then every half hour until 6 hours had elapsed, after which only two readings were taken during the remainder of the 24 hours, no difference in the slime settling being noticed. During the first 30 minutes the consistency of the pulp increased in an almost straight line from 8.3 to 1 at the start, to 2½ to 1. From this point to a consistency of 1½ to 1 at the end of 3½ hours more the curve is very regular; after this point the curve for the remainder of the 24 hours is a straight line. It is obvious from this curve that the slime settled all that it would in 4 hours. At the end of 2 hours the slime had settled to a consistency of 1.7 to 1, which is perfectly satisfactory pulp to go to the cyaniding department, and also it had reached the limit of time which could profitably be spent settling it.

The Callow tank has justly come into considerable repute as a dewatering device for concentrating mills. This tank consists of an inverted conical shell of steel with a gooseneck discharge spout at the apex. The pulp is fed into the center of the now upturned base of the cone through a small cylindrical funnel which is only long enough to discharge the pulp below the top surface of the solution in the cone. The slime settles to the apex and flows continuously through the gooseneck while the clear water containing practically no slime overflows the rim into a launder connected with the water storage tanks. At the Goldfield combination mill during a test of concentrating machines the following data were obtained from the Callow tank: Flow to Callow tank, pounds per minute, 137.7; dry solids to Callow tank, pounds per minute, 16.36; water to Callow tank, pounds per minute, 115.5; overflow in clear water in per cent., 48.15; per cent. of solids in discharge, 22.62.

The pulp having the desired fineness of 200 and the desired consistency of about 1½ to 1 or 2 to 1, it is ready to be cyanided. One of the two devices which has made economical slime treat-

Fig. 5. The Oliver Filter

ment possible is the Pachuca tank. Like many other things extremely simple, its introduction was radical, even though its present form is the result of development. Agitation and aeration of the pulp had been tried by means of many different methods and devices, but without accomplishing the desired results commercially, until Brown, of New Zealand, built and experimented with a high tank of relatively small diameter equipped with an air lift. This tank, known in Australasia as the Brown tank, was soon introduced into Pachuca, Mexico, where it was developed so rapidly and adopted so widely that it has become known in this country and Mexco as the Pachuca, tank shown in Fig. 3. It consists of a plain cylindrical steel tank 13 feet diameter by 55 feet high. Inside the cylinder, extending from within about 30 inches of the top to within about 18 inches of the bottom, is a tube 18 inches in diameter, open at both ends. At the bottom of this tube is a small pipe through which is furnished compressed air. The tank is filled with pulp to the top of the tube or within 30 inches of the top of the tank. Compressed air at a pressure of 22 to 25 pounds per square inch is admitted at the lower end of the 18-inch tube, and by expanding carries the pulp up through the tube where it is discharged into the tank proper. The operation is continuous so long as the compressed air is supplied, in fact it is the old principle of the air lift and ejector applied in a rather

novel way. The result of this operation is a perfect agitation during which every particle of this slime is afforded the most intimate contact with the cyanide solution. The advantages or disadvantages of aeration as affecting the cyanide itself are a somewhat mooted question, but it has been found that by increasing the height of these tanks and thus increasing the aeration the cyanide consumption is lower than in tanks only moderately high, and very much lower than in low tanks. A tank 13 feet diameter by 55 feet high will hold a charge of about 100 tons of dry slime in a 1½ to 1 pulp. The free air required varies from 4 to 6 cubic feet per minute at an expenditure of about 2 horsepower. The time of the agitation depends upon the kind of ore and the economic point of extraction, because beyond a certain point it does not pay to increase the extraction even though it is mechanically and chemically possible. From experiments made by Grothe and Carter, of Mexico City, results of which have been given by them, the curve shown in Fig. 4 has been plotted which shows strikingly where the economic point was reached on the ores under treatment in this particular case. The curve is a straight line during the first 27 hours of agitation, during which time the extraction was in direct proportion to the length of time of agitation. After this the curve turns up very steeply, and the point at which the profitable extraction ceases is easily determined from the known conditions at the camp where the treatment is to take place. So far as known to the writer, there are only two installations of the Pachuca tanks in this country, at the Goldfield Consolidated, in Nevada, and at the Gold Roads, in Arizona. These are the only large cyanide plants erected in this country since the Pachuca tank came into vogue.

The second important improvement in the mechanics of the cyanide process is the perfection of the vacuum filter. Without it most of the low-grade ores would still be impossible of profitable treatment. The first type to come into successful practice was the leaf filter, as exemplified in the Butters. This filter consists of a number of leaves made of thin wooden frames covered on both sides with cocoa matting and canvas, the space between the canvas sides being connected with a vacuum system by piping through the top of the frame. These leaves are suspended in a tank into which the cyanided pulp is discharged. The problem now is to extract from this pulp all the gold-bearing cyanide solution. When the tank is filled with pulp so that the filter leaves are immersed the vacuum pumps are started and the gold solution is sucked through the filtering canvas and matting and is discharged into a gold-solution storage tank. The solid particles of the pulp adhere to the sides of the leaves forming a cake of slime. This process continues until the cake has reached a thickness beyond which it is not economical to continue. This thickness is from ¾ to 1½ inches. The pulp remaining in the tank is then drawn off and the tank filled with a weak solution of cyanide. During these operations a sufficient vacuum is maintained to prevent the cake from falling off the leaves. The weak solution is now drawn through the cake and is discharged into weak-solution storage tanks and the filter tank is filled with wash water. After the cake is thoroughly washed and the wash water remaining in the tank drawn off, the slime cake which is now more or less barren is discharged to the tailing dump. This tailing carries ordinarily about 30 per cent. wash water, whereas had the cake been discharged at the point where the drawing off of the gold solution was completed, this 30 per cent. moisture would have been the gold solution. Thus the advantage of washing the cake is clear.

The Moore filter process is similar to the Butters, except that instead of the filter leaves remaining in the tank with the filling and drawing off of the pulp and the various solutions, the Moore leaves are arranged in nests which are lifted from one tank compartment to another containing, respectively, pulp, wash solution, and wash water.

There are several modifications of the vacuum filter, but the one which at present is making greatest strides in application, due both to its efficiency and remarkably low first cost, is the Oliver continuous vacuum filter. The intermittent feature of the other processes is of course objectionable.

The Oliver filter, whose operation is continuous, has given astonishingly good results, is simple to operate, and requires probably smaller initial investment than any other. It consists of a large drum 11 feet 6 inches diameter by 8 feet wide, with open heads, mounted on a light cast-iron spider, as shown in Fig. 5, and is about two-thirds submerged in a wooden tank into which the pulp flows continuously. The drum makes a complete revolution in 5 minutes. It is covered with canvas and other filtering material supported by light framework within the drum. The circumference is divided into sections which are connected by pipes which lead through the hollow trunnions of the drum, by which it is supported and revolved, to the various pumps. Through that portion of the drum's surface which is submerged the gold solution is drawn by the vacuum pump while the slime cakes on the canvas cover of the drum just as on the leaves of other vacuum filters. As the drum revolves and a section emerges from the pulp the excess solution in the cake is drawn through and as this section reaches a point near the top of the path of revolution it encounters a spray of wash solution. At this point an ingenious automatic valve prevents the wash solution from mingling with the gold solution, each solution going to separate storage tanks. The section of the drum passes from the weak-solution wash to another spray of wash water and then just before it is about to be again submerged in the tank full of pulp a scraper fixed against the drum cleans off the now practically barren cake of tailing which drops into a hopper and thence goes to the tailing stacker. With the Oliver filter, the solutions are kept as separate as in any of the leaf filters, the tailing is as clean and carries as little moisture as in the other types, and the tailing is removed from the filter by a mechanical scraper.

The precipitation of the gold from the solution by means of zinc is the *sine qua non* of the process. Where formerly zinc shavings were used, now zinc dust is employed with very different equipment and results. At one time the precipitating room contained a series of boxes filled with zinc shavings through which the solution flowed. Now the zinc is fed in the form of dust to the gold cyanide solution before it is pumped to the precipitating plant. During its course through the pump it becomes intimately mixed and the pipe line is made sufficiently long to furnish time for precipitation to become complete while the emulsion is traveling through. The emulsion now consists of a barren cyanide solution in which the gold precipitate is in suspension, it is therefore delivered to filter presses from which the barren solution flows continuously while the precipitate is collected in the press in the form of a cake. At regular intervals the filter presses are opened and the cakes removed, melted down, and refined. By this method the gold precipitate is always locked up in the presses where it is perfectly safe. This filter press consists of a series of triangular cast-iron frames about 3 feet on the edge by 1 inch thick, supported on horizontal guides and clamped together by means of long longitudinal rods. Between these cast-iron frames are sheets of filtering canvas through which the solutions pass and between which the cake of gold precipitate accumulates.

This method of zinc dust precipitation is patented by C. W. Merrill, of the Black Hills, S. Dak., and the filter press used is his adaptation of the ordinary filter press used for clarifying mill solutions.

Mercury is produced in the district of Bachmut, state of Ekaterinoslav, Russia, from three mines belonging to the Auerbach company. From 90,000 metric tons of ore mined in 1910 360 metric tons of mercury was obtained. This ore according to the figures, carried but .88 pound mercury per ton, if all was recovered; a practical impossibility.

UTILIZING ZINC TAILINGS

Written for Mines and Minerals, by Lucius L. Wittich

In the Missouri-Kansas-Oklahoma zinc and lead district, more than 95 per cent. of the material taken from the ground and run through concentrating plants is waste, so far as metallic value is concerned, but the growing demand for concrete as building material is creating an ever-strengthening market for the mining waste, for it has been determined that the mining gravel, when properly mixed with sand, cement and water, forms a concrete that will wear indefinitely.

Business Development by Which Tailing and Refuse from Zinc Mines Are Used for Concrete

The present-day evolution of the tailing pile is illustrated by comparing the tailing pile, Fig. 1, and the structural framework of the modern Joplin, Mo., business block, Fig. 2, that was mostly made from it. The use of the mill waste in the $80,000 Union station is more in evidence than in the construction of the business block, for the concrete mixture constitutes the exterior walls of the structure, while in the business block, the framework will be covered with pressed brick and cut stone, and therefore will not be visible.

In the mining industry the necessity of reducing waste is paramount, and in the Joplin district the ore producers are awakening to the commercial value of mining debris that was regarded as almost worthless.

Perhaps no material used in modern construction has attracted the attention of the general public as has Portland cement, which forms the chief ingredient of concrete. The possible uses of this latter material are yet unknown.

Enormous heaps of waste tailing and veritable mountains of flint and lime boulders have accumulated since the first mining plant of the Joplin district was launched into operation back in the early seventies. During the last year alone, 14,000,000,000 pounds of material came from the crushing plants, exclusive of millions of tons of boulders that were not milled, and of this enormous tonnage that actually passed through the mills only 700,000,000 pounds were zinc-blende and lead. In other words, about 5 per cent. of the district's mill production was ore, while 95 per cent. was what might be termed waste, although much of this waste has played its part in beautifying the prosperous cities of Oklahoma, has been built into bridges and viaducts in Arkansas and Nebraska, and has helped build modern office structures and stations in Missouri.

As ballast for railroads the mining tailing is unexcelled; and white ribbons, indicating the roadbeds, stretch for hundreds of miles in every direction from the zinc and lead district. It is possible for the traveler to leave his home in the mining district, walk down steps made of concrete, then over sidewalks in which mill tailing plays a prominent part, then over macadamized roadways of crushed and packed gravel. He will purchase his ticket in a railroad station built almost entirely of the mining waste and will board his train only to be whisked over a roadbed, level as a dancing floor through the liberal use of tailing as ballast, and over concrete arches, rigid as though hewn from solid rock, while on either side, marking the boundary of the right of way, will stretch mile after mile of wire fencing, held firmly by posts that will defy the deleterious effects of weathering—for these posts have been constructed of waste mill tailing and cement.

In the construction of the new union station at Joplin, and this formula will apply for average building purposes, 15 parts out of 22 consist of waste products from the Joplin mines. In detail the formula was: 10 parts tailing, 5 parts "Chitwood" sand (mining debris from sludge mills), 3 parts river sand, and 4 parts Portland cement. A similar formula was followed in the preparation of a mixture that went into the large business establishment shown, the conglomerate mass being thoroughly mixed and moistened and placed in molds built of boards around spirals of steel reenforcement, delicate and fragile enough in themselves, but the whole forming the framework of a structure, which there is every reason to believe, will wear indefinitely.

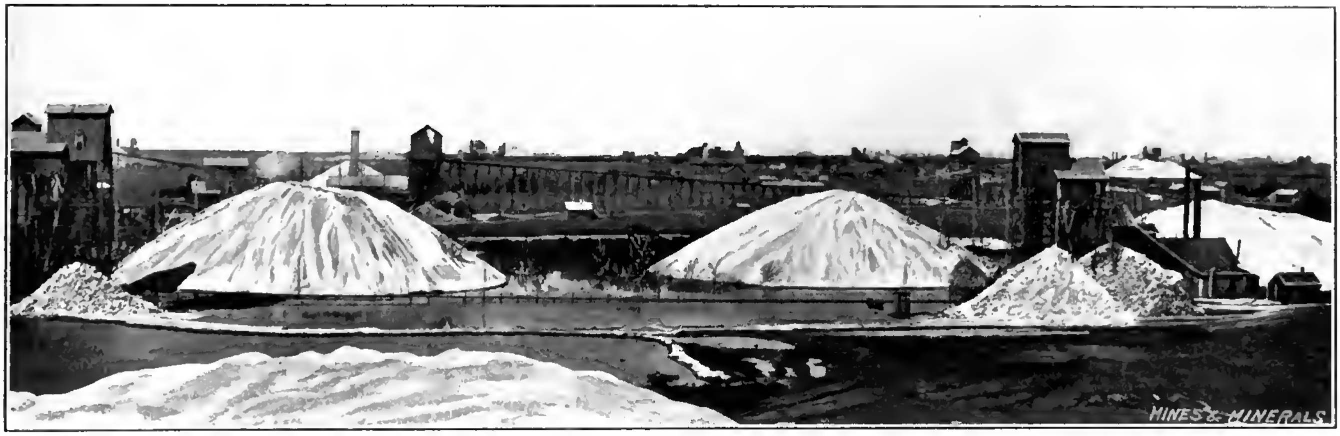


Fig. 1. Zinc Tailings That Were Converted Into a Skyscraper

For more than 30 years the Joplin district has been producing millions of tons of waste gravel annually, and thus the accumulation of this waste reaches the grand aggregate of more than 100,000,000 tons, with an estimated valuation of more than $20,000,000. In addition there are millions of tons of boulders on which it is impossible to place even an approximate valuation, but which are beginning to figure in the building progress of the immediate district, many handsome dwellings and other buildings and retaining walls having been constructed from this crude material. Boulders, glistening with calcite crystals or splotched with zinc or lead ores, are in greatest demand where an artistic finish is desired; and the effect secured from the use of such material is decidedly pleasing.

The outside demand for mine tailing has resulted in a number of "jobbers" entering the field and the new merchant or gravel wholesaler has become established. However, only those gravel heaps that are adjacent to railroads are in demand for average purposes, for the industry of shipping mining waste is yet in that stage where the supply is far in excess of the demand; hence the shipper is not forced to carry his quest far before finding the exact article that meets his requirements. Approximately 2,000,000 tons of waste mine gravel are now being used annually in building construction, railroad ballasting, road building, and in the manufacture of concrete merchandise. Yet, great as this tonnage is, it represents less than one-third of the waste mill material produced in the district in a year's

time. But the demand is growing rapidly; new industries are being created and new products composed of concrete are being manufactured; in fact, the demand for the tailing is growing much faster than the corresponding growth of the tailing production, despite the fact that each year sees an increasing production from the mines of the Joplin district. Therefore it is a question of time only before the margin between the production and the consumption narrows to a point that should materially increase the market price of the tailing.

At present, gravel for building purposes finds a ready market at prices ranging from 15 to 35 cents a ton (jobbers' figures), the average being about 20 cents a ton, which means that the district will waste, in 12 months time, a product from the concentrating plants valued at almost $1,500,000, of which only about $400,000 is reclaimed. To the producer and to the land owner the return is less, the price paid by the jobber seldom running more than 2 or 3 cents a ton in the pile, or from 6 to 8 cents a ton where the producer attends to the loading; yet this added income of approximately $100,000 is no small factor in computing the values of materials produced in the mining region in a year's time, especially when it is taken into consideration that comparatively few of the tailing piles of the district are situated adjacent to railroads or spurs, and that

Fig. 2. Concrete Building Made of Zinc Tailing

the $100,000 represents a profit gathered by comparatively few operators. It cannot be expected that the industry of wholesaling mine tailing should spring, like a mushroom, into full-fledged development in a night. Just as it required many years for the zinc industry of the Joplin district to attain recognition, so it is by degrees that the worth of the mill tailings is being realized. In the mining of metals, lead ore was the first and only metallic product produced for commercial purposes; zinc ore, known as "jack," was discarded as worse than useless; it was condemned as an impurity that penalized the value of the lead; it was discarded in thousands of tons, and for years the waste piles of the early camps were rich in purest zinc, wanting only a simple concentration treatment to convert the ore into a marketable product, especially desirable because of the choice quality of the zinc blende. And when at last a St. Louis chemist for the M. & H. Zinc Co. discovered the presence of the zinc and made public his discovery, it was only a short time before the production of lead ore took second place in the importance of the district's mineral resources; at the present time the mining zone of Missouri, Kansas, and Oklahoma is furnishing about 300,000 tons of zinc concentrates annually, compared to 50,000 tons of lead. Similar conditions have prevailed in other mining camps throughout the world. A recent striking example is found at Leadville, Colo., where the reported discoveries of carbonate of zinc deposits have recently been made.

What was true of zinc ore is now true of mine tailings. The commercial importance of the material has been overlooked up to a very recent date. The same thing is true of zinc ores carrying heavy percentages of sulphide of iron, which penalizes the zinc. Thousands of tons of iron-bearing ore are left untouched in mines, but it is not going beyond the bounds of reason to predict that within the course of a few years a new market will be created for this class of ore. Already the American Zinc, Lead and Smelting Co., the largest producing and smelting company in the district, is investigating conditions with the view of installing a Huff electrostatic separating plant, similar to the ones now in successful operation in Wisconsin and Colorado. If the company can be assured of 75 to 100 tons of concentrate per day, carrying 8 per cent. or more sulphide of iron, the plant is to be established. By-products of established value are now going to waste with every ton of Joplin zinc ore smelted. Few smelters utilize these by-products, although the aggregate value of materials going to waste will reach the hundreds of thousands of dollars annually. Among the more important by-products are cadmium, which is used in the manufacture of the finest paints; sulphuric acid, for which the demand is heavy; iron pyrites, the very material which lowers the value of the zinc and lead ores, and numerous other by-products of lesser value.

As a road-building material the merits of mine tailing have been appreciated for many years. The enterprising producer who first conceived the idea of converting the waste into macadam struck a keynote of popular progressiveness, with the result that scores of others followed his example until today Jasper County, Mo., ranks among the first counties of the state in the possession of hard, white roads that are moisture-proof, and which are as good for travel when the rain is pouring down as when the sun is shining.

The construction of concrete sidewalks, in which more than 50 per cent. of the material used is waste mine gravel, was the next step that added to the commercial value of the tailing and did more than any other one thing toward relegating to history the old-fashioned board walk of the small outlying village. Then, in quick succession, came the manufacture of concrete retaining walls, concrete curbing and guttering, and concrete skyscrapers—not of blocks but of solid mixture, welded together.

In view of the expanding field of mining operations, embracing new regions to the west in Oklahoma and to the east in the fastnesses of the Ozark Mountains, it is evident that the annual output of gravel for years to come must be enormous.

Flint and lime tailing, the former predominating, are brought from the ground and sorted, the boulders which contain little or no ore are discarded while all other material goes through sets of crushing devices and heavy rolls. Through gravity jigging the zinc blende and lead particles are separated from the chats, or tailing, which are discarded. As a certain percentage of ore, sometimes a very high percentage where milling practice is inadequate, is destined to escape in the tailing, the tailing mill has become common in the district. Here the waste from the first milling is rehandled and a further recovery of ore made. Recently, however, according to the Illinois Zinc Co., the alarming discovery has been made that tailing piles from many of the pioneer mills contain a much lower percentage of ore than when originally deposited, the deleterious effects of sulphuric acid waters intermingling with air having resulted in much of the zinc blende content being leached out. Tailing heaps, which at one time could have been retreated at a profit, have, in many instances, lost their value.

From the tailing mills, where the reclamation process is conducted without regrinding of the chats, the reworked tailings are especially desirable for use in concrete where important work is to be done, for from such tailing mills the recleaned gravel is freed from the finer particles, and the larger the individual pieces of gravel are the greater will be the demand for it in

concrete; but where the original tailings are reground in order that the ore separation may be made more perfect, the refuse is not so desirable for building purposes; but for cheaper grades of concrete, for road macadam or for railroad ballast, it still is good.

Tailing that passes through screens of ¾-inch mesh, after the finer particles have been eliminated, is used for the best qualities of concrete; from this the size of the mesh ranges down to ⅜ inch, the average of the entire district being about ½ inch. With the growth of the tailing shipping business, it is customary for jobbers to arrange with the producers for the tailing to be flumed directly into freight cars, ready for transportation. The method saves the cost of loading after the waste has been discharged from the mill; but where it does become necessary to load the gravel from heaps, various methods are employed the steam scoop shovel being widely used. Three types of this mechanical device are found in the district: the ordinary dipper, the "clam-shell" bucket, shown in Fig. 3, and the "orange peel" bucket. The "clam shell" operates at the end of a long steel beam and is different from the ordinary dipper in that it has two parts which grasp the material to be loaded, the big scoops opening like the shell of the clam after which the device is named. The "orange peel" has three shells that open to clutch the material to be lifted. Where any of these powerful machines are employed standard-gauge tracks are required, paralleling the railroad on which the freight cars to be loaded are stationed. The machines are self-propelling. Belt conveyers sometimes are employed to remove the gravel waste into freight cars, there being several distinct types of these machines in use in the district.

In the selection of tailing for concrete, color is of importance, and it is desirable to secure gravel that is as free as possible from sulphide particles, as iron sulphide and zinc blende have a tendency to leach if brought in contact with water and air. Blue or white flint particles, other considerations being equal, command the highest price; the rusty, saffron gravel, hinting of the presence of iron pyrite, is not used for building purposes if it is possible to secure the white or the blue. The sheet ground formation, as a rule, produces the best tailing as it consists of a fine-grained, deep blue, or white, chert, which when milled, separates almost perfectly from the zinc and lead ores. In mines where the ore is thinly disseminated through a big face of rock, the tailings show a tendency to be chatty, that is they are ore bearing, and if separation is made in a satisfactory manner it is necessary to grind the rock to a size which prohibits its use in concrete. Pure flint tailings are in strongest demand, although some maintain that a certain percentage of lime is desirable. The lime content should not exceed 30 per cent. Where it is desired to obtain lime for tailings that are purely flint, the stone quarries of the Carthage district, 18 miles east of Joplin, are a satisfactory source of supply. Analyses of stones taken from various places near Carthage, and some from Shoal Creek valley, near Joplin, show the material to be remarkably free from impurities. One test, for instance, showed: Silica (SiO_2) .21; alumina (Al_2O_3) .15; calcium carbonate ($CaCO_3$) 98.53; magnesium carbonate ($MgCO_3$) .88; total 99.77.

NO "SMOKE FARMERS" THERE

The Perth Amboy plant of the American Smelting and Refining Co. comprises a smelter, lead refining plant, electrolytic copper refining plant, electrolytic silver refining plant, and basic lined converters for Bessemerizing leady copper mattes. This plant refines about 7,500 tons of copper, 8,000 tons of lead, 3,000,000 ounces of silver, and 20,000 ounces of gold per month. To do this it smelts about 15,000 tons of ore and Bessemerizes about 1,500 tons of leady copper matte. Basic lined converters are the latest improvement in the metallurgy of copper.

ELECTRIC MOTORS SAVE MINE

The property of the Ernestine Mining Co. is located about 100 miles from Silver City, N. Mex., and includes ore bodies that make it one of the well-known individual holdings in the southwest. Its reputation is based upon the successful application of new methods of reducing low-grade ores, and partly upon the fact that prior to 1905 two companies had failed, with enormous financial loss, in their efforts to operate these mines.

Power was formerly generated by a Corliss engine, and distributed through belted line shafting. Pine wood, transported a distance of 10 miles on the backs of burros, at a cost of $10 per cord, is used for fuel. At some isolated points oil engines were installed, while in the mines proper, compressed air was used for operating the Ingersoll-Rand drills and also for an air-driven hoist between the upper and 800-foot levels. Acetylene gas was used for lighting.

When electrification was decided upon, Westinghouse apparatus, including generator, switchboard, motors, wiring, and lamps, were ordered complete. The generator is a three-phase, 220-volt, 60-cycle machine of 150-kilowatt capacity, and the total connected motor load aggregates 217 horsepower.

Cooper-Hewitt mercury vapor lamps were specified for the main building, metallic filament lamps for house lighting, and Nernst lamps for the company store. The excellent illumination of this store has given it the inside track in competition with

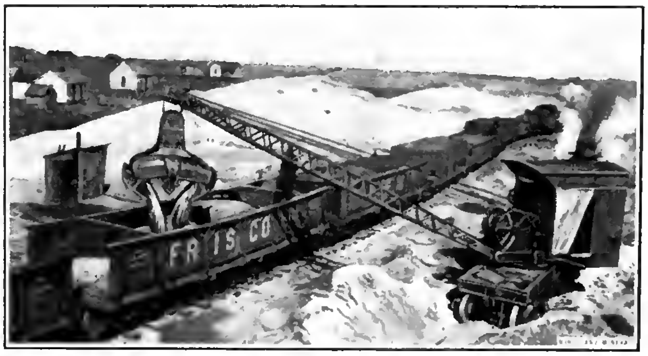


FIG. 3. LOADING TAILING FOR RAILROAD BALLAST, JOPLIN

the other stores, which do not enjoy the privilege of electric lighting from the private plant.

An incidental yet important advantage of electric drive lies in the ease with which a uniform consistency of slime can be maintained by observation of the electrical input to the pump motors. An ammeter is installed in each pump motor circuit and the reading noted under normal conditions; and change in the consistency of the slime increases or decreases the load on the motors and is quickly noted by the attendant. This plant was put into operation January 1, 1910, with the resulting improvement in operating conditions outlined below.

The tonnage has been increased by the addition of 20 new stamps weighing 1,050 pounds each and dropping 100 to the minute, replacing 20 stamps of 850 pounds each and dropping 96 to the minute. The 20 new stamps, and occasionally 10 of the old, are now run to meet the capacity of 90 tons per day, instead of 55 tons per day.

With this increased capacity and the electric drive, the power required has been increased only 38 per cent. over what the mill previously took when belt drives were used exclusively, and when the company was milling only 50 tons per day instead of 90 tons. In this increase is included the power used by the electric hoist, which was formerly air driven by an independent compressor.

The cost per horsepower year has been reduced $107.30. This results, with the amount of power now necessary to operate

the mill, in a net yearly saving in the fuel bill of $10,766. If with the increased capacity, the power per ton of ore handled were the same as with the old equipment, the yearly saving would be $13,905.

In addition to the above, the use of calcium carbide for lighting purposes has been eliminated and the gasoline engine has been replaced by a motor, making a net saving of $300 per month, or $3,600 per year.

As the electric hoist is used instead of compressed air, the total capacity of the air compressor is available for the drills, increasing considerably the amount of ore that can be mined per day, and also the amount of development work possible.

An additional interesting feature of the installation is the fact that, when the electric plant had been completed, the change from the belt to the electric drive necessitated the shutdown of the mill for only 15 minutes.

The actual result of practical value is demonstrated by a comparison of the cord-wood consumption, now and formerly. There is no increase in cord-wood consumption under the increased capacity of the mill; in other words, the company is mining and milling 35 tons per day more than formerly with no increase in fuel expense.

FINE CRUSHING WITH CONICAL MILL

The illustration, Fig. 1, is the flow sheet of a combination conical ball-and-pebble mill at a cyanide plant, and it will be noted that there is not a screen in the entire plant. The ore fed to the ball mill will not all pass a 2½-inch ring, yet this mill, installed for crushing to from 8 to 16 mesh, produced a product 33 per cent. of which passed 200 mesh, the oversize going to an 8-foot diameter conical pebble mill and 66 per cent. of its product passed 200 mesh even though the pebble mill contained a little more than one-half of its full charge of pebbles.

An interesting experiment with a ball mill was made in the presence of J. M. Callow in crushing a Butte lead-zinc ore which had all passed a Blake crusher. Steel balls were used, the mill making 32 revolutions per minute with an initial feed at the rate of 30 tons per 24 hours and a ratio of water to ore of 2.67 to 1. For comparison, a sample of the crusher run of this ore was recrushed in seven successive stages with rolls and hand screens, screening out the fines and returning only the coarse to the rolls after each screening, and the following results were obtained by the two processes:

	Once Through Hardinge Mill Per Cent.	Rolls 7 Returns Per Cent.
Held on 60 mesh	10.18	30.20
Held on 80 mesh	9.92	21.03
Held on 120 mesh	23.15	18.62
Through 120 mesh	40.32	21.34
Through 120 mesh as slime	16.83	8.81
	100.40	100.00

It will be noted that in the crushing with rolls an equivalent to seven screens was used, while in the conical mill the sizing was automatic and without any screen whatever. The latter only produced an excess of 8 per cent. as slime, which could have been materially lessened by the use of additional water or any one of three other adjustments to which the mill is adapted.

As to power and range of capacity of this smallest size of mill, the data given below are furnished by one of the largest iron companies in the world, which is testing the conical mill, grinding for concentration a very low-grade, hard, silicious, amorphous hematite: Mill used, 4½-foot diameter conical ball mill; charge of balls, 3,000 pounds; revolutions per minute, 30; power, less than 15 horsepower; capacity per hour, 7,850 pounds; size of feed, crusher product passing 1 inch; size of product, all but 2 per cent. passed 8 mesh.

The above shows considerably more than 5 tons per horsepower day, and is fully three times greater capacity than could be realized for similar crushing with stamps, rolls, or Chilian mills, and for simplicity of operation and freedom from necessity of oversight, there is absolutely no comparison.

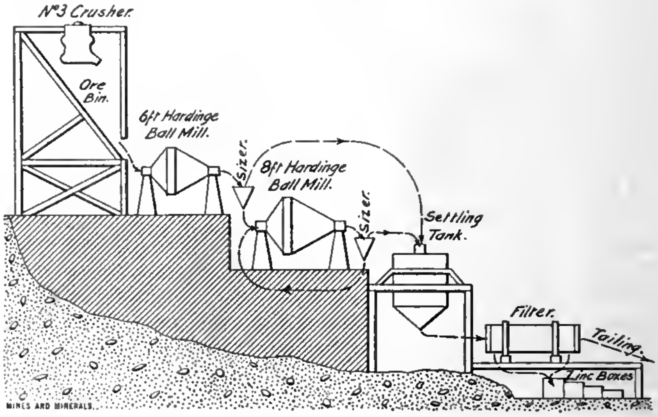


Fig. 1. Flow Sheet of Cyanide Plant

The Calumet Copper Co. is constructing three 8-foot diameter Hardinge mills that will be belt driven. Driving such large machines as these with a belt is an innovation that might be considered from the mechanical standpoint.

There is no question but that the conical mill has come to stay, in fact it has within the short period of 2 years found its way to Australia, New Zealand, Philippine Islands, South Africa, South America, Mexico, Canada, Norway, Sweden, and Belgium.

CYANIDE TREATMENT OF ORES IN MEXICO

Vice-Consul C. M. Leonard, Chihuahua

With the early completion and operation of two cyanide mills at Parral, by two American companies, that mining camp promises to renew its former activity, which for the past 3 years has been much curtailed, owing to the high cost of ore treatment and the presence of water in most of the mines.

Parral is a silver camp, the ores being silicious. Mining has been carried on there since 1640. Practically all of the mineralized ground in the district has been taken up, although many properties of much promise have not been developed. Labor is plentiful, good miners receiving only 75 cents per day, and common labor 50 cents. All values mentioned in this report are in gold.

The cost of mining and pumping is placed at $2.50 per ton. The normal output of the camp is about 25,000 tons per month, but it is conservatively estimated that this tonnage can be quadrupled with a cheaper method of treatment.

Based on the lixiviation process, the following statement shows the cost of treatment and the profit per ton: Average value of the ore, $12.50, but the percentage of extraction is only 80, reducing the net value to $10; cost of mining and milling, $2.50 and $5.50, respectively, leaving the net profit $2.

The cost and profit per ton by the cyanide process are given as follows: Percentage of extraction, 90; making the net value $11.25; cost of mining and milling, $5; leaving an estimated profit of $6.25; saving by cyanide process, $4.25 per ton.

The two American companies already mentioned are constructing cyanide mills of 300 and 400 tons daily capacity. It is expected the 300-ton mill will start in February, 1911. Much interest is manifested by mine owners at Parral in the result to be obtained by these mills, and if the process fulfils expectations other mills will be built, and many mines now idle will become good producers.

CONCENTRATION AT CALAMA, CHILE

*Written for Mines and Minerals, by F. A. Sundt**

Methods Used in Concentrating and Smelting Fine, Soft Copper Ores

The concentration plant of Humachuco belongs to the Compañia Poderosa de Chuquicamata, and is located at a short distance from Calama, Chile. It has the advantage of a branch line of the railroad. Its principal object is the concentration of the llamperas which are abundant in the mines of the company and too low grade for transportation. It produces 100 tons monthly of concentrated minerals of 17 per cent. copper and treats 16 tons daily of llamperas with 5 to 6 per cent. of copper. The concentration is performed on pyrite carrying copper and upon colored minerals, especially brochantite and atacamite. The last mentioned, which in their highest degree of purity are chemical combinations richer in copper than the iron, give a product of higher value but whose specific gravity is lower, and consequently the loss in the operation is also greater. The average loss in Humachuco reaches an average of 30 to 34 per cent. which quantity is not excessive if the low density of the concentrated minerals is taken into consideration. However great the loss may be, the method cannot be considered as a bad one. It must also be remembered that the plant in question was installed for minerals formerly considered of too low value to trêat and these products are occasionally furnishing a profit of from $4,000 to $5,000 per month to the owners.

There are 12 men employed at this plant, which has been so profitable for its owners they have accumulated a great supply of ore. This, together, with the quantities of llamperas which are thrown away at Chuquicamata on account of their low grade offer an excellent opportunity for capital.

The llamperas that can be concentrated at Huamachuco have to stand very high freight charges from Chuquicamata, being $3.05 per ton by rail, and $3.50 for the transportation in cars from the mines to the railroad.

The llamperas, on account of the matrix being easier to break than the mineral, are separated in the mines on flat or inclined bars 25 millimeters apart. The sulphide llamperas are mixed with colored ores. At Humachuco the llamperas are classified by means of revolving and cylindrical screens which give the following sizes: 25, 19, 12, 6, and 3 millimeters. The four larger sizes concentrate in screens. The product of 3 millimeters which forms the greater part is concentrated on two Ferrari vanning tables. There is a ball mill, No. 4, that works at night grinding some of the larger sizes, as the concentration of the tables is better than that of the screens. The mill produces 8 tons in 12 working hours with a screen of 2 millimeter openings. The grinding is carried on dry.

The two vanning tables treat 15 tons in 12 hours and produce 2.5 tons of concentrated minerals. No work is done at night on account of the limited capacity of the mill.

There are four screens which treat 6 tons in 12 hours and produce 1 ton of concentrated mineral.

The concentrated mineral contains sulphides and oxides of copper. The tailings from the tables, as well as from the screens, contain from 2.4 to 2.6 per cent. of copper. The tables make 120 vibrations per minute and like the screens they consume about .75 kilowatt.

The screens are made by Holman Bros., England, and do not have a filtering bottom. They are simple and have a movement of 60 vibrations per minute.

The water required for the concentration is elevated by three pumps through pipes 5 centimeters in diameter.

The power used is water. There is a wheel composed of troughs, measuring 3 meters in diameter by 120 in width. It is all made of iron and constructed at the factory of Orchard Bros., Antofagasta. It revolves 16 times per minute and furnishes 10.7 horsepower.

The cost of the wheel amounted to $1,000, and a like amount was spent in its installation.

The smelting plant of Chorrillos is situated a few minutes walk from Calama at an altitude of 2,300 meters. It belongs to the Compañia de Minas y Fundicion de Calama.† The principal business of this company is the operation of its mines at Chuquicamata, besides some other mines at Abra. Although the latter mines are not being worked, they formerly furnished the sulphur necessary in the smelter. Later on, the Chuquicamata mines produced the sulphur needed for matte in great quantities and as these mines are at a shorter distance from Chorillos, and do not contain water as do those of Abra, the preference has been given to them.

The smelters at Chorillos treat ores that average 10 per cent. copper. No poorer minerals could be found even in the midst of the desert of Chile. The Loa River furnishes the hydraulic power.

Chorrillos could have easily ranked among the great and modern copper smelting plants of Chile, for its production in previous years confirms this assertion. Chuquicamata, as it has been demonstrated before, could supply more than sufficient material to it.

The plant contains a section for copper smelting, and an installation for the conversion of matte to blister copper. There are also two tables for concentrating minerals. These two tables are considered as auxiliary to the other operations.

There is a rectangular blast furnace with seven tuyeres on each side with a crucible in the interior. Its area in the pipe section measures 3.60 × 1.05 meters. It has two series of jackets placed one above the other. The lower ones are inclined, and above the upper ones there are several meters of brickwork.

The furnace has a dust chamber of masonry resting on the ground. There are also two furnaces one of circular section and the other of octagonal section with eight lateral jackets in one series. They have only one fume pipe, dust chamber, and one metallic funnel. They have rectangular forehearths 2.10 × 1.45 meters. The furnace jackets are cooled by water.

The slag is carried away in cars of which there are five cone shaped and seven rectangular ones.

The converters are situated at a lower level than the furnaces so as to charge them by taking advantage of gravity. There are eight converters upon trunnions, revolved by hand. Four of them measure 2.40 meters in length by 1.70 meters in diameter with 14 pipes of 22 millimeters in diameter. The other four measure 2.30 meters in length and 2.17 meters in diameter and have 11 pipes. There are two converting sections. They have no fume chamber and the fumes are conducted away by means of two metallic hoods.

The three furnaces have only one air pipe. The air is delivered by two revolving blowers.

The converters are supplied with air by a Reidler compressor, having two coupled air cylinders with a diameter of 55 centimeters and a length of 1.12 meters. The compressors are water cooled.

The main furnace receives its power from a gas engine using producer gas. The engine was made by Fielding & Platt, Ltd., Gloucester, England, and has two coupled cylinders. It is a four-cycle engine with the mixture of gas and air exploded by the use of an electric igniter and is rated at 32 horsepower when making 160 revolutions per minute. The air for the starting device is compressed to 12 kilograms per square centimeter and the receiver contains a sufficient quantity for three starts. The gas producer is of the suction type using anthracite to produce the gas. It has a scrubber and a purifier. The steam necessary for the producer is made with the hot gases escaping from it. The producer requires only one man at a

* Injeniero de Minas, Professor Extraordinario de Metalurjia en la Universidad de Chile, Santiago, Chile.

† This company was formed in 1903 with a capital of £250,000. The home office is located at Valparaiso.

salary of $5 per day. The gas engine requires also a man with $7 salary per day.

The gas power is used when there is scarcity of water for the operation of the hydraulic motors which supply electric power.

The installation of the gas power plant cost $95,266.81.

There is a machine for briquetting the ore and dust, driven by belt attached to an electrical motor of 22 horsepower, 6.9 amperes, 2,000 volts, and 958 revolutions per minute.

There are two hydraulic power installations; one in the smelter, consisting of a small turbine which runs the concentrating machines, the other is the hydroelectric installation at Topater, distant about 3 kilometers. A canal from the lower river draws the water to a double turbine of horizontal axis. The turbine is of American construction, and has a capacity of 250 horsepower. It is connected by belt with a three-phase electric alternator of 2,000 volts. A man with a daily salary of $8 is sufficient for the generating station. At the receiving station, there is an alternator of 1,950 volts, 48 amperes, 180 horsepower, at 987 revolutions per minute. It drives the main shaft. These machines require a man with a daily salary of $6 and an assistant with $3. The complete hydraulic installation of Topater cost $102,015.40, according to the record in the books of the company, that is $400 per horsepower.

There are two Ferarri concentrating tables and a ball mill No. 4, with screens of 8-millimeter openings. The grinding is done dry in preference to using water.

The tables were used to concentrate the crushed quartz from the Abra Mine carrying 5 per cent. copper. The tailing composed of 85 per cent. silicon was used to line the converters. The concentrated mineral is pressed into briquets while the Chuquicamata crushed mineral with 8 per cent. copper is pressed directly.

There is a foundry containing a cupola that smelts 1 ton of iron per hour. There is a rectangular crane of 4 tons capacity. The jackets for the blast furnaces and the molds for the copper bars, as well as the castings for the converters are made here. The foundry employs 60 men.

Chorrillos smelters treat aluminum-silicate ores that require considerable coke and limestone flux. The smelting has cost as high as $25 per ton.

Limestone is obtained cheaply from the Calama quarries on the banks of the Loa. It costs $2.80 per ton delivered at the smelter and the principal work is to extract it. Chorrillos is located on a calcareous soil.

The bed of limestone on the banks of the Loa River has a thickness of from 30 to 50 meters. The limestone as a rule has a light color—yellowish and gray; is very compact and homogenous, and it is no doubt an excellent material for construction. In some parts it has a fibrous structure and is almost transparent.* It contains no more than 5 per cent. insoluble matter, and when calcined produces white lime. Quicklime is used in Chorrillos for the manufacture of briquets in the proportion of 10 per cent. to 90 per cent. concentrate and flue dust. The chalk of Calama used in the lining of the converters costs $6 per ton The clay of Lota costs 70 shillings per ton.

The gangue for the lining of the converters is quartz sand obtained when concentrating ore from San Jose del Abra It costs $24 per ton.

* Lorenzo Sundt, Estudios Geologicos y Topograficos del Desierto y Puna de Atacama, Vol. 1, 1909.

The coke has cost £3 11s. 4d. without dust, delivered at Chorrillos. The transportation charges are about 14s. 11d. Delivered at the mine. The anthracite costs £3 7s. 4d.

In 1908, 23,758 tons of ore was smelted and 1,199,660 kilograms of unrefined metal containing 590,604 tons of copper were sold. Besides, 1,319,844 kilograms of copper in bars were produced. In the first 6 months of 1909, the work at the smelters has been as follows: Smelted minerals, 13,029; unrefined mineral produced, 299,766 kilograms with 111,925 kilograms of copper; residuum, 101,112 kilograms with 37,751 kilograms of copper; bars of copper, 7,520 with 809,241 kilograms of copper; total produced, 875,317 kilograms.

The loss of copper figures 357 tons during this time, and it is considered equal to 16 per cent.

In June, 1909, the work of the smelter was as follows:

	Kilograms
Lime	990,170
Converter refuse	567,300
Oven refuse, after screening	19,950
Unrefined mineral siftings	89,100
Oven refuse	35,750
Pyrite flux	1,400,200
Sulphide, dross of metals	484,250
Light colored matte	556,400
Dark colored matte	303,550
Sulphide briquets	563,725
Clay colored briquets	250,000
Manganese flux	18,950
Total mineral	3,161,825
Coke	598,000
	Per Cent.
Lime from mineral	32.07
Unrefined mineral and dross	31.44
Coke from mineral	18.85
Coke from total	12.10
Bars	2,258

In the conversion 38 tons of Lota clay and 50 tons of chalk were used.

The converters produce 4 tons of copper. The lining is applied by hand. The clay is ground in mills, and is mixed with an automatic mixer in the same apparatus with the quartz. A very thin cloth is used to sift it. A mixture of two parts of sand and one part of Lota clay is used for the tuyeres and three parts of sand and one part of clay are used for the body of the converter.

DRIFTING RECORDS

The Vekol Mining Co., near Casa Grande, Pinal County, Ariz., is advancing a drift in hard blue limestone from the 120-foot level. The size of the drift is 8×6×4½ feet, and the working face is now about 500 feet from the shaft. The drilling is done with a Sullivan 2¾-inch air drill on a double screw mining column. Two drill runners recently had a contest to see which one could break the most rock in five shifts, with the results shown in Table 1.

Table 1

Shifts	1	2	3	4	5	Averages
Minutes to set up	26 min	16 min.	19	21	17	19.8
Number of holes drilled	11	11	12	10	12	11.2
Number of feet drilled	94	95	102	85	92	93.6
Average depth of holes in feet	8.54	8.6	8.5	8.5	7.66	8.36
Total working time	7 h. 54 m.	6 h. 25 m.	7 h. 15 m.	7 h. 10 m.	7 h. 2 m.	7 h. 9 m.
Feet of advance per shift	7′	6′ 11″	6′ 9″	5′ 1″	5′ ½″	6′ 2″

The time for setting up included the time required by the men in walking from the shaft station on the 120-foot level until the air was turned on at the drill. The total working time included setting up, drilling, tearing down, loading, and blasting. Seventy-five pounds of 1¼-inch powder were used per round of holes. A Sullivan straight-line, two-stage compressor furnished power for the drills at 110 pounds receiver pressure.

The above information was furnished by W. J. Forbach manager of the Vekol Mining Co.

THE GEOGRAPHY OF PETROLEUM

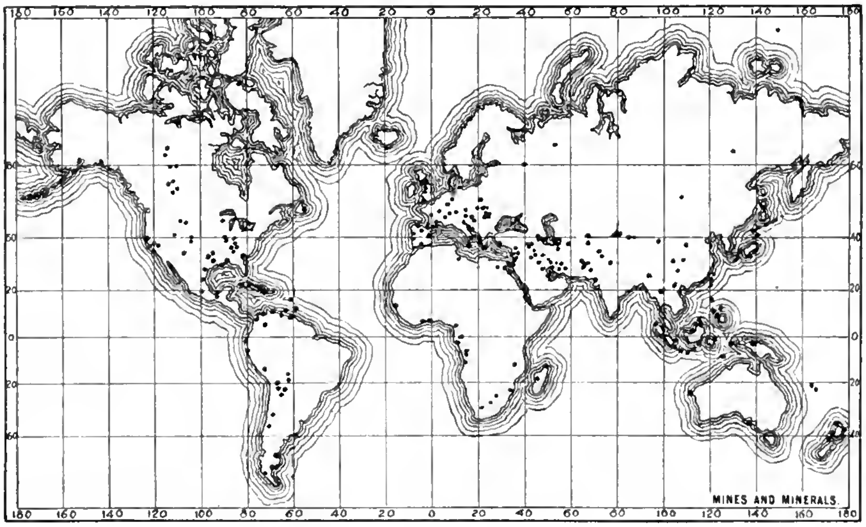
Written for Mines and Minerals

Qualities and Relative Quantities Produced by the Different Countries of the World

It is only a little more than 50 years since Colonel Drake drilled his first oil well near Titusville, Pa. While it is supposed that the Chinese antedate this discovery, the fact remains that they did not make practical use of their oil as did the Americans, for which reason the discovery of petroleum, so far as mankind is concerned, was made in Pennsylvania.

Since the discovery of petroleum, the United States has been the largest producer, manufacturer, and distributor of petroleum products, and seems destined, for some years at least, to occupy that enviable position, although there are now large quantities of oil produced in other countries. It is a strange coincidence that the best petroleum so far discovered, from a commercial standpoint, was the first discovered; and it is also a coincidence that it should be near the Atlantic seaboard so that it might be exported readily.

The first most abundant source of supply was Pennsylvania, but gradually that field became exhausted until now the first quality oil is obtained mostly from New York, West Virginia, and Ohio. This oil comes from the Devonian geological period known as the age of fishes. The next most important field, so far as quality is concerned, is known as the Lima-Indiana oil field. Before the advent of the gushers in Texas and California, this field was almost as prolific as the Appalachian field, which now includes New York, Pennsylvania, Eastern Ohio, West Virginia, and Kentucky, and will probably be further extended in time on the west side of the Appalachian mountain system. In 1896, Ohio as an individual oil producer took the lead from Pennsylvania; in 1903, California wrested the title for the most productive state from Ohio and has since continued to lead, while Ohio has been passed by Texas in 1904, Kansas and Illinois in 1906 and Oklahoma in 1907. While the Appalachian field is not now so productive as some others, the value of its oil exceeds that of the other fields individually. This is due to its nearness to the Atlantic seaboard and to the quality of its oil, which so far has seldom been equaled and never surpassed. Appalachian oil comes from the Devonian rock system, known as the age of fishes, and it is probably derived from the ancient fishes indirectly by natural chemical phenomena.



Map of World Showing Petrolem Regions

About 1885 what is known as the Lima-Indiana oil field was discovered. This field, as stated, reached its zenith about 1904 and then declined. Oil from this field carries considerable sulphur, which makes it more expensive to refine, for which reason it never did command the price of that of the eastern field, at one time its only competitor. By treating the oil with copper oxide, Mr. Frasch was able to rid it of sulphur and today it brings a fairly good price. It has a paraffin base and comes from the Trenton rock formation during which period there were large quantities of shell fish and trilobites in existence. It is probable that the oil of this horizon originated from their organisms in much the same way as the Appalachian oil was formed.

The Illinois oil field was discovered about 1889 and has been developed rapidly, being in 1909 the third most productive state. The principal oil-bearing sands belong to the upper and lower coal measures but in some places the Chester rock group of the subcarboniferous contains oil. Undoubtedly Southern Indiana will contain oil not far beneath its coal measures, and in case it does Western Kentucky will also. This statement is not made haphazardly but is based on oil indications obtained from various sources that have been submitted to the writer. The specific gravity of crude petroleum is mainly important as measuring the relative yield of light and heavy oils on distillation. The light oils, such as naptha, benzol, etc., are distilled from the petroleum at less heat than kerosene and lubricating oils. The Illinois oil, while it has a paraffin base the same as Pennsylvania and Ohio-Indiana oils, and carries less than 1 per cent. of sulphur, grades from 28° to 32° Baumé, consequently it has a low market value and oil grading so low as 28° Baumé is generally considered a fuel oil.

The production of the various states for 1910, as compared with their respective crude oil outputs during 1909, will be gathered from Table 2, the figures being in barrels of 42 gallons,

Illinois produced nearly three times as much oil in 1909 as Pennsylvania, nevertheless its total market value was but \$5,300,000 more.

Table 1 is made up from the United States Government statistics for 1909. It gives the production by states; the average price per barrel of 42 gallons during 1909; the increase in production over the year 1908 is preceded by +; the decrease by −; the per cent. of the total production from each state; the per cent. of the total value; and the total value. The Mid-continent field, which includes Missouri, Kansas, and Oklahoma, produced 49,804,922 barrels of oil in 1909, which averaged 36.8 cents per barrel and netted \$18,314,355, or practically \$2,900,000 more than Pennsylvania, with an output of 9,299,403 barrels. The oil in this field comes from the coal measures of the Carboniferous period, and while practically from the same horizon as the Illinois oil, has less market value owing to its having an asphaltic base. It varies considerably in specific gravity. The oil below 30° Baume sold for 28 cents per barrel and that above 30° B. for 38½ cents a barrel on the average. While it may be possible that location influences the selling price of this oil, it should on quality bring the same price as Louisiana or Texas oil.

The Gulf oil field includes Louisiana and Texas. Louisiana production comes mostly from the Caddo field in the northwestern corner of the state, which is in all probability a continuation of the Beaumont field.

Texas has two producing fields one termed the Corsicana or Northern Texas and the other the Beaumont or Coastal Texas; another, south of San Antonio, is not yet developed. The Louisiana oil and the Coastal Texas oil come from the Pleistocene formation of the Cenozoic period. It has an asphaltic base, but

on account of its location averaged 71.2 cents per barrel. There has been a decrease in production of about 66 per cent. in the Gulf oil field during the past 4 years. In 1905 the production was 36,526,000 barrels, in 1909 it was 11,912,000 barrels. It gained notoriety from the gushers at Spindle Top, from the oil-speculating schemes inaugurated on the strength of the gushers, and afterwards the fire which wasted the oil.

California enjoys the reputation of having the deepest oil wells in the world (4,400 feet), the greatest single producing wells, the most expensive wells to drill, and of being the greatest of all oil producers, both at home and abroad. The most remarkable well, the Lakeview, started in March 4, 1910, flowing 10,000 barrels daily; this increased to 40,000 barrels; July 20, the well declined to 31,000 barrels; July 30, to 22,000 barrels with 3 per cent. water; and in December it spouted less than 10,000 barrels and about 56 per cent. water. Mayo well came in with 30,000 barrels; Pioneer-Midway with 20,000 barrels; Union Oil Co. with 25,000 barrels; Santa Fe, 20,000 barrels; American Oilfields, 30,000 barrels; Union Oil Co., 30,000 barrels; Midway Five Oil Co., 24,000 barrels; Midway Premier, 30,000 barrels when it came in, and 2,500 barrels at the end of the year. A number of these gushers sanded and went out of business soon after being brought in. California oil comes from the Cretaceous

Mexico produced 2,288,742 barrels of high-grade oil which on the Isthmus of Tehuantepec has a gravity of 30° B. Owing to the oil war between the Waters-Pierce and the Pearson interests, 27,554,581 barrels of crude petroleum were shipped into Mexico and refined. Recent reports from Tampico,* January 8, 1911, credit that state with a well flowing 100,000 barrels daily. The well is 30 miles south of the famous Dos Bocas gusher, which was credited with a like production, then caved in, and next got on fire.

Petroleum is found in the Piura and Tumbes department of Northern Peru. Here the Lobitos, Negritos, and Zorritos districts produced 1,240,015 barrels out of a total of 1,316,118. The petroleum deposits of Lake Titicaca district in Southern Peru, near Pusi, in Huancane province, are expected to prove of more importance than those in Northern Peru.

Trinidad was prospected for pretoleum in 1909, not only because of seepages of oil and gas, but because of the success attained in Mexico in searching for oil near asphalt deposits. Oil has been found at Guayo.

Barbados contains small quantities of petroleum in connection with asphalt deposits.

In Venezuela crude oil is found in two classes according to Consul R. J. Totten.

Table 1. Production of Petroleum in the United States in 1909

State	Barrels	Average Price Per Barrel	Increase in Production	Per Cent. of Total Quantity	Per Cent. of Total Value	Total Value
California	54,433,010	$ .564	+21.35	29.89	23.92	$30,675,267
Oklahoma	47,859,218	.364	+ 4.50	26.28	13.59	17,428,990
Illinois	30,898,339	.640	− 8.28	16.96	15.43	19,788,864
West Virginia	10,745,092	1.642	+12.83	5.90	13.76	17,642,283
Ohio	10,632,793	1.244	− 2.08	5.84	10.31	13,225,377
Texas	9,534,467	.712	−14.92	5.23	5.30	6,793,050
Pennsylvania	9,299,403	1.659	− 1.33	5.11	12.03	15,424,554
Louisiana	3,059,531	.661	−47.15	1.68	1.58	2,022,449
Indiana	2,296,086	.870	−30.07	1.26	1.56	1,997,610
Kansas	1,263,764	.389	−29.86	.69	.38	491,633
New York	1,134,897	1.665	− 2.17	.62	1.46	1,878,217
Kentucky	639,016	.811	−12.20	.35	.40	518,299
Colorado	310,771	1.022	−18.14	.17	.25	317,712
Wyoming	27,887	1.655	+24.55	.02	.03+	44,478
Michigan		1.362	−62.28			
Missouri		1.362	−62.28			
Utah		1.665	+24.55			
Totals	182,134,274	$ .704		100.00	100.00	$128,248,783

system of the Mesozoic period; it is a thick heavy oil with an asphaltic base and therefore most suitable for fuel. The gravity varies from 12° to 26° B. The other states producing oil in

Table 2

Producing Field	1910 Barrels	1909 Barrels
California	74,300,000	58,190,000
Mid-continental	53,500,000	46,800,000
Illinois	35,000,000	30,900,000
Pennsylvania fields	26,000,000	25,000,000
Texas and Lousiana	14,900,000	12,500,000
Indiana and Ohio	5,000,000	6,600,000
Colorado, Kentucky, Tennessee, Wyoming, and smaller fields	1,500,000	1,700,000
Total	210,200,000	181,690,000

small quantities are Colorado, Missouri, Michigan, New Mexico, Wyoming, and Utah.

It is probable that oil will be found in other states, but quite improbable that it will be found east of the Alleghany Mountains, in states above North Carolina.

The production of oil in Canada came mostly from Ontario, but a small quantity came from New Brunswick. In 1909 the total production was 420,755 barrels worth $1.33 per barrel. Oil is said to have been tapped in small quantity at Parson's Pond, Newfoundland It is to be hoped that this is true, as that country needs it for fuel and light.

One has a specific gravity of .8837 at 15° C. and flows readily. On distillation .5 per cent. was driven off below 120° C.; .5 per cent. below 170°; 14 per cent. between 170° and 235° C. (illuminating oil); 28 per cent. between 235° and 270° (heavy illuminating oil); 51 per cent. between 270° and 370° (lubricating oil, etc.); and 6 per cent. coke. The thick oil was like tar, and gave no gasoline or burning oil.

Oil has been found in the northern part of Argentina, and in Patagonia.

Chili is said to have an oil well near Carelmapu flowing 500 barrels per day.

South America seems to have oil in almost every country, although at present is not a great producer owing to its latent development in this line.

The principal competitors of American petroleum are the Russian and Java oil fields.

In Russia the two principal fields are Baku and Grosny, on the Apsheron Peninsula, in the Black Sea. In 1909 the total production from these two districts was 65,970,350 barrels.

Recently considerable booming of the Maikop field, in Kuban province, in Southern Russia, has been carried on in London. News from this field is confined to indirect talk, apparently to keep alive the interest in the share market.

The production of petroleum in Galicia in 1909 amounted to 14,932,799 barrels, thus making this part of Austria-Hungary of importance to Europe.

*See Mines and Minerals, February, 1911.

Roumania has steadily increased its output of petroleum until in 1909 its production amounted to 9,321,138 barrels, and in 1910 there was an increase of about 25 per cent. Roumania is well supplied with refineries, there being more money in the manufacturing than the uncertain producing end. This shows the fine Italian hand of the Standard Oil Co. is being copied in Roumania, from which a cargo of kerosene was sent to America.

In 1909 Alsace-Lorraine, Prussia and Bavaria produced 1,018,837 barrels of petroleum. Italy produced 57,564 barrels in 1909; Japan, 1,879,542; and Formosa, 132,867 barrels. Borneo, Java, and Sumatra, the Dutch East Indies produced 11,041,852 barrels of petroleum and are keen competitors of the United States oil in the East.

Oil has been discovered in the Philippine Islands of Tayabas and Cebu, but has not been developed. It is high in volatile hydrocarbons, has a paraffin base, is free from sulphur and is supposed to come from Tertiary blue slates. The production of oil in New Zealand is about 4,000 gallons weekly, and that comes from a depth of 3,030 feet.

The preliminary figures of the American petroleum exports during 1910 have been issued by the Bureau of Statistics, Washington. These figures show the total exportation during 1910 of 1,397,000,000 gallons valued at $81,571,000. Compared with the figures for 1909 there is a falling off in quantity of 100,000,000 gallons and a reduction in value of over $11,000,000.

ILLINOIS RESCUE STATION COMMISSION

The Mine Rescue Station Commission of the state of Illinois has issued a Report to the Governor and General Assembly of the work accomplished from August 1, 1910, to December 31, 1910. The Governor of Illinois appointed on this Commission the following men:

Representing the United Mine Workers of Illinois: Charles Bennett, La Salle, Ill.; Charles Krallman, Glen Carbon, Ill.

Representing the Mine Operators of Illinois: John L. Schmidgall, Murphysboro, Ill.; J. W. Miller, Gillespie, Ill.

Representing the Mine Inspectors of Illinois: Hector McAllister, Streator, Ill.

Representing the Federal Bureau of Mines: J. A. Holmes, Washington, D. C.

Representing the Mining Department of the University of Illinois: H. H. Stoek, Urbana, Ill.

Immediately after the organization on August 2, 1910, the Commission held a conference with the state architect and authorized him to draw up plans for three rescue stations in accordance with suggestions submitted to him by the Commission.

After several meetings of the Commission in Chicago and Springfield, it was decided to locate the rescue stations at the following places: At La Salle, for the northern part of the state; at Springfield, for the central part of the state; at Benton, for the southern part of the state.

In order to secure suitable men for the positions of superintendent and assistants at the three rescue stations, Richard Newsam was appointed manager of the stations, and with the sanction of the Commission adopted a rule that only those holding an Illinois certificate as mine manager would be eligible for appointment as superintendent of a rescue station, and only those holding an Illinois certificate as mine manager or mine examiner would be eligible to appointment as assistant at a rescue station.

As a result of the preliminary theoretical and practical tests, the following appointments were made:

La Salle station: Superintendent, Thomas English, of Streator; assistant, Peter Donnelly, of Streator.

Springfield station: Superintendent, G. H. Walmsley, of Peoria; assistant, Charles Swan, of Danville.

Benton station: Superintendent, J. C. Duncan, of Murphysboro; assistant, Frank Rosbottom, of Peoria.

These men were trained at the Urbana station of the United States Bureau of Mines, and after 2 weeks were sent to the United States Bureau of Mines at Pittsburg, where they were given further training in rescue and first-aid methods. After returning to Urbana they were given additional training in first-aid to the injured.

During the period of their training all the men accompanied R. Y. Williams and James Webb, of the United States Bureau of Mines, to the Electric mine in Danville, Ill., to assist in fighting a mine fire. On December 10, three of the men accompanied Mr. Webb to Middletown, Ill., and assisted him in fighting a mine fire there. The men in charge of the Illinois stations therefore have had an unusually thorough theoretical and practical training.

An essential part of the rescue equipment at each station is the mine rescue car which stands on a switch at the station and is equipped with rescue apparatus and ready upon call to be taken to the scene of an accident as soon as a locomotive can be attached. Two of the cars were fitted out and contributed by railways, and the third was purchased for $750 from the Pullman company, and refitted at the railroad shops in Peoria.

The rescue stations are intended to serve two distinct purposes. First, to furnish a trained rescue corps of men to assist at a mine in case of accident. Second, to train men in the use of rescue appliances so that ultimately there will be at every mine in Illinois a corps of men who can enter a mine with a suitable rescue outfit.

Each station can accommodate from 12 to 15 men in training at one time, and the Commission recommends that this number should be at each station at all times receiving instruction in rescue appliances and rescue methods.

EARTHQUAKES AND COLLIERY EXPLOSIONS

F. Napier Denison, F. R. M. S., has interested the Victoria, B. C. Board of Trade in the establishment of an observatory to record seismic disturbances. From observations made by Mr. Denison an average of 76 earthquakes were recorded in Victoria yearly, and of these 36 occurred within a radius of 1,000 miles of Victoria. Mr. Denison stated that since 1898 he had kept a constant record of earth unrests on the local seismograph. He had been able to an extent, by the collection of statistics, to establish a certain relation between periods of great seismic unrests and colliery disasters. He pointed out that the greatest number of these disasters occurred when the pendulum reached its greatest westerly swing. In the records of correlation obtained the greatest westerly turn of the pendulum was during the months of November and December of last year, and coincident with this the greatest number of colliery disasters during the last 12 years occurred.

Of 114 explosions in North America, in which more than 5 lives were lost, 48 of these occurred upon the day or within 24 hours of some great earth unrest. Of 9 European explosions, in which similarly more than 5 lives were lost, 77 per cent. of these occurred on the day or within 24 hours of a great earthquake.

In a letter, T. R. Stockett, general manager of the Western Fuel Co.'s mines, at Nanaimo, said that the peculiar coincidence of Mr. Denison's records and colliery statistics warranted further investigation, and that the Dominion government might well extend the local facilities for studies of this important subject.

In elaborating on his theory Mr. Denison did not wish it understood that earthquakes were the direct cause of disasters in collieries. During the time of maximum disturbances he pointed out that the earth undergoing tremendous strains, and stress at different points would cause changes in rock strata that would lead to the escape of gases which would only need ignition to cause explosions and subsequent disasters.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

WESTERN EDITORIAL OFFICE: 728 Equitable Building, Denver, Colo.
CHICAGO OFFICE: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to MINES AND MINERALS, Scranton, Pa.

CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER............ MANAGER
EUGENE B. WILSON, SCRANTON, PA............ EDITOR
GEORGE F. DUCK, E. M., DENVER, COLO............ WESTERN EDITOR
P. G. MOORE............ CIRCULATION MANAGER
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., LTD., DRYDEN HOUSE, 43 GERRARD ST., SOHO, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 PITT STREET, SIDNEY.
SOUTH AFRICAN AGENT: E. HERBERT BYSSHE, CAPE TOWN, MANSION HOUSE; JOHANNESBURG, TRUST BUILDING; DURBAN, NATAL BANK CHAMBERS; BLOEMFONTEIN, FLORENCE BUILDINGS.

SUBSCRIPTION RATES

Single Copies............ 25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada............ 2.50
To all other countries............ 3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always gives your FORMER as well as your present address.

TABLE OF CONTENTS

PRODUCTION OF IRON ORE IN SPAIN, by H. A. McBride............ 577
MINING LAWS IN SANTO DOMINGO, by C. A. Haussler............ 580
MINES OF MISSOULA COUNTY, MONTANA, by J. P. Rowe............ 581
GENERAL ORE MINING NOTES............ 584
REPORT ON POVERTY GULCH MINE, by Charles W. Henderson............ 586
DR. R. W. RAYMOND'S RESIGNATION............ 588
DRY PLACER MINING MACHINES, by E. B. Wilson............ 589
HAND JIG FOR SMALL MINES, by J. M. Calderwood............ 591
THE SOUTH UTAH MINE AND MILL, by Leroy Palmer............ 592
THE CANADIAN MINING INSTITUTE MEETING............ 596
WORLD'S PRODUCTION OF ZINC IN 1910............ 596
DEVELOPMENTS IN CYANIDE PRACTICE, by Percy E. Barbour............ 597
UTILIZING ZINC TAILINGS, by Lucius L. Wittich............ 601
ELECTRIC MOTORS SAVE MINE............ 603
FINE CRUSHING WITH CONICAL MILL............ 604
CYANIDE TREATMENT OF ORES IN MEXICO............ 604
CONCENTRATION AT CALAMA, CHILE, by F. A. Sundt............ 605
THE GEOGRAPHY OF PETROLEUM............ 607
ILLINOIS RESCUE STATION COMMISSION............ 609
EARTHQUAKES AND COLLIERY EXPLOSIONS............ 609
EDITORIAL—
The Protection of Surface Over Coal Mines—The Lesson of the Price-Pancoast Disaster—The Prevention of Mine Accidents............ 610
BOOK REVIEW............ 612
OBITUARY—
Heber S. Thompson—Samuel F. Emmons—David H. Moffatt—John R. Perry—Walter Knight—Isaac Dawe—Joseph E. Evans—James B. Daniels............ 613
CORRESPONDENCE—
Clay Shrinkage—Size of Shaft Pillar............ 615
TECHNICAL SOCIETIES—
Kentucky Mining Association—American Mining Congress—West Virginia Coal Mining Institute............ 615
THE PRICE-PANCOAST DISASTER............ 616
THE SCRANTON MINE CAVE ENQUIRY............ 620
HAZLETON PLUNGER JIG............ 621
JEFFREY-GRIFFITH CROSS-OVER DUMP............ 622
PERSONALS............ 622
COAL DUST EXPLOSIONS, by John Verner............ 623
CATALOGS RECEIVED............ 628
GASOLINE MOTOR HAULAGE, by Geo. E. Sylvester............ 629
POCAHONTAS COAL AND COKE SHIPMENTS............ 630
BAILING WATER AT COLEMAN SHAFT, by F. Ernest Brackett............ 631
TRADE NOTICES............ 632
COAL TRUSTS AND SAFE MINING, by Sim Reynolds and W. H. Reynolds............ 633
MINING AT LETHBRIDGE, ALBERTA, by A. T. Shurick............ 635
ELECTRIC SHOCK IN MINES, by Sydney F. Walker............ 637
MODERN HOLMEN COALING STATIONS, by Clyde P. Ross............ 639
NOTES ON THE DELAGUA, COLO., EXPLOSION, by Geo. F. Duck............ 641
EXAMINATION QUESTIONS—
Hoisting Engineers, Des Moines, Iowa, October 12............ 643
NEW INVENTIONS............ 644
SQUIBS—
Aluminum Bronze, 584—American Mining Congress and Workmen's Compensation Law, 585—Poisonous Gases from Ferrosilicon, 591—Mercury in Russia, 600—No "Smoke Farmers" There, 603—Drifting Records, 606—Natural Gas Leakage in Coal Mines, 615—United States Mine Rescue Car No. 2 Schedule, 621—The Kruskoff Process, 630—Sectionalized Power Plant, 636—Properties of Concrete, 636—Coal in Chile, 638—Power Gas and Its Development, 640—Powder-Thawing Device............ 640

THE PROTECTION OF SURFACE OVER COAL MINES

OWING to the extent of the coal mining operations under the city of Scranton, and the value of the surface and buildings thereon, the question of the protection of the surface and surface improvements is such that on the initiative of Hon. J. Ben Dimmick, former mayor of the city, the city councils and the school board jointly engaged Messrs. William Griffith and Eli T. Conner, both well-known and able mining engineers to make an exhaustive investigation of the hundreds of acres of mine workings under the city, and to suggest plans for the proper support of the surface. In addition, the voluntary services of the following gentlemen were secured as an advisory board to supplement the suggestions of Messrs. Griffith and Conner: John Hays Hammond, W. A. Lathrop, L. B. Stillwell, D. W. Brunton, and Dr. R. A. F. Penrose, Jr.

Messrs. Griffith and Conner, after several months work and study, submitted a voluminous and comprehensive report to the advisory board, which recently met in Scranton, and on their approval of it, they submitted it to the mayor, councils, and school board. A summary of the conditions found to exist and the recommendations of Messrs. Griffith and Conner are published in another column. That the work of the two able engineers would be conscientiously and ably done was predicted by the writer when he with others familiar with their ability urged their appointment. That their report and maps are worth infinitely more to the city than the fees paid them, is a statement which we make without fear of contradiction from any mining engineer or mine official of experience and common sense.

The method suggested by Messrs. Griffith and Conner—flushing the worked-out areas full of culm, sand, gravel, ashes, or other available material—is what we predicted it would be. Of course there are details regarding the methods of treating various areas suggested in the report that could only be suggested after personal examinations of the areas in question by the engineers. These details, however, are among the most valuable portions of the report.

In their report Messrs. Griffith and Conner mention that the method of protection they suggest is largely and successfully used in France and Germany. They might also have called attention to the fact that it has been and is now being successfully used at Shenandoah, Pa., by the Philadelphia & Reading Coal and Iron Co. In fact, the late Mr. R. C. Luther, when chief mining engineer of that company, was the first man to suggest flushing anthracite mine workings full of culm to protect the surface, and the first work of the kind done anywhere, was in the western section of the town of Shenandoah, some 25 or 30 years ago, and the writer was at the time an assistant to Mr. John H. Pollard, resident engineer, under whose personal direction the surveys were made, the bore holes put down and the work was accomplished.

Descriptions of this operation, and of similar opera-

tions on somewhat smaller scales in other portions of the anthracite region were published in Mines and Minerals, read by subscribers in France and Germany, and the method was adopted in those countries. A peculiar and interesting fact in this connection is that in the former country the method is known as the *Scranton Method* probably because its use was first brought to the attention of the French engineers by this journal, published in Scranton.

THE LESSON OF THE PRICE-PANCOAST DISASTER

On another page we publish an absolutely correct account of the Price-Pancoast mine disaster with a map showing the affected mine area, ventilation plan, etc. It has been prepared with a mind free from prejudice and all sentiment has been eliminated, so that there might be a plain statement of facts that would serve a useful purpose in suggesting means for the prevention of other mine disasters.

There is no positive evidence as to what started the fire, and we base our belief that the coal oil lamp in the Dietz lantern exploded, on the fact that it was the only probable agent that could possibly start the fire.

Such lanterns have been used for many years in mines, just as coal-oil lamps have been used in homes. In proportion to the number of lamps used in homes, the number of explosions is very small. When the lamp is encased as in the Dietz lantern there is very much less chance for an explosion than there is in the case of an ordinary house lamp. Nevertheless there is a chance of explosion, and the starting of a mine fire which may soon get beyond immediate control. Therefore the use of coal-oil lamps, of any type, in mine workings should be deprecated. In fact no great hardship would be entailed if their use was prohibited by law.

Progressive mine managers are now making inside engine rooms, pump houses, etc., of steel and masonry, masonry, brick, or concrete. However, there are in practically all large mines, engine rooms, pump rooms or offices for foremen, similarly constructed to the engine room at the Price-Pancoast. Such engine or pump rooms are safer from fire than those built entirely of wooden planks, and we do not recollect another instance in which a fire starting in such a room caused so great a conflagration or so much loss of life. It is therefore evident, that if an engine room such as that at the Price-Pancoast mine could, through a few unfortunate circumstances, initiate such a fire, the more flimsy shanties sometimes used in mines as engine rooms, pump rooms, and offices, are an element of great danger, and their use should be terminated as quickly as possible. It is not always practicable to construct inside engine and pump rooms and offices of masonry, brickwork, or concrete, and it is not always necessary to do so, but it should be done wherever it is practicable, and the flimsy plank or board shanties should be prohibited.

THE PREVENTION OF MINE ACCIDENTS

In an address recently made at the Case School of Applied Sciences, in Cleveland, Ohio, Dr. Joseph A. Holmes, Director of the Federal Bureau of Mines, endorsed strongly the position maintained for many years by Mines and Minerals, in regard to the selection of state mine inspectors. In the course of his address Doctor Holmes said: "The selection of state mine inspectors by popular vote must be stopped if there is to be a reduction of accidents in the coal and metal mines of the United States. The state mine inspectors should have greater permanence in office and freedom from political and other outside influences. Their selection and continuance in office should depend upon their training and experience. They should be examined by a non-partisan board of mining men. They should be appointed upon the recommendation of such a board from the applicants that have shown the highest skill and best experience. Under no circumstances should they be selected by popular vote. In other words, politics should have nothing whatever to do with their selection or their continuance in office. The inspectors should have better support in the way of compensation. In fact, the salary and other conditions should be such as to enable the state to secure the best possible type of men for this important work."

He also recommended "the use at each mine of a limited number of men well trained and experienced concerning the best methods of using explosives, electricity, the handling of gases and coal dust, the methods of timbering with a view to preventing falls of roof, the methods of preventing and extinguishing mine fires, and the methods of rescue and first-aid work. These trained men can serve to good advantage as special inspectors or foremen.

"There must be active, determined cooperation between the miners and the mine management and the state mine inspectors in the enforcement of the mine rules and regulations, and the punishment of every person, whether mine worker or mine manager, who disobeys these rules and regulations."

We heartily concur in all that Doctor Holmes says regarding the selection of state mine inspectors and their compensation. In regard to the latter, we have always claimed that the salaries of the state mine inspectors should be at least $3,000 per year, if men of proper caliber are to be secured for the positions and retained in the service of the state. In many instances men of superior ability as mine inspectors have resigned their positions to accept superintendencies of mines at larger salaries than the inspectorship paid. In other words, mine owners paid more for real mining ability than the state did, and as a result the state lost the services of men who should have been retained as inspectors.

Doctor Holmes' statement as to the desirability of a limited number of well trained and experienced men at each mine is also sound. Such men are located at

each mine in states requiring the employment of certificated mine foremen, assistant foremen, and fire bosses, but unfortunately, many mining states do not have laws requiring the employment of such men in official positions. When the states without such laws enact and enforce them, a great step toward remedying present conditions will have been taken. With first-class inspectors and properly qualified and certificated mine officials, the cooperation between the miners, the mine management, and the state mine inspectors, in the enforcement of rational mine rules and regulations, which Doctor Holmes rightly considers essential, can easily be secured.

BOOK REVIEW

Motion Study, by Frank B. Gilbreth. Published by the D. Van Nostrand Co., New York. The price of this book is $2 net. According to the writer tremendous savings are possible in the work of everybody. They are not for one class, they are not for the trades only, they are for the offices, schools, colleges, stores, households, and the farms; but the possibilities of benefits from motion study in the trades are particularly striking, because all trades, even at their present best, are badly bungled. At first glance the problem of motion study seems an easy one. After careful investigation it is apt to seem too difficult and too large to attack. There is this to be said to encourage the student however: First, the study of one trade will aid in finding the result for all trades. Second, work once done need never be done again. The final results will be standards. The book is divided into five chapters and is an interesting book in that it shows how workers may not only lessen their work but increase their capacity at the same time.

Elementary Chemistry for Coal-Mining Students, by L. T. O'Shea, M. Sc., Sheffield, B. Sc., London, Professor of Applied Chemistry in the University of Sheffield. The author has been delivering popular and more or less systematic courses of lectures to mine foremen and other coal-mine workers, who have little or no knowledge of chemistry, but are desirous of learning something of the subject which may be useful to them in their daily occupation. The work has been prepared with the object of meeting the wants of these as well as other students of coal mining. It is really a compilation of those parts of chemistry, pure and applied, that are cognate to the coal-mining industry and does not pretend to be a textbook of chemistry. While there is considerable of higher chemistry in the book, there is much which is elementary and practical. There are 15 chapters and an appendix, in all 319 pages. The following is a synopsis of the subjects treated: The Air or Atmosphere, Pressure of a Gas and Atmospheric Pressure, Water and Hydrogen, Atoms and Molecules, Sulphur, Carbon, Flame and the Safety Lamp, The Oxides of Carbon, Coal, By-Products and Their Recovery, Explosives, Explosion in Gases, The Effect of Temperature and Pressure on the Density and Volume of Gases, and the Diffusion of Gases, Chemical Calculations, Useful Tables, Index. The price of the book is $1.80 net.

Elements of Geology, by Eliot Blackwelder, Associate Professor of Geology, University of Wisconsin, and Harlan H. Barrows, Associate Professor of General Geology and Geography, University of Chicago. Published by the American Book Co., New York, N. Y. The book contains 475 pages and 485 illustrations. The price is $1.40. There are 25 chapters and an index and the book is divided into two parts—Physical Geology and Historical Geology. It is exceedingly well illustrated, and the illustrations help much toward aiding the student in geology to an understanding of the subject.

Biennial Report of the State Geologist, by H. A. Buehler, Director of the Missouri Bureau of Geology and Mines, Jefferson City, Mo. The investigations made in the preparation of this report are, coal deposits, iron ores, lead and zinc deposits of the Aurora area, geology and structural materials of Jackson County, the geology of Phelps County, and ore deposits of the southern Ozark region.

Forestry Wood Industries, Vol. V, 1911, of the West Virginia Geological Survey, by A. B. Brooks; also Bulletin No. 2, 1911, Levels, Coal Analyses. While the former book pertains to the lumber business, coal mining and lumbering are so closely connected in many instances that the subject is of interest to both industries. The book teems with information, and Mr. White is to be congratulated on obtaining Mr. Brooks as forester on this subject. "Bulletin Two" is a cloth-bound book of 385 pages, divided into two parts, the first giving levels in the various parts of the state and the other analyses of the coal beds in the different geological horizons. A list of the publications of the West Virginia Geological Survey and the prices can be had by addressing the Secretary of State, Charleston, or Dr. I. C. White, Superintendent of the West Virginia Geological Survey, Morgantown, W. Va.

BOOKS RECEIVED

United States Geological Survey Publications, Washington, D. C. Bulletin No. 445, Geology of the Pegmatites and Associated Rocks of Maine, including Feldspar, Quartz, Mica, and Gem Deposits, by Edson S. Bastin; The Production of Iron Ore, Pig Iron, and Steel, by Ernest F. Burchard; The Production of Sand-Lime Brick in 1909, by Jefferson Middleton; The Production of Gems and Precious Stones in 1909, by Douglas B. Sterrett.

Bulletin of the Bureau of Labor for November, 1910, Department of Commerce and Labor, Washington, D. C. Working Hours of Wage-Earning Women in Selected Industries in Chicago, by Marie L. Obenauer; Labor Laws Declared Unconstitutional, by Lindley D. Clark; Old Age and Invalidity Pension Laws of Germany, France, and Australia; Review of Labor Legislation in 1910, by Lindley D. Clark; Laws of Various States Relating to Labor Enacted Since January 1, 1910.

Proceedings and Collections of the Wyoming Historical and Geological Society, Vol. XI. Published at Wilkes-Barre, Pa. The price of the book is $3, and it can be had by addressing Rev. Horace Edwin Hayden. Among the interesting articles in this publication are the reminiscences of Rev. Jacob Johnson, M. A., first pastor of the First Presbyterian Church, Wilkes-Barre, 1772-1790. Another interesting

article is the influence of the Iroquois on the history of archaeology of the Wyoming valley and the adjacent region, by Arthur C. Parker.

Biennial Report of the State Geologist of North Carolina, by Joseph Hyde Pratt, Chapel Hill, N. C. The report is a recapitulation of the work done in 1909 and 1910 on highways, hydrography, forestry, magnetic survey, survey for the North Carolina Fishing Commission, and topographic and traverse mapping of the state.

Report of the Mining Investigation Commission of the State of Illinois to Hon. Charles S. Deneen, Governor of Illinois. The report of this commission covers 68 pages and deals principally with bills revising the laws of the state of Illinois regarding coal mining. It also includes a bill to amend an act entitled "An Act in Relation to Sinking, Filling, and Operating of Oil or Gas Wells," approved and put in force May 16, 1905. It also includes an act to amend an act to require fire-fighting equipment and other means for the prevention and controlling of fires and the prevention of loss of life from fires in coal mines, approved March 8, 1910, and put in force March 8, 1910; also a bill to amend an act entitled "An Act to Establish the Mining Investigation Commission of the State of Illinois, and Prescribing Its Powers and Duties and Making an Appropriation Therefor," approved June 10, 1909, and put in force July 1, 1909.

Annual Meeting of the West Virginia Mining Association, Washington, D. C. Address, Neil Robinson, secretary, Charleston, W. Va.

Iowa Geological Survey, Vol. XX. The annual report for 1909 includes the Geology of Butler County, by Melvin F. Arey; Geology of Hamilton and Wright Counties, by Thomas H. MacBride; Geology of Iowa County, by Stephen W. Stookey; Geology of Wayne County, by Melvin F. Arey; Geology of Poweshiek County, by S. W. Stookey; Geology of Harrison and Monona Counties, by B. Shimek; Geology of Davis County, by Melvin F. Arey. This book can be had by addressing the State Geologist, Des Moines, Iowa.

Mine Rescue Station Commission of the State of Illinois, Report to the Governor and General Assembly of the work accomplished from August 1, 1910, to December 31, 1910, Springfield, Ill.

State Geological Survey, Bulletin No. 16, Oil Resources of Illinois With Special Reference to the Area Outside of the Southeastern Fields, by Raymond S. Blatchley, University of Illinois, Urbana, Ill.

Preliminary Report on the Mineral Production of Canada During the Calendar Year 1910, John McLeish, B. A., Chief of the Division of Mineral Resources and Statistics, Ottawa, Can.

Mineral Production of Ontario for 1910, Bulletin No. 7, Bureau of Mines, Toronto, Ontario.

Department of Mines, Mines Branch, Ottawa, Can. Eugene Haanel, Ph. D., Director, Ottawa, Can. Magnetic Concentration Experiments With Iron Ores of the Bristol Mines, Quebec; Iron Ores of the Bathurst Mines, New Brunswick; A Copper-Nickel Ore From Nairn, Ontario. Bulletin No. 4, second edition, Investigation of the Peat Bogs and Peat Industry of Canada During the Season of 1909-1910, by Aleph Anrep, Jr.

An Act to Consolidate and Amend the Coal Mines Regulation Act and Amending Acts, Bill No. 3, 1911. Address the Minister of Mines, Victoria, B. C.

The Carnegie Foundation for the Advancement of Teaching, Fifth Annual Report of the President and of the Treasurer for 1910. This report may be obtained by writing to The Carnegie Foundation, 576 Fifth Avenue, New York City.

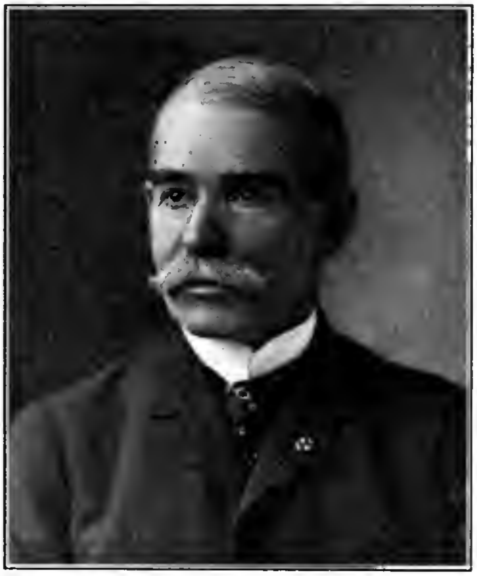
Chrysotile-Asbestos, by Fritz Cirkel, M. E. issued by the Canada Department of Mines, Eugene Haanel, Ph. D. Director, Ottawa, Canada. It contains 300 pages, 66 photoengravings, 88 drawings, two maps of Quebec, and is the only complete treatise on chrysotile, its occurrence, exploitation, milling and uses. It is most practical technical report.

OBITUARY

MAJOR HEBER S. THOMPSON

Major Heber Samuel Thompson, chief engineer of the coal lands of the Girard estate, located in Schuylkill and Columbia counties, Pa., died at his residence in Pottsville, Pa., on Thursday, March 9, in the seventy-first year of his age.

Major Thompson was a lineal descendant of James Thompson, who with his brother John, emigrated from County Antrim, Ireland, and settled at Cross Roads, Chester County, Pa., and was a grandson of William Thompson, who as a young man served as an American soldier in the Revolutionary war. Samuel Thompson, Major Thompson's father, was for many



MAJOR HEBER S. THOMPSON

years a successful hardware merchant in Pottsville, where the latter was born, the youngest of four children, on August 14, 1840. He spent his whole life in Pottsville, except his college years and the years he devoted to the service of his country as a gallant soldier and officer during the Civil war.

His preliminary education was secured in the public and private schools of Pottsville, after which he entered Yale University, where he pursued a classical course. He was one of a large number of patriotic Yale students, who at the outbreak of the war enlisted in the Union army and rendered valuable service to their country. Notwithstanding Major Thompson was in his senior year and within 3 months of graduating, he felt he owed his country his services and enlisted as a private in the Washington Artillerists of Pottsville, and with his company he won the distinction of being among the "First Defenders," or the first troops to reach Washington in response to President Lincoln's call for 75,000 volunteers. While in service, he was granted a furlough to return to Yale, participate in the graduating exercises of his class and receive his diploma with the degree of Bachelor of Arts. Ten years later he received from the same institution the honorary degree of Master of Arts. His first enlistment was, in accordance with the President's call, for 3 months, and he was mustered out of service on July 29, 1861. He reentered the service on October 22, 1861, as First Lieutenant of Company F, Seventh Pennsylvania Cavalry, and on July 1, 1863, he was promoted to the captaincy of Company I of the same regiment.

Major Thompson had an especially honorable career in the war. He participated in all of the campaigns of his regiment,

receiving special mention in orders for distinguished service at the battles of Sparta and Shelbyville, Tenn., Chickamaugua, and Lovejoy Station, Ga. His regiment is classed by Colonel William F. Fox in his "Regimental Losses of the Civil War" as one of the 300 "Fighting Regiments." On March 18, 1864, Captain Thompson was placed on detached service, acting as Inspector General of the First Brigade, Second Cavalry Division, Army of the Cumberland. He was taken prisoner at Lovejoy's Station, Ga., April 20, 1864, released on parole at Charleston, S. C., December 20, 1864, and finally discharged under the General Orders of the War Department, issued January 24, 1865. While still on parole, he was promoted to major, but as he could not take up active service because of his parole, he did not accept the commission.

On his return from the army, Major Thompson, then a young man of 25, felt that with his educational advantages, he was called on to devote his talents to a broader field than was offered by connection with the hardware business established by his father, and then conducted by his elder brother. He accordingly secured a position on the engineering corps of Harris Bros. (a firm composed of the late Stephen and Joseph Harris), at that time ranking as one of the ablest engineering firms in the country.

On the death of Stephen Harris, who for a number of years had engineering charge of the Girard estate, Major Thompson was in 1874, appointed his successor, and he filled the position to the greatest satisfaction of the board of managers of the great charitable trust, and with honor to himself.

An evidence of the responsibility of the position can be shown by the fact that of over 55,000,000 tons of anthracite coal shipped to market from the coal lands of the Girard estate, over 48,500,000 tons were shipped during the 37 years of Major Thompson's service as chief engineer.

As the collieries on the Girard estate are operated by lessees, Major Thompson was not in charge of the actual mining operations, but it was his duty to see that those operations were carried on in accordance with the terms of the leases and in a manner which would conserve the property. In this work he was in constant intercourse with the officials of the Philadelphia & Reading Coal and Iron Co., the Lehigh Valley Coal Co., and other less important lessees, and those officials were always willing to cheerfully accord Major Thompson high rank as an able mining engineer. His work in the construction of water works and the securing of a plentiful supply of water for the collieries, towns, and villages on the estate, marked him also as a civil engineer of high ability.

As a man and citizen the writer's acquaintance with Major Thompson extended over a period of nearly 40 years, beginning when in early boyhood he first met him. To attempt to describe in adequate words his nobility of character, his lovable manliness, his personal integrity, his charity, which not only covered the assistance of the poor, but covered also his personal relations with people of all types, his sympathetic encouragement and assistance to those who needed it to raise them from depths into which they had fallen, and thus enable them to rise to new respectability, and withal his positive convictions on all things he believed to be true and right, and his ability to oppose successfully methods or ideas that were wrong without antagonizing his opponents, but rather increasing their respect for him, would require very much more space than is available in this journal.

Major Thompson at the time of his death was president of the First Defenders' Association, a member of the G. A. R., of the Union Veteran Legion, and of the Loyal Legion of the United States. He was for a number of years president of and a member of the Board of Trustees of the State Hospital at Ashland, Pa., and was also a member of the Board of Trustees of the Pottsville Hospital. He was a member of the county visiting committee of the State Board of Charities and a member of the State Committee on Lunacy. He was also an officer and director of several important local fiduciary corporations.

The scientific and historical societies, of which he was a member, were the American Institute of Mining Engineers, the American Philosophical Society of Pennsylvania, the Historical Society of Pennsylvania, and the Historic Society of Schuylkill County. In religious belief he was a Presbyterian, and was for 37 years an elder of the First Presbyterian Church of Pottsville. His religious belief, while conforming strictly to Presbyterian doctrine, was so broad that he recognized and respected the religious views of all Christian denominations and in his life he made doctrines subservient to true broad Christianity and charity. A true patriot, a consistent Christian, an able engineer, a brave and gallant soldier, who when the war was over held no feeling but that of charity for a defeated foe, he was in fact a man.

SAMUEL F. EMMONS

Samuel Franklin Emmons, geologist, born in Boston, Mass., March 29, 1841, died in Washington, March 28. He received the degree of A. B., Harvard, 1861; A. M., 1866; studied at Ecole Imperiale des Mines, Paris, 1864; Frieberg, 1865; and received the degree Sc. D., Columbia and Harvard, 1909. He was geologist United States Geological Survey exploration 40th parallel, from 1867-1877; geologist United States Geological Survey, 1879; and treasurer National Academy of Sciences since 1902.

His contributions were always welcomed by mining men and geologists, whether they were official reports or written for journals, because of the conscientious way in which the subjects were treated. Few, if any, have done more for the metal industry of the West than the late S. F. Emmons.

DAVID H. MOFFATT

David Halliday Moffatt, banker, died in New York City, March 18, aged 71 years. He was born at Washingtonville, Orange County, N. Y., in 1839.

He became the first cashier of the First National Bank in Denver, Colo., in 1865, and soon after was made president, which position he held until death.

He was one of the leaders in the construction of the Denver & Rio Grande Railroad and was president from 1884-1891. He built the Florence & Cripple Creek Railroad, mostly at his own expense, and at the time of his death was building the "Moffatt" road, to open an important mining area in Colorado. At one time he was largely instrumental in developing Leadville, Creede, Aspen, and Telluride, and while some of his mining ventures were not profitable, others were so successful he was held up as an example by western mine promoters. Dave Moffatt's money, coupled with his mining partner's brains, did much to develop Colorado.

PRICE-PANCOAST MINE HEROES

John R. Perry, miner, and Common Councilman of the First Ward, Scranton, Walter Knight, foreman, and Isaac Dawe, fire boss, died in the Price-Pancoast mine Friday, April 7, 1911. These three men are the heroes who lost their lives attempting to save those of their fellow workmen.

Joseph E. Evans, foreman of the United States rescue car stationed at Wilkes-Barre, Pa., lost his life at the Price-Pancoast colliery fire Friday, April 7, 1911. Mr. Evans was 39 years old, married, and a resident of Scranton, where he was born and respected all his days. He was a member of the Melita Commandery, No. 68, Knights Templar, and was buried by brother knights from his own and other commanderies at the Washburn Street Cemetery, Scranton. Mr. Evans' life is summed up in the words: "Conscientious in all things."

JAMES B. DANIELS

James B. Daniels, manager of the Portland Mills, the man who is said to have given Benjamin and Samuel Guggenheim their first insight into smelting, died at the St. Francis Hospital, Colorado Springs, Colo., April 3. Mr. Daniels was 51 years old and died from the effects of an operation for appendicitis.

CORRESPONDENCE

Clay Shrinkage

Editor Mines and Minerals:

SIR:—It is probably quite incorrect to attribute the settlement of St. Paul's Cathedral, London [see MINES AND MINERALS, March, 1911], to the construction of the tube railways. Nor is it quite correct to describe the London clay as only 64 feet thick under London. The annexed table gives the thickness of this clay at a few places in London. In addition to the London clay proper, there are beneath it certain colored or mottled plastic clays known as the Reading clays, from their development and outcrop near the town of Reading. These clays are variously mottled in different layers of red, blue, purple, yellow, and these colors persist well in dried specimens. Then, beneath the Reading beds come alternations of sand, clays, pebbles, loose and conglomerate, known as the Woolwich beds, and below these comes a bed of greenish sand—the Thanet sand which lies directly upon the chalk. The chalk is the stratum from which most of the many hundred Artesian wells draw their supply in London. Not many years ago some of these actually had a natural overflow. One a few miles out near Wimbledon, drove an overshot waterwheel. That at the Bishop's palace, at Fulham, overflowed. The natural water level of all the chalk water was of necessity once higher than mean sea level. Today it is over 100 feet below sea level, owing to the many years pumping of the many wells, and every year sees an additional foot of lowering, and this is likely to increase until the external hydrostatic head becomes so great that the influx of tidal water, already believed to be in progress, in the east end of London, will become so great as to spoil all the wells, and put a stop to most of the pumping. When this time comes the water level will rise again—it is, and always has been, highest on Monday morning—and the salt water will be pushed out and some wells may be pumped again when this has gone on long enough to discharge the salt. St. Paul's has settled by reason of the shrinkage of the clay beneath it. Arches which run in the crypt from the great piers which carry the dome have sheared off, I am told, some inches vertically because of the greater settlement of the heavier load.

TABLE SHOWING THICKNESS OF STRATA AT VARIOUS PLACES UNDER LONDON

Stratum	General Post Office	Shadwell, East End	Shoreditch, Northeast	Bank of England	Bayswater, West	Oxford Street	Bethnal Green, East	Blackfriars	Chiswell St., City	Cripplegate, East	Fulham, West	Hampstead
	Ft.	*Ft.*	*Ft.*	*Ft.*	*Ft.*	*Ft.*	*Ft.*	*Ft.*	*Ft.*	*Ft.*	*Ft.*	*Ft.*
Surface gravel	86	38	18	26	35	33	68	39	26	23	25	
London clay	104		60	111	176	58	68	89	62	71	135	289
Reading beds / Woolwich beds	47	55	40	59	53	55	23	53	89	56	70	40
Thanet sands	62	55	39	39	30	14	43	35	32	16	20	49
Into chalk	388	274	106	100	90	153	166	152	150	110	66	72
Totals	647	419	263	335	389	313	300	368		276	316	450
Surface above O. D.	58			39	87			14		55	15	230

Naturally there was an upward pressure of chalk water below the London clay. Above the London clay there still lies water in the superincumbent layer of about 30 feet of river gravels. Thus the upper clay is still moist; but below, the removal of the deep water, which now probably lies everywhere below or near the upper surface of the chalk, has allowed the whole body of clay and sand to dry out and shrink, and in the writer's opinion, this is the cause of the settlement of the cathedral. Indeed the settlement is much too severe in height to have been quickly caused by the railway work. In driving the tubes of these tunnels the small space behind the cast-iron tubes was very quickly pumped full of grout and the tubes went through very dry clay, as might be seen by watching the carts which carried it away from the shafts. Nor was there any emptying of such a sand pocket as would affect the cathedral. The writer is thus of opinion that the cause must be sought in the removal of upward water pressure and water contact with the sands and clays overlying the chalk. This depletion has been going on many years, but the drying out of the London and succeeding clays only began to operate with serious intensity when this lower surface was left high and dry above water level. The accompanying well records show a curious uniformity in the combined thickness of the Reading and Woolwich beds. The Thanet sand, a seashore sand, thins out in the west and thickens in the east country. About 1,100 feet down, ancient rocks supposed to be Devonian are found. This fact gave rise to the idea that there might be coal under London, or between London and the known coal measures across the sea in Belgium and France. This has since been proved and coal will shortly be raised in Kent.

Thus at 1,100 feet deep the anticipated sequence, as shown by the outcrops to the northwest of London, is missing to the extent of the breadth of the country. London's position was once dry land against which were deposited the lower green sand and other rocks, such as the oolites, new red marls, etc., whilst the Gault clay and the chalk and London clay were laid down at a higher level when perhaps the old rocks had sunk beneath the waves.

W. H. BOOTH, F. G. S.
19 Chatsworth Row, West Norwood, London, N. E.

Size of Shaft Pillar

Editor Mines and Minerals:

SIR:—Is there any rule for finding size of pillars of coal commensurate with the depth of shaft. For instance, a shaft 200 feet deep, 6-foot coal, rock roof, fireclay bottom, has pillars 40 feet wide; under the same circumstances what size pillars would be required for a depth of 400 feet or 600 feet?

W. L. MORGAN

TECHNICAL SOCIETIES

Prof. H D. Easton, Lexington, Ky., has issued a call to the coal operators of Kentucky to meet in conference in Lexington for the purpose of organizing a Kentucky Mining Association. The object of the association is to promote the interests of mining men and improve mining conditions through cooperation in Kentucky. The West Virginia Mining Association was organized three years ago, and is an influential factor in the mining affairs of that state. A similar association in Kentucky would produce more effective results than where individuals act alone. The meeting will take place May 29.

The next and fourteenth annual meeting of the American Mining Congress will take place in Chicago sometime in October, 1911.

The West Virginia Coal Mining Institute will hold their summer meeting June 12, at White Sulphur Springs.

NATURAL GAS LEAKAGE IN COAL MINES

Two coal mines in West Virginia were recently sampled and analyzed for natural gas leakage apparently due to a natural gas well passing through a barrier pillar. Gas was found bubbling up through the water, which, upon examination, corresponded with the natural gas obtained from the well. Samples of mine air vitiated by explosives fired in mines were examined and found to contain in some cases harmful amounts of noxious gases. Samples of natural gas from Nevada were examined, and in one case found to be exceedingly rich in methane. A sample of natural gas obtained from Washington contained 97 per cent. of nitrogen.—*Bureau of Mines.*

THE PRICE-PANCOAST DISASTER

Written for Mines and Minerals

Account of a Mine Fire Which Caused the Death of Seventy-Three Men

The Price-Pancoast colliery which, on April seventh, was the scene of the second greatest disaster in the history of the Anthracite Coal Regions, is owned and operated by the Price-Pancoast Coal Co., and is located in the borough of Throop, Lackawanna County, Pa., about 3½ miles northeast of the city of Scranton.

The coal seams worked are those locally known as the Diamond, the bottom split of the "Fourteen Foot," the Clark, the Dunmore No. 2, and the China seams. All hoisting of coal is done from a four-compartment shaft, 750 feet deep to the Dunmore No. 2 seam. The two end compartments of this shaft are used as upcasts, and there is an exhaust fan 35 feet in diameter on one, and another 20 feet in diameter on the other. In addition there is an exhaust fan 20 feet in diameter on another air-shaft sunk from the surface to the Diamond seam. The downcasts are the two hoisting compartments of the main shaft, the supply shaft, an old slope, and a special downcast located on the hill, some distance from the main shaft and breaker. The accompanying map, which shows the mine workings directly connected with the disaster, shows the main hoisting shaft at A and the supply shaft at B.

The colliery is one of the oldest, largest, and best-equipped mines in the Lackawanna region. Its annual production exceeds 600,000 tons. Besides extensive live workings there are extensive areas of worked-out territory.

The mine is a gaseous one, but with the three fans supplying an aggregate of 365,000 cubic feet of air per minute, and that air conducted through the workings in numerous splits and in a most practical and commendable manner, the gas is so diluted and carried off that open lights are generally used, though safety lamps are frequently used at the working faces, when the amount of gas given off makes their use advisable. The workings in the Diamond seam and the bottom split of the "Fourteen Foot," are practically separate from those in the Clark, Dunmore No. 2, and China seams, and the system of ventilating those workings is separate and distinct from that ventilating the workings in the last-mentioned seams, and is effected by the 20-foot fan on the new air-shaft, which yields over 130,000 cubic feet of air per minute. The coal from these workings is, however, hoisted from the main shaft. There are 400 men and boys usually employed in these workings, and about 500 in the other workings.

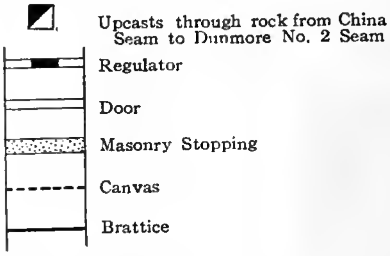
As on several occasions in the past, mine fires of minor importance had occurred through the lighting of blowers of gas, all the working sections of the mine are equipped with a water-pipe line. This pipe line consists of a 3-inch line down the shaft and 2-inch lines into every gangway or main haulage road, with plugs at the entrance of each chamber to which a hose can be attached.

The mine is under the personal direction of Mr. Joseph V. Birtley, general superintendent, and he is assisted by two mine foremen, three assistant mine foremen, and eight fire bosses and assistants. Mr. Birtley has been employed at the mine for over 20 years, by the former owners as general mine foreman, and by the present owners as general superintendent. He is a man of superior intelligence and executive ability and has a deservedly high reputation as a mine official of broad and successful experience.

As the workings affected by the fire were of comparatively limited area, and were in the China seam and connected directly with the Dunmore No. 2 seam only, the map shown herewith, for sake of clearness and because also even a greatly reduced map of all the workings could not be shown on one of our pages, shows only the China seam workings and their connections with the Dunmore No. 2 seam and the main and supply shafts.

The workings in the China seam are shown with solid lines, and those in the Dunmore No. 2 seam with dotted lines.

The course of the air is shown by arrows and the various upcasts to the return air in the Dunmore No. 2 seam, regulators, doors, masonry stoppings, wooden brattices, and canvas sheets, are shown by the following marks:



The mine workings are also equipped with telephones, connected with each other and with a phone in the mine offices on the surface.

All the fatalities occurred in what are known as the Tunnel Workings in the China seam. This seam ranges from 3 feet 6 inches to 4 feet thick and underlies the Dunmore No. 2 seam, with 20 feet of rock between. The China seam is reached by a slope from the Dunmore seam, through rock, and the workings from this slope are at a lower level than the "Tunnel Workings." These latter are reached by a tunnel from the shaft level of the Dunmore No. 2 seam, and as the dip of the seam is only 3 degrees to the north, a main haulage road with a parallel opening runs south, as shown on the map, and from it gangways, as shown, were turned east and west.

On the day of the accident there were 88 men and boys employed in the "Tunnel Workings," 16 of whom escaped, leaving 72 who, with Joseph E. Evans, foreman of the United States Bureau of Mines Rescue Corps, lost their lives.

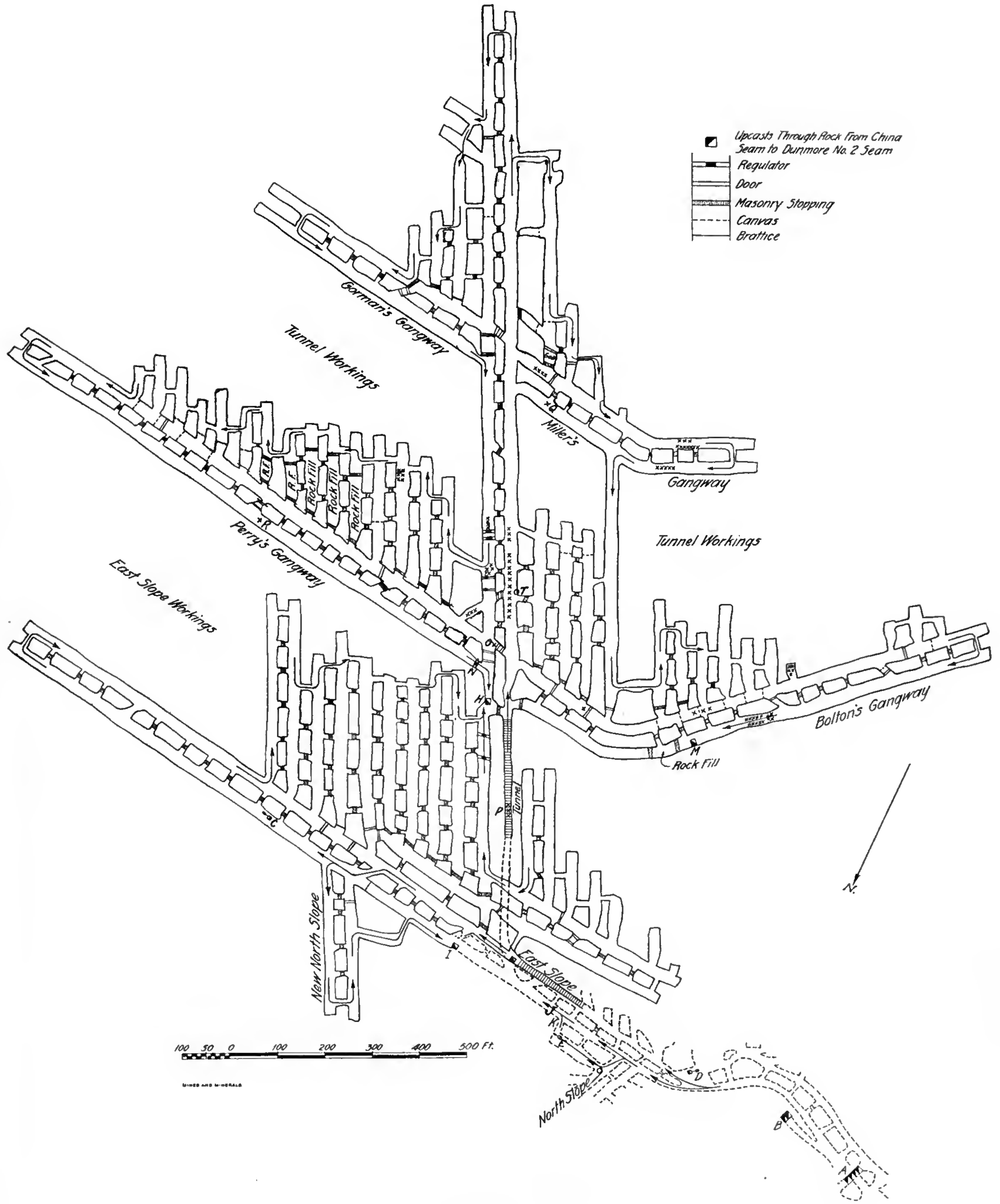
By reference to the map it will be seen that near the main shaft there is a slope in the Dunmore No. 2 seam known as the "North slope," and about 200 yards east of this slope there is one in the China seam known as the "New North slope." These slopes have a dip of from 3 to 5 degrees and are not very long.

As the maximum daily hoist from the "North slope" was but 35 mine cars per day, and from the "New North slope" in the China seam, where rock work only was being done, was 10 mine cars, and as the slopes were not far apart, and the amount of hoisting from both was not enough to occupy a total length of time of 2 hours per day, but one hoisting engineer, James Moran, an intelligent man of about 30 years of age was employed.

The steam for the hoisting engines, C, at New North slope, E, at North slope and the tail-rope engines D, nearer the main shaft, is carried down the main shaft and along the mine passages in well-covered steam pipes, and the exhaust steam from the engines is conducted by pipes to the upcast compartments of the main hoisting shaft.

The engine room E at the North slope is 20 feet long, 15 feet wide, and 7 feet high in the clear. It is cut in the coal and rock, and the roof is supported by sets of 8″×10″ squared yellow pine timbers, each set consisting of two legs and a collar. These sets were from 3 to 4 feet apart. There was also a small wooden platform for the engineer to stand on, and a small closet, or cupboard, in the room. The rope from the drum passed out through the opening shown to a sheave at the head of the slope where it was deflected to the line of the slope. Leading from the drum to the sheave there was a line of planks, under the rope, supported by ordinary mine props from 6 inches to 8 inches in diameter.

The engine room was lighted by a Dietz lantern. This was a large square lantern with three glass sides, containing a tin coal-oil lamp, with chimney, and a reflector back of the lamp to throw the light on the engine and drum.



Map of China Seam Workings, Price-Pancoast Mine

The engines were of the geared type, and, as is usual with this type of engines, considerable oil was used on the gears, and, though the room was kept exceptionally clean, some oil naturally splashed on the timbers.

Shortly after 8 o'clock on the morning of April 7, Moran, the engineer, after hoisting a trip from the North slope lowered the flame in the lantern and went to the New North slope engine, *C*, to do some hoisting there. He had only been there a few minutes when he went back to engine room *E*. In his testimony Moran estimates that he was occupied from 20 to 25 minutes in doing this, and that when he reached engine room *E* he found the interior in flames. A water pipe with a tap was in the engine room and he made an effort to get at it and open it, but was driven back by the flames. He then hurried to the tail-rope engine *D* where there was another tap, opened it, and secured water to play on the fire and called for assistance

William Mitchson, the engineer in charge of the tail-rope engines testified that this was about 8:20 or 8:30 and that he called up six telephone stations in the mine, but got an answer from but one, that in the Tunnel Workings in the China seam shown at *T*. Mike Kozey, the old door man, located near this phone, received the message and immediately started a trip rider named Bray to warn the men in "Bolton's gangway," while he went into "Perry's gangway." Kozey having met Perry and his laborers, left the Tunnel Workings with the laborers and escaped through the slope workings in the China seam. Bray, however, was lost with the seventy-one others.

In the meantime, the air-current used to ventilate engine room *E* carried the flames through the cross heading *K* to 15 empty mine cars standing on the empty track of the turnout *J*. The air-current passing through this turnout is normally about 48,000 cubic feet per minute, and this naturally made the fire more intense.

All this time men were at work endeavoring to extinguish the fire in the engine room, and as they had no possible way to get past or through the smoke, they were not for some time aware of the fire having reached and ignited the mine cars.

Mr. Birtley, when the fire broke out was in his office on the surface, and though not feeling well, was attending to his duties. As soon as he received word of the fire by telephone he went down the shaft, and reached the scene of action a few minutes before 9 o'clock. His first inquiry was whether the men had been notified, and on being told of Mitchson's telephoning, he, knowing the ventilating system so well, knew that all that was possible at the time had been done to warn the men in the Tunnel Workings of the China seam, which was the only section which would receive much smoke, bent all his energies and those of his assistants to extinguish the fire. He soon discovered that the fire was gaining on them and had reached the empty mine cars. Finding that the majority of the men working in the Tunnel Workings of the China seam had not come out, Fireboss Jos. Vickers and Edward Reed started to go in to them by way of the return airway, as it was impossible to get past the fire to go in the intake. In fact, even if they could have passed the fire, the smoke and gas in the intake would have prevented their entering. They had not gone far until they were driven back by the smoke and gas, which by this time was thick in the return air, and Vickers fell. He was taken to the main shaft at once and was with difficulty revived, and later taken to the surface. At the same time a call was made for outside assistance. The local Throop Hose Co. responded and soon had line of hose down the shaft, but as soon as the water was turned on the hose burst owing to the length of the water column. This caused considerable delay, but shortly after noon new hose was in place, and water from the hose and the pipe lines was poured on the fire continuously until about 4 P. M., when it was entirely extinguished.

In response to Mr. Birtley's call for assistance, Mr. C E. Tobey, assistant general superintendent of the D., L. & W. Coal Co., after ordering the Lackawanna company's Rescue Car to the mine, and telephoning Mr. Charles Enzian, superintendent of the United States Bureau of Mines Rescue Station at Wilkes-Barre, of the call, went at once to the assistance of Mr. Birtley. Shortly after he arrived at the mine, he was joined by General Superintendent W. L. Allen, of the Scranton Coal Co. Shortly after 3 o'clock, the D., L. & W. Co.'s Rescue Car, in charge of Harry G. Davis, division superintendent, with a first-aid corps, composed of Wm. White, Lewis Richards, David Bryant, Gwyllim Jones, John Williams, Richard Thomas, and John Driscoll, after a fast run of 25 miles from Kingston, arrived at the mine. This corps was joined at the mine by Ben Hughey, John Fulton, and Alfred Williams, three other Lackawanna men. Mr. Davis and his men brought four oxygen helmets with them, and immediately made preparations to enter the mine. By this time Mr. Enzian, with his corps, accompanied by a Lehigh Valley Coal Co. rescue corps, and two helmet men from the Delaware & Hudson corps, arrived and joined Mr. Davis' party. Among other mine officials who arrived about this time were Gomer Parry, inside foreman of Richmond No. 3 mine of the Scranton Coal Co., and Daniel Young, district superintendent of the same company. Shortly after this, State Mine Inspector Evans arrived, and he was soon followed by State Mine Inspector D. T. Williams. All of these men rendered all the service possible both physically and by consultations and advice.

Shortly after Mr. Davis and the helmet men reached the scene of the fire it was under control, and they were able to get through the slope workings in the China seam to the Tunnel Workings. Mr. Davis, who was unhelmeted, remained in the slope workings near upcast *H* and directed the men in the work. While the D., L. & W. men were entering the affected workings, Foreman Joseph E. Evans, of the United States Rescue Corps, a brother of Inspector Evans, also equipped with a helmet, hurried to join those ahead of him. He entered Perry's gangway through the regulator *N*, and was found sometime later by the returning D., L. & W. men, unconscious at the point *R*. It is supposed that wearing the heavy helmet, he overexerted himself in an effort to catch up with the others and assist in caring for any men who might be alive in the workings, and the oxygen and overexertion affected his heart. Although physicians and first-aid men worked over him for hours he could not be revived. He was a young man of ability, courage, and high ideals, and was the first of the United States Bureau of Mines men to lose his life in an effort to save others.

The origin of the fire will never be positively ascertained, but it is the opinion of the writer that it was caused by the coal-oil lamp in the lantern exploding and throwing burning oil on the dry and oil-bespattered timbers.

The 72 employes in the Tunnel Workings of the China seam were unquestionably killed by carbon monoxide, or "white damp," carried with the smoke from the fire through the workings by the ventilating current. A study of the map and the course of the air will show any one at all familiar with mine ventilation that the main split of the air-current, flowing into the workings through the tunnel and then up the main haulage road to the face where it was split and part conducted through the workings off the Miller and Bolton gangways and part through the workings off Gorman and Perry's gangways, must have first carried smoke on to the men and then some carbon monoxide, which was later increased in quantity when the fire was at its height and large quantities of water were being poured on it. Why all the men did not accept the first indications of smoke as a warning and endeavor to escape, as did the 16 who did get out, will never be explained. John R. Perry, after whom the gangway in which he was working is named, took the smoke for a warning, and with his two laborers went to the mouth of his gangway. Here he met Foreman Walter Knight and Fire Boss Isaac Dawe, who told him they were going in to get the men out, as they realized from the smoke that something

serious was wrong. Perry sent his laborers past the upcast H into the East Slope workings, whence they went in safety by way of the East Slope to the foot of the shaft. Evidently there was then an agreement between Knight, Dawe, and Perry to apportion off the districts each should take. That the quick-acting carbon monoxide was in the air in large quantities by this time is evidenced by the fact that Perry fell and died just inside of the door at O. Dawe's body was found at P, which is the point he reached after probably attempting to warn the men in the workings off Bolton's gangway. Knight's body was found at Q, in Miller's gangway.

Mr. James E. Roderick, Chief of the Department of Mines of Pennsylvania, when notified of the accident, started immediately from Harrisburg and reached the mine early Saturday morning, and Dr. J. A. Holmes, Director of the United States Bureau of Mines, who was in Philadelphia, having heard of the disaster, hurried to Scranton by first train and reached the mine shortly after Mr. Roderick. Both gentlemen joined the other mining men in their consultations and work of rescuing the bodies of the dead. Between 5 and 6 o'clock on Friday evening, Mr. Birtley, who was ill, and who had been working continuously all day, gave out and requested Mr. Allen to relieve him in directing further work for a while, so that he could sit down and rest. Mr. Allen insisted that he should go home and receive medical attention. He finally did go to the surface and to his office where a physician gave him some medicine and advised that he go home to bed. He remained in the office, however, till 9 P. M., when Mayor Von Bergen, of Scranton, with a friend, arrived in an automobile. Mr. Von Bergen, seeing the condition Mr. Birtley was in, practically forced him to go home in his automobile and to bed.

As the fire had been extinguished before Mr. Birtley gave out, the only work to be done was to recover the bodies, and Mr. Allen directed this work. As the ventilation was unaffected, after the fire was extinguished, the gas and smoke soon disappeared and searching parties started the work of looking for the bodies and bringing them out.

The crosses on the map show where the bodies were found, and the locations give evidence that they nearly all tried to escape, but too late. Some bodies were found in positions that indicated death before they realized any danger. In one case a man was seated with his lunch can open and a piece of bread in his hand, as if death had overtaken him while taking a lunch.

None of the bodies showed any evidence of death being caused by any other agent than carbon monoxide, though there must naturally have been enough smoke in the air to render breathing difficult and painful. Some of the men went to the faces of workings hoping, no doubt, to escape the smoke and gas by getting inside the cross-cuts and destroying the brattice, and thus keep the smoke from reaching them. Some took the most natural way of going out the main intake and they of course must have met death very quickly. Others tried the return but were too late, as the deadly gas overtook them. In the case of the 12 men overtaken in the return air-course of Bolton's gangway, 10 had passed through the regulator, and the gas caught them before the two who followed could pass through and close the slide. Had they all passed through and closed the regulator, it is more than probable that they would all have escaped.

Mr. Roderick has expressed an opinion that all the men lost their lives within an hour after the fire started, and the writer after making careful inquiry and examining the statements of survivors and those on the ground, and a careful study of the ventilating system agrees in that opinion.

As usual in the case of large mine accidents the first accounts published in newspapers were inaccurate owing to the haste in which information must be gathered for them. Besides, sensational and demagogical journals, not only had inaccurate statements, but they added to them sensational misstatements that gave the public a wrong impression of the normal condition of the mine and the characters of the mine owners and mine officials.

One false impression created was that the engine house in which the fire started was a flimsy wooden shanty and a fire trap of the worst kind. Instead of being such, it was a room in which an extensive fire would not occur, unless a deliberate attempt was made to start one, or an exploding coal-oil lamp would originate it. Naturally no one would under the circumstances deliberately set it on fire, therefore the exploding lamp theory is most tenable.

Another widely circulated false statement is that there was no second outlet from the Tunnel workings in the China seam. As a matter of fact there were three: one past the upcast shaft H into the East Slope workings, and thence out, and two up the upcast shafts H and M, to the Dunmore No. 2 seam workings and thence out.

Considerable adverse criticism has been made by news writers and others who do not understand mine ventilation and the conditions caused by the fire, that the fans were not stopped as this would keep the smoke and gas from being carried into the China seam Tunnel Workings. Such critics, and many of their readers do not know that if that had been done, the lives of between 400 and 500 employes in the Clark, Dunmore No. 2, and East Slope workings would have been placed in great immediate jeopardy, and besides the smoke and gas would have come back on the men fighting the fire, caused their death or their retreat, and a consequent general mine fire which would have probably cost the lives of several times the number in the Tunnel Workings.

To have reversed the current, as some uninformed writers think would have been advisable, would have brought the fire, gas, and smoke on to the men fighting the fire quicker even than a stoppage of the fans would.

The knowledge that the smoke and gas was being carried into the Tunnel Workings of the China seam was not lost sight of by Mr. Birtley. He knew of a speedy way to cut off the ventilation from that section of the mine, but neither he nor his assistants could get to the proper regulator to effect this, owing to the fire, which was between them and the regulator.

Another criticism on the mine management was that the breaker was operated and coal hoisted for some time after the fire was discovered. In the first place every man in authority on the ground knew that the only men in jeopardy were those in the Tunnel Workings of the China seam, and they thought of nothing else but extinguishing the fire and saving those men if possible. As soon as it was seen that the fire could not be speedily quenched, and that nothing further could be done to get at the men in actual danger, all mining and hoisting was stopped, and all men were ordered from the mine except those actually engaged in fighting the fire and in endeavoring to save the victims.

To sum the whole situation up—here was a well-laid out, well-equipped, and well-ventilated mine, with more than ordinary safety appliances, under the direct supervision of a competent superintendent, who not only knew practically all the employes personally, but who for 20 years had lived among them, shared their joys and sorrows, and who had the respect and good will of all.

A peculiar chain of circumstances, absolutely unforseen, caused the disaster:—A fire that evidently broke out a very few minutes after Moran left the engine room, when he was not there to extinguish it; an otherwise commendable current of air blowing through the engine house, which fanned the fire and carried it through the heading to the empty cars, which were standing in the position provided for them; and an air-current flowing past the burning cars that was so voluminous and well conducted, and which under normal circumstances was a beneficial and healthful agent, but which at this time was turned into an agent of death.

SCRANTON MINE CAVE ENQUIRY

Written for Mines and Minerals

Summary of the Report of the Mine-Cave Commission on Methods of Supporting the Surface

Scranton, Pa., is built over coal beds from which has been extracted, according to the estimates of William Griffith and Eli T. Connor, 198,000,000 cubic yards of coal and rock, or 24,000,000 cubic yards more of material than it is estimated will be excavated from the Panama Canal. The total quantity of coal and rock extracted is estimated at 221,000,000 tons, from which 177,000,000, tons, or 80 per cent. of coal, has been marketed.

It is estimated that the original coal area of 58,110 acres has been reduced by mining to 32,134 acres, which at the present rate of mining, 6,000,000 tons per year, will last 21 years, and if no coal is left for pillars, it will last 31 years.

Messrs. Eli T. Connor and William Griffith, who were appointed by the Mine-Cave Commission to examine the conditions existing under Scranton, handed to the Mayor of Scranton, April 1, a report showing what in their estimation should be done to prevent future mine caves.

The summary of their report follows:

After spending more than 40 days time studying maps and after testing various materials used and considered for the roof support in the mines, and tabulating and considering the information gained by these investigations the conclusions your engineers have reached are as follows:

While some other devices are locally useful, the only method that combines the necessary requisites of strength, ease of application, and reasonable cost, is filling the underground openings by what is known as the flushing method, using for this purpose culm, sand, crushed rock and other fine materials that can be washed into the mines with water.

This method was originated in the anthracite region of Pennsylvania, and has been extensively adopted in European mines, where at great expense, sand, loam, or crushed rock is flushed into the mines following the removal of the coal by the long-wall method of mining.

The tables and estimates of cost contained in the body of the report give in detail the results of our investigations and conclusions. Your engineers, therefore, offer as their only recommendations that the flushing method be adopted, under the plans and specifications, contained in the body of the report and in the atlas before referred to, from which the following general conclusions are drawn:

1. That, speaking broadly, the surface of the city can be supported by the methods recommended, and at a cost not in any sense prohibitory when considered with relation to value of the property and activities for which the support is absolutely essential.

2. That, while there are points in the city, as indicated in the detailed report, where, at the present time, in our judgment, there is distinct and immediate danger to life and property; yet, the total of such area immediately threatened constitutes but about 15 per cent. of the entire area of the city, mainly from surface beds.

3. On the west side the middle series of beds are thick, and close together, and the pillars not columnized, creating a dangerous situation where the workings have not been closed by previous caves. Particular areas thus threatened cannot be definitely specified, on account of the inaccessibility of much of the mined-over area. Detailed investigation should be made of the portions of the mines not already closed. Relatively we do not believe that a large part of the territory mentioned is threatened, on account of so much having been previously closed by caves. Special attention is called to the conditions under schools Nos. 13, 23, and 29. They should be promptly attended to.

The lower series of beds; namely, the three Dunmores, are so thin and so far below the surface that with usual system of mining we do not think they constitute a serious menace to the improvements on the surface except along the margin of solid blocks of unmined coal and near the outcrops. In the deep lying Dunmore beds we believe these solid blocks should be mined.

4. It would seem, therefore, to be not only the part of wisdom, but absolutely obligatory to at once commence to give support to the points menaced, and thereupon proceed upon a general policy of giving support to the entire area of the city; for it must be borne in mind that with the mine activities that are constantly going on, other and additional points of danger are not only liable to, but in all probability will occur with each passing year; it might almost be said, with each passing month.

5. Where the owner of the surface has undoubted rights to support thereof, by coal pillars, in our opinion he could permit the removal of such pillars, and the value thereof would, under average conditions, pay for such artificial support as we have recommended, assuming that the pillars were mined and the support constructed by the same operating company. This observation, however, is based upon the assumption that in such case the operating company is one of the large transportation companies, inasmuch as while there might not be a profit in the immediate transaction of mining the pillars and installing the support, there would of course be a profit to such companies carrying the coal to market.

6. Culm flushing should be used only in coal beds having light cover up to 200 to 500 feet, according to amount of settlement expected, but sand being 4 or 5 times as strong as culm, is best and more suitable for all beds under Scranton and should be preferred.

7. We believe that the conclusion adduced from the tests made, and the calculations and tabulations based thereon, are reasonably reliable; yet, we desire to record the opinion that there are conditions existing in the mines where they might not apply. For instance, in the localities where several seams of coal are separated by a thin stratum of shale and slate, or even sandstone, and the pillars in the two or more seams are not over one another, and it is proposed reclaiming all or any part of the pillars.

Even though an application of the above tables might appear to fit the conditions, we believe that the only permissible procedure would be to first fill with flushed material all of the openings in the lowest vein of the series, and thence upwards, until all are filled, care being taken to have the flushed areas over one another. After all of the openings in all of the seams have been filled, the pillars in the uppermost seam may then be attacked and as each pillar is removed, the space to be at once filled. No pillar reclamation to be permitted in any of the other beds until all of the pillars in the upper bed have been removed and the over burden has come to rest on the flushed material; after which the pillars in the next lower seam may be attacked and handled in like manner.

8. Harmonious plans and procedure between the coal companies, the city, the school authorities, and the public are essential to the successful carrying out of any relief measures that are herein or may be hereafter suggested. It is a fact which should be evident to all that the prosperity of the city and community is to a large extent dependent upon the coal companies, and that drastic laws or regulations that may curtail the mining of coal will necessarily react on the prosperity of the community, and any ameliorating plans or compromises as between the city and the mining companies which it may be possible to effect, tend to prolong the life of the mining industry in Scranton and vicinity, and should be promoted.

It should therefore be the aim of all interested in mine cave protective measures and the companies operating the mines, to adopt plans that will best conserve the welfare of all concerned.

For the business-like carrying out of the plans suggested it is recommended that a protective commission be established,

consisting of not less than three or more than five men, representing the city authorities, school board, and the coal companies; this commission to have full and complete authority for the execution of the plans, approved by the proper legal action.

THE HAZLETON PLUNGER JIG

Written for Mines and Minerals

It has been the custom almost from the time jigs were first installed in the anthracite fields to use those of the pan type, known as the Clark, or Lehigh, jig. Jigs of this description have the bottom of the pan made of wire netting or perforated plates upon which the material to be separated rests and is raised up and down in water.

The principle of this jig, like all other wet jigs, depends for making its separation upon the fact that the coal is lighter than slate and therefore travels farther on the down stroke and falls less rapidly on the up stroke. Thus there is a small movement during both strokes when the coal tends to separate from the slate. But this continued repetition eventually causes the coal to rise to form a top layer, while the slate forms a bottom layer. Ordinarily all jigs in the anthracite field look alike, and the writer's attention was directed to the Hazleton jig by the rotary coal discharge. On further investigation it was discovered that the jig was of the plunger type, that is, instead of raising the pan up and down in water the sieve was stationary and the material resting on it was raised up and down by the movement of a plunger working in water.

While the plunger jig is not a recent invention, nevertheless its adaptation to coal preparation after so many years of the pan jig is something of an innovation.

According to the information given, the plunger jig was first introduced in the Audenreid breaker in 1901, where it was placed alongside one of the pan jigs and made to perform exactly the same kind of work under the same conditions. The result of this comparison for 1 year showed that the cost of repairs for the plunger jig was $10.65, while that of the pan jig was $53.01. This company then adopted more of these jigs, and in the Lehigh region they are becoming popular. Jigs of this description have a large capacity and are capable of treating from 125 to 250 tons, according to the size of the material, per day of 10 hours. In Fig. 1 (*a*) is shown an elevation of the plunger jig and in Fig. 1 (*b*) is shown the plan.

In the figure, *a* is a plunger worked up and down in water compartments by the eccentric *b* on shaft *c*. On the down stroke it forces water from compartment *d* up through a perforated plate *e*, into compartment *f*, and in so doing raises the material resting on the perforated plate or sieve. As the coal is lighter than the slate it travels faster upwards, but when the water recedes as the plunger makes its up stroke the slate falls the faster. Thus separation to a limited extent is produced coming and going.

The material to be treated enters compartment *f* near the sieve through the hopper *g*, then spreads over the sieve as fast as either coal or slate is removed. As the coal accumulates it floats over the gate lip *h*, is scooped up by the rotary-coal discharge *i* and deposited in the chute leading to the pocket. When the slate has accumulated and has become so thick on the sieve as to show signs of running over to the discharge, the doors *j* are opened by the levers *k* to permit the slate to flow into pockets *l*. So fast as the slate accumulates in the pocket it is raised by the revolving buckets *m* and discharged to a chute leading to the slate pocket not shown.

In the Lehigh jig the coal is discharged over a perforated plate which wears out quickly owing to the acid water and abrasion it undergoes. The rotary-coal discharge combines the dewatering part played by the plate of the Lehigh jig; it is also an elevator, thus little water is lost, and in addition, it is said to wear out two sets of plates.

The mud which accumulates in the hutch *n* is discharged by means of the double-gate valve *o* worked by the levers *p*, *q*. However, it may be necessary with dirty coal to entirely draw off the water and sludge in the hutch once or twice daily, and this will necessitate refilling the jig with clean water. A Hazleton jig occupies a floor space of 12 ft. $7\frac{3}{8}$ in. × 8 ft. $10\frac{1}{2}$ in. over all, including the driving gear. To run such jigs requires $4\frac{1}{2}$ horsepower. When working on run-of-mine coal it is safe to assume that from 25 to 50 per cent. refuse goes to the jigs.

Run of mine in this instance means coal that has been crushed to size but not cleaned previous to going to the jigs. The number of revolutions and lengths of stroke given the

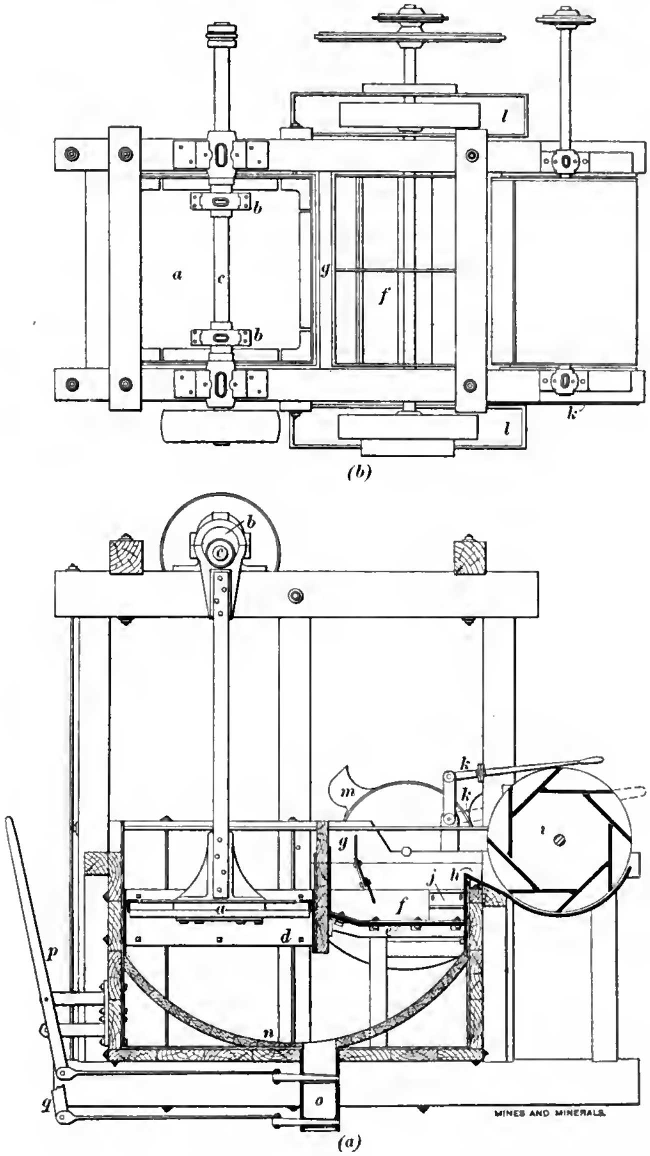


FIG. 1. PLAN AND ELEVATION OF HAZLETON JIG

plunger are determined by the size of material treated. Pea and buckwheat are jigged on strokes $2\frac{3}{4}$ inches and $2\frac{1}{4}$ inches, respectively, at 90 revolutions per minute, while chestnut is given a 3-inch stroke, and egg a 4-inch stroke, at correspondingly lower speeds.

MINE RESCUE CAR SCHEDULE

The following schedule is announced for mine-rescue car No. 2, operating from Trinidad: April 30 to May 10, Starkville; May 11 to 20, Dawson, N. Mex.; May 21 to 31, Raton, N. Mex.

JEFFREY-GRIFFITH CROSS-OVER DUMP

The Jeffrey-Griffith cross-over dump possesses features which differ materially from any dump heretofore described in Mines and Minerals. The claims of the manufacturers are that this dump can be made to work so fast that almost a continuous flow of coal takes place, and they guarantee that it has a greater capacity on this account than other cross-over dumps, besides it has 50 per cent. fewer parts.

In Fig. 1 is shown the dump with the car just entering. The lever by means of which the mechanism of this dump is regulated is shown in Fig. 3. The dump consists of two continuous rails to which are attached substantial horns securely fastened to a heavy shaft. This shaft has a bell-crank and quadrant arrangement with spring securely fastened to one end, while at the other end is located a brake wheel and lever for releasing the car after it has been unloaded. When in operation by the activity of the man in charge and the time it takes for a car to kick back out of the way. It is also claimed that there is less breakage of coal and less wear and tear on the cars. Large dumping capacity such as this is due to the quick and continuous action of the dump, shorter movement of the cars, and the increased safety in bringing loaded cars close to the dumping position.

This is the only dump where the car maintains continuously its natural position throughout the operation. Ordinarily there is a break in the track or the dump is on a pivot. In this case, however, the track is continuous and it is the track horns which move, the car always remaining on the track. When this is understood it can be seen that there is no break or hinge in the tracks which would cause derailment of the cars and that the dump occupies less space than other cross-over dumps, consequently, the cost of the tipple building for a long run to the kick-back is avoided.

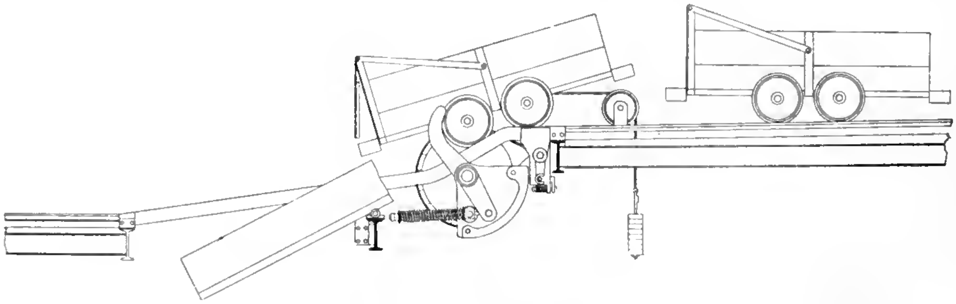


Fig. 1. Car Entering Dump

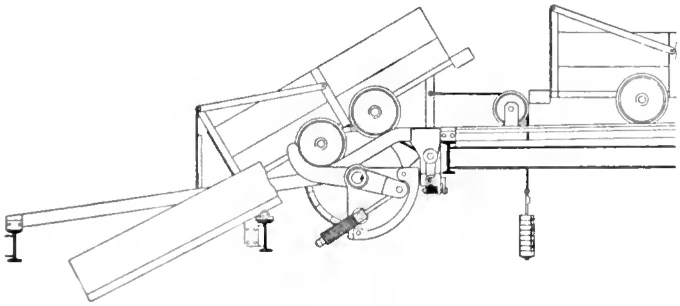


Fig. 2. Dumping

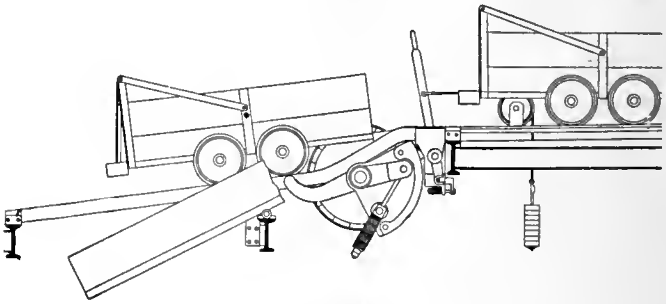


Fig. 3. Releasing

the cars pass on to the dump, as shown in Fig. 1. The impact of the car when it strikes the horns is taken up by means of a spring which absorbs the shock and removes it from the car boxes. One of the most objectionable features to all end dumps has been this shock, which eventually worked the car boxes loose or otherwise injured the car body.

Fig. 2 shows the car in position when it is dumping and is practically a continuation of the movement which occurs when the car runs on to the tipple and strikes the horns. Consequently the front part of the car does not swing and come with a jar against the chute. As soon as the coal is out, the car is released by means of a lever which depresses the horns and permits the car to move over the slightly inclined track, as shown in Fig. 3. There is enough incline to this track to give the car sufficient impetus to run to the kick-back and thence back out of the way of the next car. After the car has passed the horns they automatically resume the position shown in Fig. 1. This is accomplished by means of the spring, which is of sufficient strength to bring the horns into position, together with the rotating action of the quadrant when acted on by gravity. Continuous rails are greatly in favor of this dump, as there is not much likelihood of their getting out of order, besides stoppage of the operation due to derailment of cars is an impossibility.

The manufacturers claim that 10 cars can be discharged over this dump per minute, so that its capacity is only limited

The dump may be seen in operation at the Prudence Coal and Coke Co., Prudence, Tenn. Further details may be had from the manufacturers, the Jeffrey Mfg. Co., Columbus, Ohio.

PERSONALS

W. R. Craig has been appointed acting superintendent of the Shawmut Mining Co., with headquarters at St. Mary's, Pa. Mr. Craig succeeds the late Mr. Ramsey.

W. J. Jones, of Akron, N. Y., has been appointed State Mine Inspector. Mr. Jones was once fire boss at the Red Ash shaft, Exeter colliery, of the L. V. C. Co., and later foreman at the Stevens colliery, West Pittston, Pa.

F. Lynwood Garrison, E. M., Philadelphia, has been examining asbestos deposits in Canada.

James M. Russell, formerly mining engineer with the Development Department of the E. I. Du Pont de Nemours Company has just returned to his office at Deadwood, S. D., after a three months' trip to Arizona, where he examined the property of the Southern Belle Mine for Col. W. F. Cody.

Lewis A. Parsons and John I. Kane, mining engineers, have formed a partnership under the firm name of Parsons and Kane and opened an office at Coles Building, El Paso, Texas.

COAL-DUST EXPLOSIONS

Written for Mines and Minerals, by John Verner

In 1894, after several years investigation of the subject of coal-dust explosions, the British commission presented the following conclusion: "Coal dust alone without the presence of any gas at all, may cause a dangerous explosion if ignited by a blown-out shot or violent inflammation. To produce such a result, however, the conditions must be exceptional and are only likely to be produced on rare occasions." The truth and the significance of this statement is so evident that it seems remarkable that so little has been accomplished in the intervening years to determine the nature of these exceptional conditions. Insufficient investigation in the direction suggested by the British commission is largely responsible for the fact that coal dust is even now considered the most potential and all-important factor in a so-called dust explosion. For many years the investigators of the explosion problem have looked to the dust cloud and followed it as the only guide to lead them to the right solution, but apparently they are still wandering in the wilderness of uncertainty.

Condition and Quantity of Air of Greater Influence on Explosions Than the Dust Content

"As a cause of explosions, I have no faith in the theory of occluded gas; in the distant discharge of explosive mixtures by pressure; in the combustion of coal by friction; or the unknown atmospheres generated by electric current.

"The cause of mine explosions, I believe, are well within the present knowledge of chemistry and physics, but it is the lamentable ignorance in the interpretation of the multitudinous and therefore complicated conditions which exist, each of which, simple of comprehension in itself, but enormously complicated in the aggregate, that has so far prevented a rational explanation of all."

The logic of the above remarks by Mr. Haas is eminently sound. There is nothing mysterious about dust explosions and there is, in my judgment, nothing unexplainable about their occurrence or behavior if cause and effect are properly correlated. In this article I shall advance no new theories. I shall only attempt to present the facts as I have been able to ascertain them through investigations and through comparison of conditions as they existed in mines in which explosions occurred, with the view of establishing their relationship to each other and their mutual effects.

To get first-hand information as to what conditions may develop in the near vicinity of a shot fired in the coal, I made the following experiments in an Iowa mine: The place selected for the experiment was an entry, slightly dipping, 6 ft. × 8 ft. in the clear, its face at a distance of from 60 to 75 feet beyond the last cross-cut. The coal is blasted off the solid and the shots were prepared by the miners working in that entry in the regular way. A few feet inside of the last cross-cut a number of inflated paper bags were suspended by short strings from a collar, a wire was connected horizontally halfway between roof and bottom with the legs supporting the collar, and other bags were suspended from the wire, these bags being about 6 inches from the floor. The observers were stationed in the cross-cut and could readily see any movement of the bags. The experiments were made in still air. The shot fired at the first observation was rather violent, the bags, both upper and lower, were blown outward with considerable force but returned instantly to their original position. Immediate investigation showed no powder smoke along the floor of the entry for a considerable distance inside the cross-cut, and a very rapid air movement along the floor toward the face. Considerable heat was found in the upper part of the entry, especially near the face. The shots at the second and third observation caused a far less violent initial outward force. The paper bags in each case were blown outward, but instantly swung back. Very little air movement along the floor toward the face was noticeable. The degree of heat present was found to be decidedly less than in the first observation.

The shot fired at the first observation plainly showed dangerous tendencies, and it may be reasonably assumed that, with a further increase in heat and flame intensity at the face, as might be produced by a blown-out shot, the force of the outward rush, the consequent instantaneous reaction, and the rapidity of air flow along the floor toward the face, would be increased proportionately. The reaction and the air inflow in the manner stated were evidently caused by the effect of the immediate cooling of part of the heated gases near the face, and the shrinkage in the size of the flame, and by nature's well-known method of action in bringing about the rapid equalization in temperature of contiguous gas bodies of different temperatures. The results developed in the first experiment described above plainly indicate the manner of an explosion's initial start and prove erroneous the generally prevailing theory regarding this process, that the concussion produced by a blown-out shot, or other violent inflammation, stirs up the dust, that the flame projects itself into the suspended dust ahead, keeps on striking into it or impinging on it, and that by this method an explosion is extended as long and as far as there is any dust for the flame to reach. In other words, according to this view, the flame must derive and sustain its vigor by attacking the rear portion of a rapidly receding fuel supply. It seems to me that it must require the absolute disregard of immutable laws and facts to believe that tremendous explosive effects can be obtained under such conditions.

The first effect of the shot in the face of the entry resulted in driving away from the face the air contained in the space near it and with it any dust it might hold. This momentary deficiency in the air supply near the face undoubtedly was a factor in preventing immediate dilution and ignition of some of the combustible gases produced, in diminishing the initial heat intensity, and in reducing the size of the flame, the latter conditions causing an immediate reaction together with a rapid inflow of air along the floor. From this it can be reasonably assumed that, should the fresh air supply along the floor reach the face before the flame is extinguished or the heat becomes too low, the flame will be immediately enlarged, the gases present will be brought to the explosive point and ignited, with the result that the extent of this second inflammation may be far greater than the first, especially if the ingoing air is dust laden. All this will transpire so rapidly that it will be practically impossible to distinguish the second inflammation from the first, except in such exceedingly rare instances as noted in the starting of the Minneapolis mills explosion. It is further evident that the same conditions that influenced an explosion's start, must necessarily obtain in its propagation. The heat and flame area varies in intensity and size during every moment of the explosion's progress; there is a continuous shrinkage and expansion of flame, producing and reproducing with immeasurable rapidity the same effects as at its starting point. It is the presence and influence of this dynamic force that produces high explosive effects, and in its absence the mere contact of flame and coal dust has proved comparatively harmless. The results of experiments and laboratory tests clearly and convincingly show that explosive effects and the propagation of an explosion can only be produced by the forcible injection of air and dust into the flame. Peckham and Peck could only produce explosive effects by blowing the dust into the flame by means of a bellows. In the experiments by Engler, by Holtzwart and Meyer, by Bedson and Widdas, and by Frazer, the dust was ignited by blowing it into the flame by compressed air. Doctor Frazer, in speaking of this manner of producing explosive effects, makes the following remarks of special significance: "Unless there is an exceptionally large amount of dust in the air experience shows that ignition does not take place from a naked flame. This fact is illustrated by the work of Galloway

and that of Mallard and Le Chatelier, who used for the source of ignition different kinds of naked flames, and accords with the conclusions which they reached as a result of their work. It will be recalled too, that Holtzwart and Meyer drew attention to the fact that no ignition was obtained if they introduced lignite dust into their apparatus, and, after establishing the spark between the terminals within, disseminated the dust in the air by shaking the tube. *But when the dust was puffed between the terminals by compressed air, ignition occurred.*"

Under the direction of the Chesterfield and Derbyshire Institute of Engineers "experiments were made in a gallery 82 feet long, 16 inches wide, and 18 inches deep, connected with a chimney to produce an air-current. Coal dust was introduced at the open end of the chamber and carried in by the current. A horse pistol was used to simulate a blown-out shot. The charge was ½ ounce of gunpowder. It was fired into the open end. Out of 134 tests with dust alone, ignition was obtained in 36 cases. Even when 6 per cent. of gas was tried no violent explosion resulted. In the opinion of the observers it was more of an inflammation than an explosion." It may be said that the failure to produce ignition was due to the limited size of the flame produced by the small powder charge, but this cannot be the reason because in the experiments of Peckham and Peck and others a much smaller flame was used as the inciting cause, yet ignition resulted in every instance and with all kinds of combustible dust. The unfavorable results of the Derbyshire experiments can only be accounted for by the fact that the pistol was discharged at the open end of the gallery and that consequently the primary air compression in the gallery from the effects of the shot must have been very small, because the major portion of the pressure produced by the explosion of the powder charge would naturally seek immediate relief through the open end, and therefore the reaction in the gallery was too feeble to carry a propagating supply of air and dust to the flame before it became extinguished.

I believe that the lack of investigation in the past of the value of the air factor, both in a physical and chemical sense, in connection with explosions has been a mistake, and judging from Mr. Rice's remarks in an article on the explosibility of coal dust, it appears that the mistake is to be continued and that investigators have about concluded that everything relating to the air factor has been discovered, making further efforts in this direction useless. Mr. Rice says: "The belief has prevailed very generally among mining men that because a dust explosion is usually manifested in the intake entries the explosion 'feeds on the fresh air' or advances against the air-current. When it is considered that the oxygen content of the return air of the average mine in this country rarely shows a decrease of more than ½ of 1 per cent. of oxygen from the normal, and that the combustion of the coal dust would not be seriously affected unless the deficiency exceeded 2 or 3 per cent. or possibly more, it is clear that there can be little real basis for this impression. In this same connection, the idea has also been expressed that the fan was blowing in fresh air for the explosion to feed upon, and that this accounted for the greater destruction at the intake frequently manifested. When it is considered that the speed of a dust explosion, measured over only a short distance from the origin, as indicated by the Altofts experiments, is 1,400 feet per second, a rate presumably less than when the explosion is under greater headway, and, on the other hand, that the speed of the ventilating current would be at most one-sixtieth part of that rate, it is obvious that the fresh air introduced during an explosion cannot play any part in its propagation, except by the momentary mechanical pressure required to overcome the inertia of the air-current. *The fact that a dust explosion does usually seek the intake entries is due to another cause; namely, that the fresh air has dried the coal dust along the roads, whereas in the return airway of mines of any size the air-current, being saturated with water vapor, has no such drying effect.* The presence of the dust is the all-important thing."

Are the conclusions presented in the above statement warranted by the facts? The mining men may not have stated the case in scientifically correct language when they said that the explosion "feeds on the fresh air," but whether the expression is entirely apt or not is immaterial, the fact remains that it is a fair exposition of the truth. These men based their conclusions on facts as they observed them, they knew that a gob fire with its air supply shut off by brick walls and coal barriers will prolong its existence by crawling toward the tiniest stream of air that finds its way through invisible fissures and will attack the smallest crack in its prison walls in its avidity for more air. They also observed that in case of fierce combustion (and Mr. Rice speaks of "combustion" of coal dust) in a place of rather limited dimensions, like an entry, the fire will advance against the air-current that its effects created or increased. The Delagua explosion is an example of this fact. The truth is that a dust explosion, whether it travels on the intake or the return, or anywhere else in the mine, will always advance against an air-current toward it that must be of sufficient volume and force to sustain intensely rapid and fierce combustion and carry to it an additional fuel supply. An explosion cannot have the weird and uncanny faculty of knowing the existence of the dusty spots and the damp places hundreds and thousands of feet ahead of it, and of chosing its course accordingly. There must be, and there is, a guiding agent.

Mr. Rice's estimate that the difference in oxygen content of the return and intake air rarely amounts to more than ½ of 1 per cent., does not agree with results as I found them. There are many mines in this country where in the winter, the explosion season of the year, the oxygen content of the air on the intake may show an excess over the oxygen content of the air on the return amounting to 5 per cent. and more. I have found temperature in mines below the freezing point at a distance of more than 1,000 feet from the intake opening, while along the return the temperature was in the neighborhood of 60 degrees above zero. No doubt, there are mines, of the Lick Branch mine type, where, under favorable conditions the low temperature zone may be found still more extensive, but with a fairly high temperature prevailing on the return. Supposing that just before the second Lick Branch mine explosion on January 12, 1909, the average temperature of the air in the intake entries near the new mine opening was 32 degrees and that the temperature of the air on the return in the old mine was 60 degrees, then the excess of oxygen content in the intake over that in the return amounted to about 5 per cent. It is true that if 100 pounds of air enter a mine at the intake in a given time, about 100 pounds of air must leave it at the return in the same time and that the effect of air expansion is compensated by an increased velocity, but this is clearly not a question as to how much oxygen passes a given point in a mine in a given time under the ordinary method of ventilation, but a question as to how much oxygen is contained in the different parts of a mine at the moment an explosion occurs and the regular manner of air flow is interfered with.

To illustrate the matter I refer to conditions as they were reported to have existed in the Marianna mine at the time of its explosion on November 28, 1908. The mine was divided in six splits and 190,000 cubic feet of air per minute passed through it just prior to the explosion. The main entries, in groups of six, were driven from the main shaft in opposite directions. The mine was still in a stage of first development and no rooms had been turned. Taken on an average, the faces of all the entries were within a radius of somewhat less than 1,300 feet from either shaft. On account of the large number of parallel entries the linear extent of the excavations amounted to about 25,000 feet. The size of the entries and airways was 9 ft. × 7 ft. The contents of the underground openings were therefore about 1,575,000 cubic feet. Considering the depth from the surface it may be assumed that the normal temperature of the mine would be around 60 degrees, but the latter

being of limited extent and other conditions favorable, it would be readily and materially affected by changes in the outside temperature. Supposing then that on the day of the explosion conditions were such as to cause a reduction of mine air temperature from 60 to 45 degrees, a difference of 15 degrees. This drop in mine air temperature would mean an increase of oxygen content in the mine of 3 per cent. above the normal and it would mean an oxygen increase, measured by weight, of 850 pounds. The explosion traversed the whole mine in a few seconds, and in that short space of time the tremendous energy stored in the additional 850 pounds of oxygen was made available for fiercer combustion and converted into an immensely powerful destructive force. Much has been said about the influence of the presence of a small amount of marsh gas (less than 1 per cent.) in materially contributing to an explosion's development, its extent and force. If the addition of this small amount of gas to the mine air is considered a potential factor, and there seems to be no good reason for doubting the correctness of the claim, the effects on a dust explosion of an increase of the oxygen contents in a mine of several per cent. above the normal must be also of consequence.

By comparing the known conditions of a large number of mines in which explosions occurred, I found that the following features apparently exerted a well defined influence on the magnitude of dust explosions: The thickness of the coal seam, and the number of openings from the mine to the surface with the entries leading from such openings to the mine workings.

In examining the long record of explosions in which coal dust was found to have been a contributory cause, the fact stands out clearly that the most violent and widespread explosions of this kind occurred in mines with coal of fair thickness—6 feet and over—and I have been unable to find any record of a dust explosion in this country in a mine with a coal seam of 3 feet or less. Explosions occurred in mines where the height of the seam was between 3½ and 4 feet, but their violence and extent were comparatively limited and their force was generally confined to the brushed entries. *In a general way dust explosions showed a decided preference for high, narrow entries, and wide rooms were far less affected.* It is but natural that this should be so, for the concentration of heat with a consequent increase in intensity is an essential feature in the propagation of a dust explosion, and the high, narrow entries are of much greater assistance in this respect than the rooms of considerably greater width, that permits the heat and flame stratum to thin out and spread in a lateral direction. In seeking the reason why the most destructive and extensive dust explosions so persistently occurred in mines with coal seams of fair thickness—6 feet or more—and why explosions were less violent and covered less ground in mines with thinner seams, it seems obvious that a difference in dust conditions could not account for it, for the measure of the height of a coal seam does not determine either the dust's quantity or its dryness, fineness, or volatile content. As blown-out shots produce about the same number of heat units per pound of explosive, and as the quantity of explosive used in a shot in the thinner seams is generally as large and often larger than the quantity used in a shot in the thicker seams, it could not be reasonably assumed that the explosive had any bearing in the matter. But when the air conditions were examined the reason for the difference became plain. In a coal seam 6 feet thick, the air content per linear foot of extent of excavation space of equal width is 50 per cent. greater than in a seam 4 feet in thickness, and what is perhaps still more important is the additional fact that the facilities for getting the greater air supply in the thicker seam to the burning fuel are proportionately better than in the thinner seam, and as a dust explosion is primarily a matter of dust combustion, the greater violence and extent of such explosions in thick seams must be due to the greater quantity and availability of the air supply. That appears to be the logical reason and is in full accord with the conclusion by Mr. Haas: *The quantity of air, rather than the quantity of dust or coal is really the measure of the magnitude of an explosion.*

I found further, that the most destructive and extensive dust explosions occurred in mines having more than the usual number of surface openings, with numerous and large-sized entries connecting these openings with the live workings. As illustrations I cite a few well-known cases. The Scofield, Utah, explosion: Two mines connected; four surface openings. The Monongah, W. Va., explosion: Two mines connected; six surface openings. The two Lick Branch, W. Va., explosions: Two mines connected; at least four surface openings. The Darr, Pa., explosion: Three surface openings. The Courrieres, France, explosion: Four mines connected; four hoisting shafts, a centrally located air-shaft, and probably other connections between these mines and the surface. Of course, the connecting of mines increases the territory accessible to an explosion, but the size of a mine alone does not measure the latter's magnitude; for explosions have occurred in dry and dusty mines of considerable extent, yet the territory affected was comparatively limited and their violence correspondingly small. It appears, therefore, that the existence of numerous mine openings and their appurtenances must have an aggravating influence, and, if this is so, again such influence can only be manifested through the action of air and air movements made possible by the presence of these openings.

Before attempting to explain this matter, I will say that coal dust cannot be considered an explosive, in the sense the term is generally used, but, like other finely divided combustible dusts, is readily inflammable, and in my view a so-called dust explosion is really intensely rapid combustion of available fuel, the combustible material affected in a mine ranging all the way from the smallest dust particle to the solid coal. Combustion produced the coking of the solid coal in mines affected by dust explosions (in the Monongah mine the coal was found coked to the depth of 3 inches in places) and consequently in investigating the actions of these explosions, the effects of the influence of the fundamental principles governing combustion should have full consideration. It should also be understood that explosions entirely due to firedamp, and dust explosions, have little in common; they are different in origin, manner of propagation, and effects produced. One point of difference is that in the firedamp explosion there is generally a quite noticeable recoil or return rush due to the rapid cooling of the remaining gaseous content in the territory affected, while in an extensive dust explosion this return rush is almost entirely absent. Many, however, believe in the existence of a return rush of considerable force in the latter case, and the following remarks by Mr. Rice may be taken as expressive of the general view on this subject: "Following such an explosion there is a continued outrush of burned gases, which often carry surplus dust and deposit it on the lee side of projections, sometimes covering the coke just deposited. Succeeding this outrush there is a return rush to fill the vacuum caused by the cooling of the gases. The return wave, if strong, may pick up dust and redeposit on the lee side of projections, or the side which faced the explosion."

Why should the cooling of the gases and the formation of the vacuum be delayed until after the explosion ceases, and what proof is there, for instance, of such delay in the Darr and Monongah cases, where the explosion traversed miles of mine workings and reached the surface? I have no doubt that the tendency toward a vacuum in the rear of these explosions, due to the cooling of the gases and other causes, existed during every moment of their progress, but it certainly did not produce in these instances the effects mentioned by Mr. Rice, either during or after the explosions. If an incipient vacuum back of a dust explosion is not immediately and fully replenished from the rear, then its demands must be satisfied from the opposite direction and in that case an explosion's speed must be affected and reduced, and if this condition continues

or becomes more pronounced, the explosion's forward movement will become slower and slower and may be stopped entirely, notwithstanding the presence of large quantities of dust, and I know from personal experience that explosions did come to a stop in the midst of an abundant fuel supply. It appears, therefore, that the existence of favorable conditions which will facilitate air movements in a mine during an explosion in such manner that they will furnish not only an abundant air supply for propagation, but also for a ready filler to prevent the formation of a vacuum in the explosion's rear, must be of very material influence in assisting the explosion's rapid advance and spread along every available channel in a mine. In this connection the importance and possible influence of the number, size, and location of mine openings becomes apparent. Numerous mine openings, with entries leading from them to the interior workings, will not only aid in developing above favorable conditions in the highest degree, but they will furnish just that many exits for the explosion and permit it to make its attack in quick succession, first in one direction and then in another. In the Monongah explosion there was a proved difference of 5 seconds between the appearance of flame at No. 8 and No. 6 openings, and, as according to the Altofts measurements an explosion travels at the rate of 1,400 feet or more per second, 5 seconds in an explosion represent a lot of time.

Through the courtesy of Mr. Laing, Chief of Department of Mines of the state of West Virginia, I received the following data regarding air movements in mines during an explosion and the force such air movements may develop. When the second Lick Branch explosion occurred on January 12, 1909, Mine Foreman Bowers and another man were within 400 feet of the Tug River opening. This opening is about 5,000 feet southwest of the old mine entrance and about the same distance, in an air line, west of the new mine opening, the old mine and the new mine openings being about 2,500 feet apart. Bowers' first intimation of something being wrong was a blast of cold air coming from the Tug River opening that knocked him down with such force that two of his ribs were broken. Then came a hot blast from the interior of the mine. Bowers, after the explosion, crawled about 200 feet before he was overcome. He was brought out alive and recovered, while his companion, who had been thrown into a ditch and was unable to make any progress toward the outside, was killed. Only on rare occasions can direct evidence be obtained as to what happened in a mine during an explosion, and Bowers' experience, therefore, is of special value, not only with regard to the proof of the air movements, but also because his testimony furnishes a reasonable explanation of the evidence, frequently noted in violent explosions, of severe struggles of forces that apparently moved in opposite directions. Somewhere in that mine there was at least one collision between the cold air blast rushing into the mine with a force sufficient to knock men down and injure them severely, and the explosion, and it can be reasonably concluded that such coming together of opposing forces must result in a fierce and destructive contest.

The occurrence of two very destructive explosions in the same mine, absolutely free from firedamp, well ventilated, and otherwise in good condition, within the short space of 2 weeks is, as far as I know, without a parallel in the mining records of this country and presents in an emphatic manner the valuable lesson that the mere presence of the dust is not the all important factor in an explosion. The Lick Branch explosions occurred in midwinter and both started within a few thousand feet of the intake openings through which large volumes of cold air were drawn into the mine. The latter's oxygen content was undoubtedly considerably above the normal, especially as all the openings were large-sized drifts, driven in a practically level seam, thus making the mine readily responsive to the effects of low outside temperatures. As the coal seam was 8½ feet thick, with good roof requiring no timbering, the mine passages were unusually high and unobstructed. The importance of this feature has already been shown. Looking over the situation as a whole, it appears that these favorable mine and air conditions had vastly more to do with starting and extending the explosions than the dust in the mine, and that the measure of the latter's influence was determined by the former.

A comparison of the Hall tests and the Altofts experiments furnishes further proof that the conditions under which the experiments were made, rather than the dust present, were the more potential factors in determining results. A number of Hall's tests were made in the Big Lady shaft, 630 feet deep. The cannon, charged with 1½ pounds of black powder, was located 540 feet from the surface. If it is true, as claimed, that the amount of dust in suspension measures the magnitude of an explosion, Hall's tests, made in a vertical shaft, under the most ideal conditions of profuse dust suspension should have produced remarkable manifestations of explosive results, yet, he says, there were many failures to cause explosions or to ignite the dust. In one test, admitted to have been the most spectacular of any, the flame filled the pit mouth and ascended 60 feet into the air, but the fact that the flame continued to issue from the shaft for 5 to 6 seconds indicates that the exhibition was really a huge conflagration and not an explosion. On the other hand, in the horizontally disposed Altofts gallery, with the same amount and kind of explosive used, and under far less favorable conditions of dust suspension, violent explosions were produced with unvarying regularity and without a single failure. What caused this difference in results, if it was not the difference and influence of surrounding conditions? If there were exceptional conditions at Altofts, and if the British commission was right that exceptional conditions must exist to cause an explosion in a non-gaseous mine, it is clear that a full explanation and understanding of their character and value must be secured to assure the final solution and settlement of the explosion problem.

There is no lack of measures that have been suggested for the prevention of dust explosions, for limiting their extent, and for providing means of escape for the mine workers after an explosion, but the wide difference of opinion regarding their effectiveness makes it very difficult to decide as to which, if any, of the proposed remedies promises to be of sufficient value to warrant general adoption, and it may help some toward the adjustment of these differences of opinion to examine the more prominently mentioned remedies in the light of the facts obtainable regarding them.

It has been proposed that numerous openings, connecting the mine workings with the surface, be provided in order to limit the extent of explosions and to give the mine workers additional means of escape from a mine should an explosion occur in it. I am of the opinion that their presence in connection with dust explosions will do infinitely more harm than good. There were numerous openings in the Monongah mine and there were many entries leading to them from the workings, yet their existence did not limit the explosion's extent and did not save a single life. There were also numerous shafts and there were many entries leading to them from the workings, yet their existence did not limit the explosion's extent and did not save a single life. There were also numerous shafts at Courrieres, but the explosion in these mines on March 11, 1906, destroyed more than 1,200 lives, and the number of the men who escaped after the explosion was pitifully small. Additional mine openings should be made wherever necessary, for there are other matters to be considered in mining besides explosions, but if they are made, it should be understood that they are not only almost useless as means of escape after an explosion, but that their presence will assist very materially in increasing its violence and extent and thus contribute to the greater destruction of life.

The use of water by sprinkling, spraying, or otherwise is also proposed as a remedy. It looks effective, but neverthe-

less there is a bewildering difference of opinion regarding its value. Some are confident that the application of water is a sure remedy, while others are just as positive in their belief that the presence of water is a detriment and tends to make an explosion more severe, and there are still others with views varying all the way between the two extremes. Proof that the presence of moisture will prevent an explosion under all circumstances is not obtainable, but there is fair proof of cases where the extent of explosions was apparently limited by the presence of considerable moisture or long stretches of water in the roadways. There is also proof that while the presence of water apparently stopped the progress of flame, it did not stop the heated and poisonous gases from passing over and spreading with deadly effect through the mine workings beyond. And again there is still other proof that the presence of considerable moisture neither prevented explosions nor had any appreciable influence in limiting their extent and violence, as the following cases will show. Mr. Newsam reports this instance: A drift opening had been extended 103 yards under ground, wet from the mouth to the face, and water dropping constantly from the roof all the way. There was no gas and no dry dust. The explosion occurred on January 31, 1907. A water car, two-thirds full of water and standing 50 feet from the face, was blown outside and the flame extended 20 feet beyond the opening. The explosion was caused by a blown-out shot, the charge being black powder and the amount less than 3 pounds. The owners of the Vulcan mine in Colorado, desiring to make the mine as safe as possible, installed an elaborate sprinkling system by which it could be kept at all times in a thoroughly damp condition. Along the entries pipes were laid, perforated in such manner that the water in them, forced out under considerable pressure in fine sprays, moistened not only the bottom of the entries, but the sides and roof as well. The owners did not stop with only providing means for keeping the entries saturated. At the mouth of each room was a suitable arrangement to attach to the main supply pipe a hose long enough to reach the room face, and before a miner could fire a shot in his room it had to be made thoroughly wet. As the owners put in this apparatus of their own accord and not because the law required them to do so, it may be reasonably assumed that they insisted on its proper and effective use, yet notwithstanding the mine's moist condition produced by this means, a most disastrous explosion occurred. The mine had been examined and was reported free of gas. Every man in it was killed and the mine itself was almost destroyed. The mine, at the time of the explosion, was of comparatively small extent and well ventilated, between 55,000 and 60,000 cubic feet of air passing into it every minute. In view of the lesson taught by these instances, and others like them could be cited, it is not surprising that men should become skeptical as to the value of the effects of sprinkling or spraying with water. Mr. Payne declares that "coal dust cannot be made wet in the usual sense. The use of fine sprays is indicative of the best results, but even then it is hypothetical if the most careful system of watering is not merely an infinitesimal portion of the ounce of prevention." This view may appear to be rather extreme, but I believe it is very near the truth, and the artificial introduction of moisture in a naturally dry mine, while desirable and beneficial, should not be considered a dependable remedy for the prevention of explosions. Each mine has its own characteristics; conditions differ as between mines, and they may differ materially in the same mine; the power and heat produced by the primary explosion are never the same, and an amount of moisture that may promise a fair degree of safety in one case may be utterly insufficient to give protection in another. It must also be considered that while water may be used in some mines without detrimental effects on the roof and bottom, copious watering may injure others materially and thus create additional danger to the men working in them.

At the Altofts and Lievin testing stations extensive experiments have been and are now under way to establish the value of fine stone dust as a factor in preventing explosions from spreading through the main roads and entries of a mine. I do not know what conclusions have been or may be reached by the investigators as to the value of this proposed means for checking explosions, but I am decidedly of the opinion that in this respect, under the same conditions, the use of stone dust will show far better and more uniform results than the application of water, and that the finer and drier the stone dust the better the results, for the same natural method that propagates and enlarges the flame of an explosion by injecting coal dust into it, must produce the opposite effect by the forcible injection of the incombustible and flame-destroying stone dust; but in either case the influence of extraneous conditions will be a large factor in fixing results. The use of stone dust at the face is evidently not practicable and consequently, while apparently a palliative of considerable merit, it can have little part to prevent the occurrence of explosions in the working places.

The heating of the intake air in the winter and the introduction of steam into the intake are also said to have an influence in the prevention of explosions. On the former proposition Mr. Rice expresses himself as follows: "The plan of merely heating the incoming or intake air to produce summer temperature without introducing additional moisture has sometimes been proposed as a remedy. The fallacy of the argument will be obvious to one familiar with the principles of relative humidity and their application to mine air." It is possible that Mr. Rice was a little hasty in pronouncing judgment. I am fairly familiar with the explosion history of the United States and I know of only one explosion occurring in a nongaseous mine in June, and I failed to discover the record of a single explosion in a mine free from firedamp, that occurred either in the months of July, August, or September during the last 30 years or before. Surely this conspicuous and unvarying absence of dust explosions in the summer time could not have been due entirely to the fact that the air entering the mines at summer temperature always carried sufficient moisture to keep the dust in a thoroughly damp condition. Mr. Haas reports that during 4 weeks in September, 1908, the atmosphere was so dry, according to his calculations, the air carried out of the mines 20 per cent. more moisture than it carried in, representing for an ordinary sized mine a daily loss of about 3,000 gallons of water, and it can be reasonably assumed that there were many instances of long dry spells during the warm seasons of the last 30 years, causing results similar to those shown by Mr. Haas, and yet, under such presumably favorable conditions, not a single dust explosion occurred in a mine, although such explosions have been produced during warm weather under specially arranged conditions in experimental galleries located on the surface.

If Mr. Haas is right, and in my judgment he is absolutely right, that "the quantity of air rather than the quantity of dust or coal, is really the measure of the magnitude of an explosion," then the proposition is equally sound that the quantity and quality of the air, rather than the available fuel supply, determines the magnitude of the initial flame or the initial explosion, and consequently air conditions tending to increase or decrease the size of the initial flame become factors of greatest importance in either promoting or preventing an explosion's start.

It is a well-known fact, that, under the same conditions, with the same air volume entering a mine, the quantity of air contained in the mine, measured by weight, is less in the summer than in the winter, and that in the latter season the oxygen content of the mine may be considerably greater than in the former. Supposing a constant volume of 100,000 cubic feet of air is going into the mine every minute, then with an outside temperature of 25 degrees this air volume carries into the mine about 175 pounds more oxygen than the same air carries into

it in the same time at an outside temperature of 75 degrees, or, in other words, 10 per cent. more oxygen is delivered into the mine under the lower temperature than under the higher, representing an average increase in mine oxygen content of 5 per cent. with the same temperature prevailing at the end of the return in either case. If to a cubic foot of air .07 of a cubic foot of marsh gas is added, the mixture remains below the explosive point, but if the marsh gas volume is increased 5 per cent., or to .0735 of a cubic foot, the mixture becomes explosive, and so it may be reasonably assumed, other conditions being the same, that a 5 per cent. increase of oxygen content in a mine may represent the difference between comparative safety and threatening danger. The heating of the air at the intake in the winter is commendable, because it will at least reduce, if it does not prevent entirely, this oxygen increase and thus create less favorable conditions for the enlargement of the initial flame. If steam is used at the intake the results will be still better, because in addition to its heat producing a decrease in the mine's oxygen supply, the latter will be still further attentuated by the steam itself. Some claim that the greatest value in the use of steam consists in providing a considerable moisture supply. This feature is undoubtedly of benefit, but its value is uncertain, for while there may be a large moisture supply at the beginning, it will gradually become less as the distance from the intake increases and will be very much weakened, if not entirely exhausted, by the time the working places are reached where fresh dust is produced continuously and in greatest quantity and where the danger of its ignition by blown-out shots is the most threatening.

But no matter what remedies may be proposed, the results of their application will remain uncertain and unsatisfactory unless the intelligent and willing cooperation of the mine workers is secured toward contributing their full share in making any of these remedies of positive value, for it will always be true that the safety of a mine primarily depends on the intelligently directed efforts of every individual in it and not on the effects of law enactments or the presence of safety devices. In my judgment the best way to secure such cooperation on the part of the mine workers is by educational methods, by giving them plain and easily understood information regarding the fundamental principles governing explosions and by explaining to them the meaning of the many valuable lessons furnished by mine explosions in the past. These men should have the chance to know the truth, and when they realize to what extent they can promote their own safety, and have the proof to convince them that their help is essential for the benefit of all, they will be intelligently and effectively prepared to give their assistance to the best advantage. The mine workers should be shown that fine and apparently safe mine conditions alone will not prevent the occurrence of dust explosions, and they should know that, as far as these explosions are concerned, large air volumes entering the mines, perfect ventilation, and the existence of numerous openings are not safeguards but, rather, factors of aggravating influence. They should know that the presence of the coal dust is not the all-important thing and that its dangerous characteristics can be developed only by the acts of man and through the influence of other conditions. They must be convinced that the only dependable way to prevent a dust explosion is through the prevention of the initial flame, and while every possible additional safeguard should be provided, it should be impressed on them that their own efforts are of the greatest value and that implicit reliance in the protection of such safeguards or palliatives, whose effectiveness is largely determined by surrounding conditions and may be easily impaired at any time, leads into danger.

Thorough knowledge of the danger incident to his work is the best aid the miner can have in providing for his protection against it, and a moral obligation is resting on the men connected with the coal-mining industry of this country to see to it that reliable advice and information are furnished to enable him to acquire such knowledge in the fullest measure. It was suggested at the Scranton meeting of the Mine Inspectors' Institute of America that the mine inspectors take up this educational work. These men will help wherever they can, but the teaching force is too small; it should include also every mine manager in the United States and every official in the Mine Workers' Union, who may be qualified to assist in the work. If it is profitable for the miners and operators of each state to come together to discuss and adjust wage scales and working conditions, joint efforts by the miners, operators, and mine inspectors for the purpose of investigating and devising means for the prevention of mine accidents will undoubtedly be productive of good results. The plan is feasible and practical it is a humane undertaking in which all should take part, for a united effort in this direction will certainly prove the most potent factor in the preservation of life.

CATALOGS RECEIVED

In writing for catalogs, please mention Mines and Minerals

The Allis-Chalmers Co., Milwaukee, Wis., Bulletin No. 1403, The Hancock Jig, 16 pages.

The Bristol Co., Waterbury, Conn., Bulletin No. 131, Bristol's Recording Voltmeters, 44 pages.

John Davis & Son (Derby), Ltd., 110 West Fayette St., Baltimore, Md., Bulletin No. 114A, Miners' Safety Lamps, 28 pages.

Electric Service Supplies Co., Philadelphia, Pa., Catalog No. 4, "Protected" Rail Bonds and Bonding Tools, 50 pages.

General Electric Co., Schenectady, N. Y., Bulletin No. 4806, Electric Fans, 42 pages.

Jeffrey Mfg. Co., Columbus, Ohio, Bulletin No. 44, Jeffrey-Griffith Cross-Over Dump, 4 pages.

Jeanesville Iron Works Co., 115 Broadway, New York, N. Y., Catalog No. J-36, Jeanesville Centrifugal Pumps, 40 pages.

Ingersoll-Rand Co., 11 Broadway, New York, N. Y., Bulletin No. 7004, Cameron Steam Pumps, 12 pages.

Milwaukee Locomotive Mfg. Co., Milwaukee, Wis., Bulletin No. 101, Milwaukee Locomotives Gas Driven—Mining Type, 24 pages.

Pacific Tank Co., San Francisco, Cal., Catalog No. 7, 128 pages.

Servus Rescue Equipment Co., Newark, N. J., The Servus Oxygen Rescue Apparatus and Accessories for Mines, 4 pages.

Sullivan Machinery Co., Chicago, Ill., Bulletin No. 58, The Sullivan Straight-Line Air Compressors, 20 pages.

Geo. D. Whitcomb Co., Rochelle, Ill., Gasoline Mine Motors, 12 pages.

The Webster Mfg. Co., Chicago, Ill., "Our Modern Manufacturing Plant," 8 pages.

De Laval Steam Turbine Co., Trenton, N. J., Catalog "A," De Laval Steam Turbines Single Stage Type, 120 pages.

The J. H. Montgomery Machinery Co.,1445 Thirteenth St., Denver, Colo., Automatic Systems of Aerial Wire Rope Tramways, 48 pages.

Stromberg-Carlson Telephone Mfg. Co., Rochester, N. Y., Circular No. 251, describing the Mine-A-Phone; also circular describing the Hear-A-Phone.

The Industrial Instrument Co., Foxboro, Mass., Catalog No. 30 (36 pages) devoted to the complete line of Dr. Horn Tachometers and Pachographs. All speed measuring instruments from simple speed indicator to count revolutions, to the elaborate precision variation recorder.

Hendryx Cyanide Machinery Co., 107 William St., New York. Supplement to Catalog No. 7, describing the Hendryx Agitators and the Hendryx Storage or Tailings Dewaterer, 12 pages.

GASOLINE MOTOR HAULAGE

*Written for Mines and Minerals, by Geo. E. Sylvester**

The first furnace south of the Ohio River, to make pig iron with coke as a fuel, was established at Rockwood, Tenn., in 1868. Previous to this time there had only been some small operations using charcoal as a fuel.

Haulage Conditions Under Which Gasoline Motors Show a Saving Compared With Some Other Systems

The locality at Rockwood proved to be well adapted to the establishment of a furnace plant, as the outcroppings of the iron ore, coal and limestone, necessary to the manufacture of iron, were all found within a few hundred yards of the furnace site.

The first furnace was small, and of the hillside variety, with a capacity of 15 tons daily. All pig iron was shipped out, and supplies brought in, by river, 4 miles distant, as there was no railroad through this section at that time.

The plant has been practically rebuilt a number of times since then, and today consists of two 150-ton modern furnaces, shown in Fig. 1, with coal and iron mines, and limestone quarries, necessary for the production of the raw material for the manufacture of pig iron. These furnaces, situated at the foot of Walden's Ridge, have yard connections with two railroads, the C. N. O. & T. P., and the Tennessee Central. The company's main coal mine is located just above the furnaces in the base of the Ridge, the mine extending under the Ridge and the Cumberland plateau to the northwest. This opening was made in 1876, openings having been previously made and worked further up the outcrop to the westward. All of these workings were sooner or later connected, most of the older workings being, however, now abandoned. More than a thousand acres have been covered by these mining operations.

Fig. 1. Furnaces, Roane Iron Co.

The question of haulage has been for a long time a serious problem to the operators of this mine, and to show the difficulties presented it will be necessary to go to some extent into the detail of the coal formation. The coal, which is known as the Sewanee seam, has an analysis as follows:

Washed, Run of Mine

Moisture	.60
Ash	9.00
Volatile matter	32.09
Fixed carbon	57.69
Sulphur	.62
	100.00

It makes a dense hard coke which is exceptionally good for blast-furnace use.

This seam of coal, which outcrops on the face of the Ridge, dips down under the mountain, and is irregular, both in the amount and direction of dip, and in the thickness of the bed. The coal is in pockets separated by barren places, which show only the formation, with from a trace to a few inches of coal, and from which the coal seems to have been squeezed out. These barren territories are, on this account, locally known as squeezes. The coal pockets are often quite large, sometimes many acres in extent, and coal to the thickness of 20 to 30 feet, and even more, is sometimes found. The very uncertainty of the extent and location of the pockets and squeezes, adds difficulty to the proposition of mining.

*Mining Engineer, Rockwood, Tenn.

The mine is worked by a slope about a mile in length, with rope haulage, the maximum pitch being 15 degrees. From this slope there are three main entries, each of which is over 2 miles in length. These entries are driven on a drainage grade of from 1 to 2 per cent., the grade being in favor of the loads. In order to keep the entries on a uniform grade, it was necessary to make them crooked, the miner paying no attention to alinement, but swinging to the right or left to keep on the seam. In running these entries, no attention was paid to whether they ran through coal or squeeze, the object being to develop territory. These main entries present much the same appearance, when shown on the map, as would a wagon road, located with a regular grade, through a mountain section, varying to the right or left to avoid or go around each local hill or hollow.

The entries are from 800 to 1,600 feet apart, most of the local development being made by cross-entries on the pitch of the seam, operated by gravity plane or air hoisting engine, with rope haulage, the general system being such as will work out the coal pockets with as little narrow work in the squeezes as possible. These barren places, being thus left intact, act as extensive pillars, and preserve the mine against any general creep.

The coal is collected to side tracks on the main entries by mules, or by rope from the cross-entries, the nearest point of collection being a mile and a half from the slope.

Mule haulage was formerly employed on these main entries, and it was this long entry haul that presented one of the greatest difficulties in operating the mine.

The total output of from 600 to 700 tons daily had to be hauled from 1½ to 2 miles on these entries, and any trouble or delay at once cut the output for that day below normal.

The management for years has been trying to find some system of mechanical haulage to replace the mules on this entry haul. Electricity, compressed air, and rope haulage, have each been carefully considered; none, however, was found which it was thought would fulfil the difficult conditions presented.

Knowing that the internal combustion engine had been applied to mine haulage in Germany, it seemed that a gasoline mine motor could be made to answer these conditions, but nothing on the market could be found which was adapted to this class of work.

About 2 years ago, learning that the Geo. D. Whitcomb Co., of Rochelle, Ill., were perfecting a gasoline mine motor, the matter was taken up with them, with the result that in April, 1910, one of their motors was installed in the mines.

The results were so satisfactory, that two other motors for the other entries were at once ordered, and were installed in October, 1910.

All the extra work in the mines, necessary for the installation of these motors, was some slight trimming of the rib and top in places, so as to give ample clearance for the motors; and going over the track to replace with 20-pound rail, the places on the entry where a lighter rail had heretofore been used.

There was no difficulty found by reason of the many curves, as the motors have a 4-foot wheel base, and can take a curve

of 25-foot radius. The locomotives are 6 tons each, and were built for the mine gauge of 33 inches. They are designed with 4-cylinder engines, of ample power to slip the wheels, and all parts are well protected, as is necessary for mine use.

The trouble feared was that the gases from the exhaust would be objectionable. This has not proved to be the case however. The motors work on the return air, of which there is from 15,000 to 18,000 feet per minute on each entry, and a

Fig. 2. Trip Coming Out of Mines of Roane Iron Co.

short distance from the motor the fumes from the exhaust are hardly noticeable.

The mine cars used are about 1,400 pounds in weight, and carry $1\frac{1}{8}$ tons of coal. As the grade is in favor of the loads, the empty cars up the entry make the load for the motor. The regular 20-car trips are handled without difficulty, and on trial trips 40 cars have been taken up the entry.

These three motors have already replaced 23 mules; the comparative estimate of mule and motor haulage on one entry being as follows:

Cost of Coal Haul on No. 2 Entry, $1\frac{1}{2}$ Miles or 3 Miles por Round Trip

10 twenty car trips equals		224 tons
By mules:		
4 drivers, at $1.65	$6.60	
9 mules, at $.50	4.50	$11.10
By motor:		
1 motorman, per day	$2.05	
1 coupler, per day	1.65	
13 gallons gasoline, at $11\frac{1}{2}$ cents	1.50	
2 pounds carbide, at 4 cents	.08	
$\frac{1}{2}$ gallon gasoline engine oil, at 23 cents	.12	
1 gallon transmission case oil	.24	$ 5.64
Saving by motor		$ 5.46
Or, 49 1 per cent.		

These motors use 12 to 13 gallons of gasoline each, per shift. The gasoline tanks, of which there are two on each motor, are so placed in the frame as to be well protected in case of derailment or accident. The tank can only be filled when detached from the motor, and in changing these tanks it is necessary to have the valve closed. There are two extra tanks with each motor and these are filled on the outside. When brought into the mine they are perfectly sealed until after being exchanged with empty tanks on the motor. There is therefore no handling of exposed gasoline in the mine.

While these motors require the same care and attention as any similar piece of machinery, the advantage of having a self-contained machine, independent of any other source of power, and ready for use at any time, with nothing to provide for it but the track, is apparent.

POCAHONTAS COAL AND COKE SHIPMENTS

The Crozier Land Co. have issued their 1910 chart of the annual shipments of coal and coke from the Pocahontas Flat Top coal field in West Virginia.

In 1883, the Pocahontas colliery, now the Pocahontas Consolidated Colliery Co., was opened and shipped 60,828 tons of coal and 19,805 tons of coke.

In 1890, there were 16 coal operations which in that year shipped 387,076 tons of coke and 1,808,943 tons of coal.

In 1900, there were 40 collieries in operation that shipped 1,197,294 tons of coke and 4,253,228 tons of coal.

In 1910, there were 69 coal operations that shipped 2,084,023 tons of coke and 10,269,581 tons of coal. Since 1907 the increase of production of Pocahontas coal has been almost at the rate of 2,000,000 tons a year. Since the opening of the field in 1883, 25,231,665 tons of coke have been made and shipped, and 102,767,042 tons of coal have been mined and shipped. The increasing shipments from this field are due to the good qualities of the coal and coke.

Taking the Pocahontas field as a whole, there is not probably another coal field in the world which is so favorably situated for cheap mining. Most of the coal mined up to the present time has been above water level, so scarcely any expense has been required in pumping compared with what is required in other fields where equally good coal exists.

The thickness of the coal bed would probably average 6 feet, and this favors mining. Advantage is also taken of the slight inclination of the coal bed and strata underneath so that loaded trips generally are hauled down grade, while empty trips are hauled on the up grade, which in most cases is very slight, probably a little less than 1 per cent.

During the publicity period when it was necessary to educate people to the value of Pocahontas coal, the selling price of this coal was low. At present, however, the coal is so well established in the markets of the United States that it should bring a price that will be remunerative to the miners.

In some dusty coal mines it is customary to sprinkle the coal with water to prevent dust explosions. It has to

Fig. 3. Whitcomb Gasoline Mine Motor

be frequently repeated, the atmosphere becomes damp, and the mine muddy, and it requires much labor. The Kruskoff process substitutes a paste for water. Coal dries in 6 hours when sprinkled, but remained damp 3,000 hours when treated with this paste. Comparative experiments showed that a dynamite charge of 5 grams produced a coal-dust explosion when water was used, but when the coal was treated with the paste, 12 successive charges of 100 grams failed to produce an explosion.—*Engineering.*

BAILING WATER AT COLEMAN SHAFT

*Written for Mines and Minerals, by F. Ernest Brackett**

During a difficulty with the pumping plant at Coleman shaft, Cambria County, Pa., the local management was forced to try out a pair of 1,200-gallon water bailers, one of which is shown in Fig. 1. A brief review of the matter may be of general interest.

Bailing Skip and Arrangement of Automatically Placed Chutes for Carrying Off the Water

The Coleman shaft, about 660 feet deep, was finished in January, 1909. Considerable difficulty was met in sinking this shaft on account of water. As gauged by the barrel the hoist shaft made during sinking as high as 800 gallons per minute, and the air-shaft added about 350 gallons to this. The maximum water occurring in both shafts simultaneously was about 1,100 gallons per minute. When the shafts were finished the water yielded by both shafts did not exceed 1,000 gallons per minute, and at the time of the trouble with the pump, the shaft and mine water together amounted to about 800 gallons. It may be of interest to know that although during sinking the increased amount of water in the hoist shaft caused the sinking of that shaft to lag behind the air-shaft, which was only about 160 feet away, this fact did not materially increase the water in the air-shaft, or decrease it in the hoist shaft, although as a rule the sand rocks were full of fissures.

The coal at the foot of the shaft dipped at about 2 per cent., the main working being to the rise. A number of drainage plans were discussed before opening the main pump room, but it was finally decided, in order to avoid rock work, to drive a few places in the coal for the sump, far enough to the dip to be below the shaft track system. The pump was then placed about half way between the bottom of this sump and the shaft. To make the suction short, a ditch from 6 to 8 feet deep was dug from the lowest point of the sump back to the pump. The cage pit being lower than the bottom of the sump was drained by a small auxiliary pump. It was the original intention to ditch from the pump room to the cage pit, so that when water reached the main pump level it would flow into the main cage pit. The peculiar contour of the coal, however, made this unnecessary, as the water drained to the cage pit without ditching.

For use in case of a bursted column pipe, or a sudden inrush of water a pair of 1,200-gallon Wellman-Seaver-Morgan automatic water skips were purchased. These are placed on the hoisting ropes after the cages have been removed; the extra length of rope required being provided by attaching an extra piece. Each skip has a pair of automatic door valves in the bottom, which when submerged open inwards by the pressure of the water in the shaft pit. One of these valves is connected by a system of levers inside the skip to a lever at the top of the skip which comes in contact with a striking piece at the top of the shaft, and opens the valve, thus allowing the water to escape. The water chutes at the top of the shaft are arranged to take their places automatically beneath the skip before the discharge gate is opened. Before the skip descends, the chute moves out of its path. These chutes, of which there are two, one for each hoisting compartment, take their motion from the skip by means of a rope connected to a yoke through which the hoisting rope runs, as shown in Fig. 2. By this arrangement in connection with the waterway from the pump room to the shaft, the mine could be unwatered by means of the skips in spite of the entire failure of the pumping plant. The pumps could then be reached from the air-shaft.

At the time of the trouble only one pump had been installed. This was of the triple-expansion outside-packed plunger type, having two water ends and six steam cylinders. The capacity of the pump at full speed is almost 2,000 gallons of water per minute. It formed what the designers considered the first unit of a plant which would ultimately be extended to several units, so that for the time being there was no relief for the main pumping plant except the water skips. There were some quite heavy pumps in the main shaft which had formed part of the sinking outfit, but these had been so far dismantled as to be incapable at the time of the shutdown to handle any considerable amount of the water.

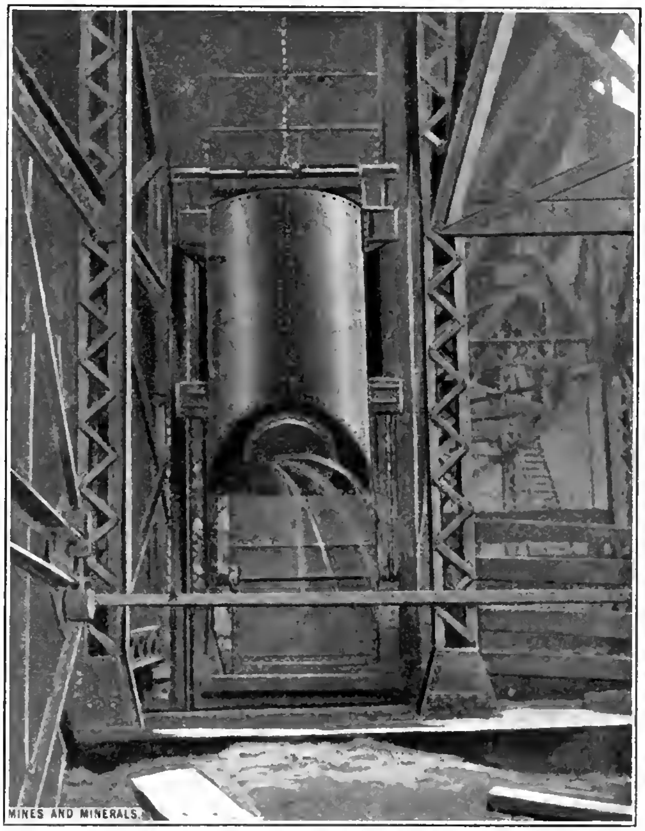


FIG. 1. REAR VIEW OF WATER BAILER

The main pump had not been working properly for several days previous to the shutdown, owing to some obscure trouble with the valve gear. It may be remarked that although the valve gear was by no means complicated, still in a duplex compound pump with 14 rotating valves, where each side drives the valve gear of the other side, and where steam pressures are passed from cylinder to cylinder, and even from side to side, the difficulty in locating valve-gear trouble in a short time is not to be overlooked. There was room for about 8 hours run of water in the sump at the time the trouble began, but between attempts to repair and failure of pump to run properly, the 9th of June found the water up to the pump's steam line leading to the shaft.

The pumpmen remained at work until about 8 A. M., when the steam became so wet that the pump stopped, and the pumpmen narrowly escaped from the room, by reason of the neighboring galleries to the exit being unexpectedly hot from the evolution of steam from the steam line. Probably this trouble would have resulted fatally if the steam line had not been covered with magnesia and tar paper.

Immediately after the pump stopped the timbers used under the cages at the bottom for rests were removed, and the valve at the foot of the shaft in the steam line to the pump was closed. At 10 A. M. the work commenced on the removal of the north cage preparatory to hanging the north water skip. This skip was working at 9 P. M. and water was bailed with it all night. In the morning it was found that the water in the mine had only lowered 3 inches. The second, or south skip was then hung. This occupied from 7 A. M. until 6 P. M., when both

* Civil and Mining Engineer, Cumberland, Md.

skips were working. In hanging the second skip it was also necessary to hang the chute, so that the same work which occupied 10 hours for the north skip, occupied only about 8 hours for the south skip, and the time could probably be shortened further by the careful drilling of men.

On the next day the water had been lowered about 2 feet and the pump room became accessible from the air-shaft. A bucket was attached to the air-shaft hoisting engine, and men were lowered to work on the pump. On June 17 the pump was again started. The bailing of the water in the meantime was carried on intermittently by the main hoisting engine. It was found that bailing from 20 minutes to 30 minutes per hour was sufficient to keep the water down.

When bailing with one skip it was found that a skip of water was delivered every 75 seconds, but by a slight effort a skip could be delivered in 60 seconds. When two skips were in use the time necessary to deliver a skip was from 31 to 38 seconds, averaging 34 seconds. Of this time 20 seconds were occupied in hoisting the skip 700 feet, and the remaining 14 seconds were occupied in slowing down and dumping. The actual dumping only occupied about 5 seconds. The capacity of the two skip hoists was therefore $1,200 \div 34 \times 60 = 2,120$ gallons per minute.

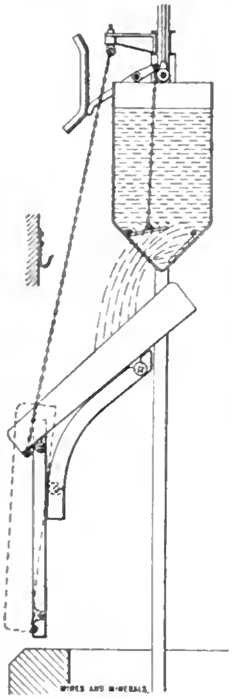


Fig. 2

The amount of coal consumed while hoisting the 800 gallons per minute made by the mine at this time was 23 gross tons per day of 24 hours. It was estimated that 85 per cent. of this, or 19 tons, was consumed in hoisting the water. The consumption of steam by the hoisting engine, as computed from its dimensions was 74 pounds per useful horsepower per hour. As it requires 141 horsepower to hoist 800 gallons per minute 700 feet, the amount of steam required would be 250,416 pounds per day. Dividing the water by the coal, the efficiency of the boilers on this kind of intermittent work is only 6 pounds of steam per pound of coal. The duty of the plant, as computed from the above data is about 15,400,000 foot-pounds of work per 100 pounds of coal, which is extremely low.

From an economical standpoint, therefore, the system of hoisting water in this case is deficient. The economy of the system as here applied was of course only a secondary consideration, but the figures are interesting, as showing what can be done. There are several main causes for this deficiency in economy. In the first place, rapid hoisting on short winds never has been an economical proposition, as there is very little opportunity to use the steam expansively. Even should an attempt be made to do so, the time occupied in getting up speed, when the steam must be admitted at nearly full stroke, occupies a large percentage, sometimes all, of the total time under steam. In the second place, there is a large amount of power wasted during every wind by the application of brakes to bring the load to rest. Besides these, the intermittent use of steam necessarily interferes with the economical operation of the boilers. Enough steam cannot be raised during the demand for it without wasting fuel during the time when there is little or no demand for it.

By careful design, especially on winds of greater length, no doubt these losses can be reduced to some extent, but as a general proposition the plan of hoisting water instead of pumping it should not be adopted, unless there exists some other reason that is of greater weight than the economical side of the question.

Trade Notices

The Chicago offices of the Atlantic Equipment Co. and the American Locomotive Co. have been moved to suite 907-912, McCormick Building, Michigan Boulevard and Van Buren Street.

LeRoy No. 2 mine, at Rossland, B. C., which was damaged by fire about a year ago, is to be reopened. The management would like to receive catalogs of mining and milling machinery for metalliferous mines, also hoisting engines, cables, rope haulage, pumps, machine drills, etc. Address A. Burnett, superintendent LeRoy Mining and Milling Co., Rossland, B. C.

The Sullivan Machinery Co. announces the establishment of a branch office at 35 Federal Street, Boston. Mr. George E. Wolcott, for several years manager of New England sales at the company's Claremont, N. H , office, is in charge and all correspondence shou d be addressed to Boston, instead of to Claremont, in order to receive prompt attention.

Coal operators in Pennsylvania, Ohio, and the northern portion of West Virginia, known as the B. & O. district, will be given prompt and careful attention by directing their inquiries in connection with Goodman haulage and coal-cutting machinery to the branch office of the Goodman Mfg. Co., 2218 Farmers Bank Bldg., Pittsburg, Pa The Pittsburg representation of this company has recently been strengthened by the acquisition of two experienced mining men, Mr. L. F. Crawford and Mr. J. P. Cameron. Mr. Crawford is a man of technical education and training in electrical and mechanical engineering. He has served as superintendent and in other capacities at mines in the West, and has had 6 years experience in the selling and installing of coal-cutting equipment in West Virginia. Mr. Cameron has had varied practical experience in the use of electrical apparatus. He is an expert, particularly in haulage matters. Mr. L L. Brande, long associated with the Pittsburg office of the company, continues as salesman and engineer.

The Boston sales office of the Link-Belt Co., has been moved to more commodious quarters at 131 State Street, Boston. Mr. Lawrence Spillan is in charge.

Large contracts for Garton-Daniels lightning arresters have recently been closed by the Electric Service Supplies Co. with the following companies: Rockford & Interurban Railway Co., Rockford, Ill.; Chicago Railway Co., Chicago, Ill.; Birmingham Railway, Light and Power Co., Birmingham, Ala.; Stark Electric Railroad Co., Alliance, Ohio, and many others.

The Lunkenheimer Company, of Cincinnati, Ohio, has opened a new branch store at 138 High street, Boston, where they are carrying a full line of Lunkenheimer high grade engineering specialties. The store occupies a prominent corner on High street, and is in charge of Mr. Wm. W. Beal.

Simonds & Burns, mining engineers, of New York, have moved their offices to 55 Liberty Street, on the 30th floor.

The Williams Patent Crusher and Pulverizer Co., Old Colony Bldg., Chicago, state that for the first three months of this year their sales of crushers were larger than during any similar period for the last 15 years. They have issued a new catalog describing the crushers which will be sent on application.

The Erdle Perforating Co., of Rochester, N. Y., who have been known as manufacturers of perforated metals for many years have recently made some extensive additions to their plant and are now prepared to furnish perforated metals of all kinds, either heavy or light.

The Sullivan Machinery Co. announces the establishment of an office at 814 Salisbury House, London Wall, E. C., London, England, which will in future have general charge of their business in England and on the continent of Europe. Howard T. Walsh, formerly manager at the San Francisco office of the company, is in charge, and Mr. Arthur F. Belding, recently of the Joplin office, is associated with him.

COAL TRUSTS AND SAFE MINING (?)

Written for Mines and Minerals, by Sim Reynolds in collaboration with W. H. Reynolds

If some man having unlimited time, patience, and means, were to interview in turn every mining official in this country, and request of each his idea as to a perfect, or at least an approximate, solution of our chief mining problems, that man's notes would be the most tessellated writing in all mining literature. Yet each no doubt would be sincere in what he expressed. From our own meagre experience in this direction we can frankly assert that he would get ideas running from a common center to the extremity of right and left angles at least, and some of the extremes would be even more than diametrically opposed.

Not Necessary to Squeeze Out the Small Operators to Ensure Safe Conditions

This much for the proverbial diversity of human opinion, and as a prelude to these few remarks venturing into the lists with Mr. John A. Garcia, Superintendent of the Brazil Block Coal Co., of Brazil, Ind., who in a recent issue of another journal advocated large combinations in coal mining, as a means to reduce the number of accidents.

Our friend finds blood, bone, and sinew awaiting us in the combining of mining operation; i. e., by practically squeezing out the many small operators and giving the control of operation, market, and prices, into the hands of those having unlimited capital to carry out certain methods of mining. The gentleman's solution of our difficulties is, to say the least, ingenious, even if impracticable. He would have us formulate a plan under which certain beds should be won, plans for all mines being suggested along the line of extreme elaboration, which on the face of them would carry a certain elimination of The Great Unfit. And while directly the article in question has reference only to the Illinois and Indiana fields, yet it bears indirect allusion to every coal region in the United States, since what will ultimately prove the solution for one coal-producing area will, modified only by local conditions, prove the salvation of them all.

Mr. Garcia refers particularly to present conditions in Illinois and Indiana, and quotes a mortality list of 213 dead and 874 injured in one year in the former state alone. This, without any extensive and unusual disaster, and for a coal field practically free of the most destructive and subtle agencies of a large mortality (explosive gas, coal dust, and draw-slate) is indeed, to quote from said article, "a pretty bad record," and carries with it a self-evident accusation of grievous carelessness anent the common laws already provided for safe mining in Illinois on the part of the employes, or gross mismanagement on the part of subordinate officials, or both. Apparently the official end is as deep in the mud as the rank and file in the mire, and, as another writer in the same issue of the same journal openly suggests regarding another field, to blame either alone would be simply an obvious attempt to shift responsibility; a case of the pot calling the kettle black. We agree with him that "this looks bad even on paper," but to the man who, being on the spot, sees with his own eyes the terrible results as exemplified in the poor, battered bodies of the dead diggers, and the not less horrible, but even more pitiful, result when the children whom these "dead diggers" cared for in their simple but honest way, are forced out upon the mercy of indifferent strangers, it tells a story of mute tragedy more vivid and fearful than the printed type can ever convey. It does indeed, to quote Mr. Garcia a little further, "bring to the mind of every worthy mining man the flush of sorrow and anger and shame, anger that such things can happen in a country as enlightened as ours, sorrow for the burden-bearers in the form of little children and women folk, and shame that we have not applied the remedy. . ." Aye! there's the rub. What is the remedy? How many who read this would say the same thing? And does not your opinion carry equal weight with mine, all other things being equal?

Mr. Garcia's theory of it is very simple. If it were only as practical! In brief he would, for the Illinois and Indiana fields at least, form plans of safe mining which would involve the outlay of much money. He would have invested in the general or state government the power to close up immediately that mine whose owner had not the means to carry out these safe-mining ideas to the letter. He would extend the same power that now applies to banking institutions to mining of bituminous and other coals. The point is well taken. Certainly the lives of men, and the conservation of vast mineral resources, are as valuable to any nation as the money deposited in bank. If one is deserving of the paternal care of a government why not the other?

With our contemporary's theory we cannot differ, but we strenously object to his method of bringing it into actuality. He would have only those engage in mining whose purses equalled in length and depth their ability and the articles of law. If they couldn't comply "kick them out and keep them out," and leave the production of coal to them who, at any cost whatsoever, would fulfill the law's requirements to the last letter, those who have ample capital to carry out the most elaborate plans of mining, this latter, in Mr. Garcia's mind, evidently being synonymous with safety! This, of course, would remove competition to a large extent, and enable the men engaged to regulate both production and prices concurrent with the necessities of the case. As a matter of fact Mr. Garcia's article is frankly headed "A Remedy for Overproduction," the safe mining being simply the outgrowth of the main issue; i. e., control of the market. In the paper before us, conservation of coal areas and conservation of life seems to us to be merely a bluster for the generic idea, the main line of reasoning seeming to run on the assumption that this admirable arrangement would give the legal status to the removal of wholesome competition in the mining world, and would offer the same reason for the regulation upward of coal prices. If the favored few carried their privilege beyond the bounds of fairness, to again quote the same article, "the Sherman Anti-Trust Law would get them quick!"

What a pity one cannot sometimes set on the printed page a hearty laugh. But while we deprecate Mr. Garcia's knowledge of current events, we sympathize with and commend his frankness. Few men in his position are willing to admit the need of a more strenuous control from the legal side. In fact few of us feel like admitting that there are any serious dangers in the mines we are interested in at all, at all. When serious accidents do occur they are usually at the other fellows' mines, and it is none of our business whether they could have averted them or not. We have enough to do to look after our own. Most mine managers will admit that unnecessary risks are sometimes taken and dangerous methods pursued, that some mines they know ought to be put in a better condition or closed down, and, yet—ask him any way he could make his own mine safer, or reduce the risks his own men are taking, and nine chances out of ten he would be dumb, deaf, and blind. Naturally so. Otherwise he would be in a peculiar position—the case diagnosed, the remedy known, and the doctor admitting he neglected to apply it. And for what reason? Excessive cost? Can we in West Virginia make these changes which mean an era of safer mining if they involve money outlay to any great extent, when over in Ohio they are still trusting to Providence and hammering along in the same old way? While I worked in Pennsylvania I frequently heard the same line of reasoning anent that state and West Virginia. Down here any old chump could open up and run in any old way or no way at all, just so he got out some coal. In common with Mr. Garcia's idea they seemed to think "safe mining" meant altogether an outlay of funds, and an utter impossibility in a stiff competitive market. But is it? We shall see. The same questions have

been asked and answered much nearer the present writers than Chicago is, and answered fairly satisfactorily for a period well nigh on two years, and answered in a manner which gives refutation to the argument that any such revolutionary methods as advocated by Mr. Garcia need be applied. Poor men, or men comparatively poor, have answered them to the indomitable department head's complete satisfaction as well as their wealthy neighbors and competitors in the same field.

For the present, however, we agree with Mr. Garcia that under certain conditions there is a considerable difference in the mine conducted safely and that one run on chance, both in result and cost. Particularly is this so when a mine has been run for a number of years in a happy-go-lucky manner, and then suffered to undergo a complete reformation. But the difference in cost per ton, if taken from the erection of the tipple to the withdrawal of the last chain stump, is not as large as he would have us believe, nor so far beyond the application by the comparatively poor operator, the multitudinous fellow one finds here and there and everywhere over this vast country. There will always be a difference, and a noticeable one, in cost of the total annual output of a mine which has stoppings and overcasts built of brick or concrete and steel beams and masonry, and the same output under less safe conditions in which flimsy brattices of paper-cloth or wood enter. But is the difference all in favor of the latter. Is not the net cost of good work, which means usually safe work, compensation in itself because of the general results accruing therefrom? Does not the cheaply run mine lose almost as much as it gains, if not more? I fancy accurate statistics would convince us that such is the case. Isn't the "cheap" mine management everlastingly trying to patch up this thing and that, or losing time with a breakdown here and a breakdown there, with expense for labor, as well as the result of stoppage of output, running on all the time? Who has not seen the pitiful result of miles upon miles of flimsy brattice (which is one of the greatest factors in safe mining) installed by some manager more economical than wise. A "safe" method does not necessarily imply an extra costly one either at the mine producing 3 flats or at that one producing 50 flats per diem. To make a complete change from the false standard to the true would of course involve considerable extra expense in a large mine, but that would be a miniature revolution, and with revolutions we are not concerning ourselves, but with the steady every-day mining plans in which each day's output stands the cost of that day's expense.

Between the producing costs of the man who does these things with a view to safety and ultimate economy of operation, and the man who dodges both law and necessity for temporary advantage (for the man who does not is either as a rule too dull to know better, or one who knowing better, does not expect to stay "on the job") one would naturally expect a temporary advantage in favor of the haphazard method. But it cannot, according to the law of economics, remain with him. The wooden trestle spanning a periodically turbulent stream may serve the engineering department of an up-to-date railroad for a season, but for making dividends year after year the preference is sure to go in the end to iron, or steel, or concrete, or masonry, with all its additional cost. The difference favoring the haphazard is merely abstract and inconsequential and individual, and not, as Mr. Garcia would have us believe, a fundamental part of the mining industry. The end in either case lies more with the weaver than the fabric. We have known men who could, and did, practically live up to the law's requirements and still produce coal out of the same mine cheaper per ton than their predecessors who had tried to sidestep it.

What we would seem to need more than more combinations of capital or more laws is more mining acumen, more practical ability in the management. An extra hundred dollars a year in this premise quite often leads to thousands in the company's treasury and a better-managed mine. The man who is alive to the economic necessities of an extensive mine is usually a strict disciplinarian, and thus the two essentials of economy and safe operation are found under the same hat. I never yet saw an ill-managed and illegally operated mine that was permanently a good-paying one. The two things are not synonymous at all, or rarely at best. And it is in the "haphazard" operations where the greater part of these single, double or triple fatalities, which run up such fearful totals, occur. A stroke of peculiarly adverse fate, which, like lightning, is apt to strike anywhere, will, sometimes, it is true, work out the same end by destroying the whole mine, such as occurred at several places in our state and other places. But these are the exceptional, and not what our contemporary is hammering at. That we are aware of, there is no evidence of any considerable lack of capital at any of the mines where large disasters have occurred in recent years. And the conservation of life which is being lost day after day, and month after month, by petty accidents due to carelessness on the part of the employes or subordinate mine officials, seems to us to call more for strict discipline in management to remove them than for any such remedy as suggested in Mr. Garcia's article. There is just as much chance for laxity in the million-dollar operation as in that which approximates nothing much more than a country pit. Here is where the elimination of the Unfit would work only good—where general well being would come out of specific sorrow. There are hundreds of mines whose operation is not quite as justified as it might be under just these circumstances, but the several state laws have made provision already for the change, and installed men whose duty it is to close them up. An enlarged capital at any or all of these mines would not imbue these public servants with a greater desire to do their sworn duty. If they are lax now they would be lax then. With good managers and conscientious inspectors they could not remain so. There is no poverty plea which will justify any of them for remaining a menace to human life. In West Virginia, for instance, we have a no small number who would have at some time during recent years been disqualified according to our friend's idea of fitness, yet, Spartan-like they have done their share in upholding the best record in safe mining ever made in this country, or, perhaps, in any other region in the coal-producing world having equal possibilities in the way of wholesale destruction of life. The whole state of West Virginia is burrowed with these "little fellows'" operations, and, almost as one man (or combine) they have shown conclusive proof that a meagre mine treasury is no valid excuse for manslaughter, even though, as was the case with this state a few years ago, they had all apparently been bent on seeing just how near they could skin the lid of Destruction. Evidently they about reached the limit, traversing a distance along that road which every mining man in West Virginia prays they may never reach again. In the entire year just ended, and for a considerable period previous to that, not a single life had been lost from those causes which previous to the present regime gave this region the blackest name in the mining world, and there have been noticeable decreases in the percentages of death and injury from every possible cause. And all this in a field which, taking it in its entirety, has more mines diffusing and making marsh gas and coal dust, not to mention the deadly "draw slate" present over a great part of the region, than any other coal area in the world of the same size. And this, too, in a region where it takes on an average less money to open a mine than anywhere on earth, considering the possibilities of output. How?

Without waiting for the slow action of legislatures, or the rocky possibilities of Federal control, without any wholesale combinations of mining capital, without forcing out of the race the scores of little "fellows" whose bread and butter and permission to mine and ship into the markets of the world was, and is, equally legitimate with their bigger and stronger neighbors, if rightly and lawfully carried out, without any of these things the departmental heads and the operators and managers—

large, medium, and small—reached what might be termed, for want of a better designation, "A Mining Gentlemen's Agreement to Quit Practicing Indirect Murder." In brief, they came to an understanding that each was to mine coal *safely* in the future, along a few simple lines formulated by a committee appointed therefor, each infractor to take what the department had in store for him without squirming. And it was given out quite frankly, and with a determination that left no room for doubt, that these unlegislated agreements would be carried out. Understanding this, everybody went to work to put his several house in the proper order for this new regime. Legislation could come or not, as it pleased. The laws already made, coupled to the amendments agreed on mutually by the mining men of the state, were quite ample, if enforced. It didn't take West Virginia mining men long to find out what was needed, once they got down to rock bottom and started to build a better record from there up, no more than it would Illinois and Indiana or any other field, if they have the grit to enforce what they agree upon. And who is better able to legislate for mining men than mining men themselves, I want to know? Who better understands what we need than we do? What to cut out and what to leave in? There was needed no other sanction than the change in men metamorphosed by their own wills from happy-go-lucky individuals into deeply earnest ones, with a renewed feeling of responsibility to some one else besides their own desire to make money—toward their better selves, for instance, the clear and perfect manhood that is in us all, whether we be delving for coal in Indiana or seeking for diamonds in Africa, toward that clearer perception which sees not only profit but the wife and children in "the mine blocks" beyond the tipple.

With this newer feeling these men went to work to erase as far as possible the bad record The Mountain State had made for herself, hoping by increasing the output and decreasing the death and injury roll to strike a fairer percentage. They started the campaign with the same firm and corporations, the same mines with the same opportunities for death in them, the same unskilled labor, much of it fresh from the vineyards and ploughed fields of Europe. It would have taken a miracle to work up any additional feeling of responsibility toward themselves in these miners, and the result had to be reached through different channels. It had to come, as it did, almost wholly through the men whom the state had given, and was giving as fast as they applied and proved themselves worthy, certificates of competency. The result has been beyond the expectations of the most hopeful among us. And no matter what an adverse fate may have in store for West Virginia in the future, the past year or two will for all time remain like a beacon-light on the sea of mining, pointing to the present and future generations of mining men what men can do when they will, without revolutionary change in the matter of ownership, Mr. Garcia's theory and the results in Indiana and Illinois coal fields to the contrary notwithstanding. It is convincing proof to any one who may be convinced that the means of safe mining lie to a greater extent than is sometimes acknowledged in the hands of the men on the spot—the men in charge—the superintendents, foremen, and fire bosses, and a coterie of public authorities in the mine inspector line, who have a burning desire and the determination necessary to compel adherence to the simple rules for safety. In short, the result in this region is another argument, if any were needed, in favor of intelligent, determined and concerted action as a means to gain any desired end.

MINING AT LETHBRIDGE, ALBERTA

*Written for Mines and Minerals, by A. T. Shurick, E. M.**

Coal mining at Lethbridge is attended with a peculiarly difficult problem due to an exceedingly soft and heavy roof of considerable thickness. This trouble is so pronounced, that before the present system of working was evolved, it was thought the coal could be recovered only by a large number of contiguous shafts of comparatively short life. Since these shafts were of considerable depth and hard to hold, due to the treacherous character of the material sunk in, this proved a serious obstacle to the development of the field.

Fig. 1. Tipple at Colliery No. 6, Alberta Railway and Irrigation Co.

The topography of the zone embracing this immediate district is transitional between mountain and plateau, its salient features being that of broad, gently rolling plains, traversed by streams which have cut deep and relatively narrow valleys.

The surface equipment of the more recently developed mines is modern and complete in every sense of the word. Fig. 1 shows the tipple and head-frame of the Alberta Railway and Irrigation Co.'s colliery No. 6, one of the most recent and probably best equipped plants in the field. The structure is of steel throughout resting on concrete footings, and is equipped with cross-over dumps, shaking screens, conveyers, elevators, and picking belts. All the tipple machinery is electrically driven with the exception of a steam ram for caging the cars; and, following the practice of the modern Pennsylvania breakers, each machine has an individual motor drive. An Ottumwa cradle loader is provided for loading the box cars.

The air-shaft is located close to the main shaft and has a low steel head-frame designed for a man-and-material hoistway. The fan, which is connected to the shaft by a concrete conduit, is a large Sirocco having a rope drive from a pair of Corliss engines; the fan is now being run with one engine, which supplies sufficient air for present needs.

The boilers, hoisting engine, compressed-air and electric-generating machinery are all located in one brick building, shown in the background of Fig. 1. The boilers are Babcock & Wilcox water-tube, equipped with mechanical stokers and ash-disposal system. Ample room has been provided for the addition of further equipment as may be required.

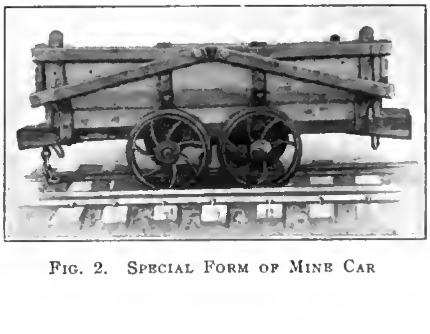


Fig. 2. Special Form of Mine Car

* Engineer Coal Companies' Amalgamated Copper Co., Washoe, Mont.

A feature that impresses the casual visitor is the extreme neatness and excellent manner in which the machinery is kept up; as the writer once heard remarked "you could eat your dinner off the floor any place."

Fig. 2 shows the type of mine car used, which has the rather unusual feature of a door at each end; this is made necessary by the methods of handling the trips underground. The car is low, very strong, and gives excellent service.

The seam lies approximately flat and at a uniform depth of about 500 feet, except in the deeper ravines where the cover is in some instances as thin as 25 feet. Where this latter is encountered, precautions are necessary to provide against inflows of surface water, but since the ravines are relatively narrow these conditions do not often prevail over extensive areas.

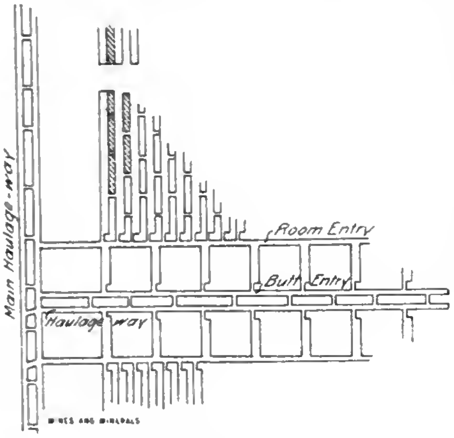


Fig. 3. System of Working

The shaft of the Alberta Railway and Irrigation Co.'s colliery No. 6 is 500 feet deep and equipped for hoisting in balance. Two cars, placed on the cage in tandem, are hoisted at one time, which necessitates a rather large cage and wide shaft. At the present time all the ponies are hoisted at the end of the shift, 2 or 3 ponies and 4 or 5 men to a cage. The ordinary cage, open around the sides is used, and after a few trips the ponies appear entirely unconcerned and indifferent to the experience.

The main haulage underground is usually an endless rope system. Numerous, and occasionally very persistent, faults having a throw of 10 or 12 feet are encountered, which would make the amount of grading necessary for economical electric haulage expensive. Roof conditions are also adverse to such a system. Colliery No. 3 of the same company has one endless-rope system 1½ miles long, the rope traveling the main entry in and back entry out. The rope runs at a speed of about 1.8 miles per hour, the engine being located on the surface and the ropes carried down the shaft. Small ponies are used exclusively to haul the coal to the rope system.

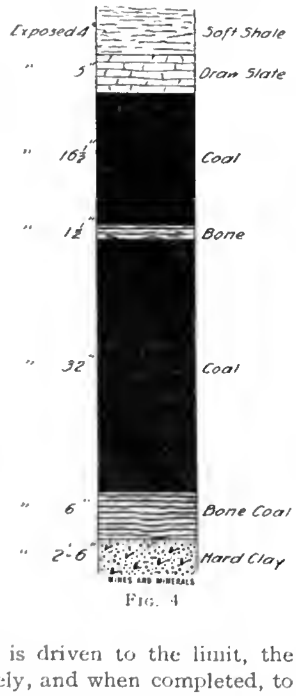


Fig. 4

Fig. 3 shows the generally accepted system of working used throughout the district. The main haulageway is the rope road, and butt entries are turned off this on both sides, with pillars of about 500 feet between. A butt consists of four separate entries, the two center ones being the permanent haulageway while the two outside ones are used as room entries. As soon as a room is driven to the limit, the pillars are brought back immediately, and when completed, to the room entry, the latter is abandoned from there out. This affords a good protection to the permanent haulageway, permits of a maximum recovery of coal, and has been found to be the only system by which the haulageways and airways can be kept open.

Extensive timbering is required on entries that are to be kept open any appreciable length of time, as in the case of the main haulageway. A timber set is made up of 4 pieces, 2 posts and a cap and sill, and on old entries these are almost without exception placed "skin to skin." In the newer workings the sets are placed on from 3- to 5-foot centers.

Fig. 4 shows a typical section of the seam. The soft shale at the top of the section is practically continuous through to the surface and accounts for the existing poor roof conditions. The mining is done in the 6 inches of bone coal at the bottom and shoveled back into the gob.

Ingersoll-Sergeant air punchers are used exclusively for mining. The coal is an excellent fuel, exceptionally hard, as compared with western coals generally, and withstands the deteriorating effects of long exposure with but little loss.

PROPERTIES OF CONCRETE*

Proportions (Cement : Sand : Stone)	Volume in Place Cubic Feet	Barrels Cement Per Cubic Yard	Cost Cubic Foot Broken Stone Concrete	Cost Cubic Foot Gravel Concrete	Ultimate Compressive Strength Broken Stone Concrete, 1 Month Old, Per Square Inch	Ultimate Compressive Strength Per Square Inch of Brickwork
1 : ½ : 1	6.8	3.97	$.370	$.340	5,340	
1 : 1 : 2	9.7	2.79	.314	.274	4,000	
1 : 1½ : 3	12.6	2.14	.283†	.237	3,200	835
1 : 2 : 4	15.6	1.73	.264	.214	2,700	
1 : 2½ : 5	19.0	1.42	.246	.195	2,300	
1 : 3 : 6	22.8	1.18	.230	.179	2,000	
1 : 3½ : 7	26.6	1.02	.219‡	.169	1,785	486
1 : 4 : 8	30.4	.89	.210	.160	1,625	
1 : 4½ : 9	34.2	.79	.205	.154	1,500	
1 : 5 : 10	38.0	.71	.200	.149	1,400	
1 : 5½ : 11	41.8	.65	.195‖	.145	1,320	347
1 : 6 : 12	45.6	.59	.192	.141	1,250	

* Derived and used by the Aberthaw Construction Co., Boston, Mass.
† First-class brickwork in cement mortar, 44 cents per cubic foot.
‡ Good brickwork in cement, 35 cents.
‖ Ordinary brickwork, 26 cents.

A SECTIONALIZED POWER PLANT

The El Tajo Mining Co., of San Sebastian, State of Jalisco, Mex., recently ordered a sectionalized power plant to supply power for its new mill. The head of water available is 500 feet, and there is a special Pelton waterwheel capable of developing 125 horsepower when running at 1,200 revolutions per minute. With this waterwheel there are two pairs of jaw clutch couplings, to render it possible to disconnect either of the two generators which are to be driven. The waterwheel and the two 50-kilowatt Westinghouse generators are sectionalized for mule-back transportation and no piece weighs over 300 pounds.

These generators are three-phase, 60 cycles, 600 volts and are to run at a speed of 1,200 revolutions per minute corresponding to the revolutions of the waterwheel. They have extended shafts for belt-driving the exciters. The generating station also provides for the usual switchboard, lightning arresters and protection apparatus.

The power generated is to be consumed at the mill by various alternating-current motors, one of which belt-drives the stamp-mill line shaft at a speed of 150 revolutions per minute resulting in the stamps making 100 drops per minute. The tube-mill line shaft runs at 200 revolutions per minute, the mill itself making 37 revolutions per minute. Other motors are scattered about the mill for the usual drives. Since the generating plant is located close to the mill, the generator gives 600 volts and the motors employed are for 550 volts. The lighting is taken care of by separate transformers.

McKeever Bros,. of Philadelphia, are largely interested in this property and the manager of the plant is Robert H. Lilly.

ELECTRIC SHOCK IN MINES

*Written for Mines and Minerals, by Sydney F. Walker**

The question of "earthing" or "grounding" electrical systems in mines has been rather hotly discussed, both in the electrical journals and at the meetings of the Association of Mining Electrical Engineers. It is now very much better understood than it was some years back. "Earthing" was supposed to be taken literally. In America they use a very much better expression, "ground." "Earth" or "ground" was taken literally. It was supposed for instance that if you drove an iron bar into the ground near the face, you made efficient "earth." As a rule you did nothing of the kind. To understand the *rationale* of "ground," it may be as well to refer back to its use in telegraphy and telephone work. It has disappeared from use with the telephone service, because of the interference of messages with each other, which the use of "ground" introduced. This, perhaps, will help to illustrate the difficulty of "earth" in connection with safety from shock in mines. With telegraphs, for instance, where a wire was employed for signaling between New York and Pittsburg, say, the return current was carried back from Pittsburg to New York, and from New York to Pittsburg, by way of the "ground." A connection was made usually to the water service in New York, and to the water service in Pittsburg, and the telegraph currents, which were very small, were allowed to find their way back as they could. In some experiments that were made in the United Kingdom in the early days of the telephone, it was found that the return current from Manchester to London, for instance, sometimes went round by way of Glasgow. For telegraph work it did not matter a bit how the current traveled, so long as the apparatus worked. With telephone work, the "ground" forming the common return, and receiving the currents sent by different subscribers often carried a number of other subscribers' messages to an individual telephone receiver, very much to the user's annoyance. In the very early days of electric light, "ground" was occasionally employed for the return current, but it was only in very rare cases that it could be so used. An instance that occurred in the writer's experience will perhaps illustrate the difficulty. An arc light had been fixed on a dock head, the generator being in an engine house at some little distance. After the light had been at work for some time, it was found necessary to move it farther from the engine, and there not being sufficient cable for the purpose, recourse was had to "ground." One end of a cable was attached to an iron bar and dropped into the dock, and the end of another cable was attached to another iron bar, and dropped into the dock, the circuit being so arranged that the water in the dock formed a part. The light worked fairly well when the dock was full of water, but when the dock was empty, it would not work. The resistance of the damp mud through which the return current had to pass, was so great that the current was reduced below the figure at which the lamp would operate. A similar set of conditions rules in most mines. It is quite easy to obtain good connection with "ground" on the surface. The engines usually make good connection with the ground. The boilers are always connected to some supply of water, which is good "ground" in itself. But when it comes to making "ground" in the mine, it is a very different matter. At the pit bottom there is usually a sump full of water, receiving the drainage of the shaft, and good "ground" can be obtained there. At the face, however, it is almost an accident whether there is good "ground" obtainable or not, and most frequently there is not. In order that good "ground" shall be obtained at the face, it is necessary to make good connection with the ground at the face, and that the electrical resistance offered by the strata between the face, where connection with the ground is made, and the sump or the surface, or wherever the other "earth" connection is made, shall be very low indeed. If the contention of the advocates of armored cable is to hold good, the resistance of the "ground" path must be sufficiently low to pass a current that will operate the circuit breaker, or the fuse, and cut off the cable, when the armor becomes "alive." In the great majority of mines, the electrical resistance of the strata varies enormously.

Importance and Difficulty of Grounding Electric Cable Armor and Apparatus

The investigations into the complaints about the electrolysis of pipes laid in the ground, owing to the stray currents from electrical tramway services, have disclosed the fact that the electrical resistance of the substances forming the earth's crust varies very much. In a great many cases the conducting power is almost directly in proportion to the quantity of water held by the strata. Porous water-bearing strata will have a low electrical resistance usually, while close non-water-bearing strata will have a high resistance. Strata which contain metals will have a comparatively low resistance, and vice versa. The strata themselves vary in the resistance offered. In addition to this, it is a very difficult matter to obtain good electrical connection to the strata. Even with telegraph and telephone work, the obtaining of good "ground" used to be a very troublesome affair at times. In dry weather, for instance, a telegraph apparatus might refuse to work because the connection to "ground" was so bad, owing to the absence of moisture in the soil in which the "ground" plate was buried, that the current necessary to work the telegraph apparatus would not pass. At the face of the coal it is particularly difficult to obtain good connection with the strata. In a large proportion of cases the strata underlying the coal seams are of clay, and clay offers a comparatively high electrical resistance, and is difficult to make connection with. It will be seen from the above how very complicated the question of the use of "earth" or "ground" is, when it is properly considered.

How Ground Protects From Shock.—The principle underlying the protection of men from shock by the use of "ground" is as follows: As stated above, men get shocks when some part of the body touches a conductor, between which and the "ground" on which they are standing a certain difference of pressure exists. The pressure required to kill differs, as already explained, with different subjects, but it is clearly established that a pressure of 150 volts alternating, either between the hands, or from one hand to the feet; or a pressure of 350 volts continuous, from hand to feet, or from hand to hand, will kill. A man receives such a shock, and places himself in a position to receive it, when standing upon the "ground" of a mine road say, or standing perhaps upon the rails, and he touches a conductor forming part of the electrical power service, or having a connection to the electrical power service, as explained above. If all conductors that it is possible for workmen to touch are made to assume the same pressure as the "ground" upon which they stand, it is obvious that shock will be impossible. For instance, the armor of a cable becomes "alive," by making connection with the conductor it is protecting, and if it is at the same time in connection with the "ground" upon which a man is standing he may touch it, but cannot receive a shock. Incidentally it may be remarked here, that it is very difficult indeed, even when good "ground" is obtained, to be sure that the pressure of an armor that has become "alive," is exactly the same as that of the "ground" upon which men in the neighborhood stand. It is usually possible, however, to reduce the difference of pressure between the "ground" and the live armor below the danger point, so that a man who happens to touch the armor, only receives a slight tingling shock that does him no harm.

The difficulty in the way of making the armor always at the same pressure as the "ground" has been explained above, and different methods have been adopted in the United Kingdom for overcoming it. The most promising in the writer's

* Bloomfield Crescent, Bath, Eng.

opinion is one that has been introduced in a large colliery in South Wales. An old wire rope has been carried down the shaft, and along the roads to wherever a cable is fixed, and wherever an electrical apparatus is at work. The armor of the cable is bonded to the old wire rope at frequent intervals, and when falls occur, the armor on both sides of the fall is immediately bonded to the rope. The rope is laid on the floor of the mine, and allowed to make as much connection with the body of the mine as it can. In the particular colliery, it happens to be a very well managed one, there are not many falls, and therefore there are not many cases of broken armor.

Two points are of importance, however, in connection with this matter. Any old wire rope will not do. Mining engineers know that in the older forms of wire rope, in which a certain number of wires were twisted together in one direction to form a strand, and a certain number of strands were laid up together in the opposite direction to form a rope, the individual wires came to the surface of the rope at certain intervals, and as the rope wore they were rubbed through. The consequence was that an old rope was made up of a number of short lengths of individual wires held together by the twist of the rope. Electrically speaking, the only connection between the individual wires was from surface to surface, and as wire ropes are kept well greased, and dirt penetrates to a large extent between the wires, the connection between individual wires by this path was not a good one. The electrical resistance therefore of an old wire rope of that type might be very high indeed, and it was only by using very large ropes, that in the very early days of electric lighting, an eminent mining engineer in the United Kingdom was able to utilize his old wire ropes in place of conductors. The modern forms of wire ropes, however, are very different. In the locked-coil rope, in the rope made on what is called Lang's lay, and in the flattened-strand rope, the wires wear very evenly, and an old rope when it is taken out of service for haulage or winding, will not have a very high resistance, because the individual wires are making good connection with each other, and there are not many wires rubbed through. With old wire ropes of this kind, the plan mentioned above will answer very well, providing that careful connection is made to it. And this is the second point that the writer would desire to call attention to.

Bonding requires to be done with great care. The old wire rope must be cleaned for several inches at least. It should be rubbed quite bright with emery cloth, and all visible dirt removed. Connection to the armor of the cable that is to be "grounded," or to switch boxes, etc., may be made either with a copper or an iron wire. A copper wire will be best, because it can be more easily wrapped around the wire rope, and around the armor, but where copper wires are used, the bondings should be inspected from time to time, as electrochemical action will be set up between the copper and the iron, tending to destroy the connection. A copper wire also will answer better because it has a lower resistance than an iron wire of the same size. In either case, whether a copper or an iron wire is used, it should be wound very tightly around the wire rope for 2 inches or more, and the armor of the cable should be treated in exactly the same way. It should be cleaned, and the bonding wire should be wrapped around it for at least a couple of inches. It is of importance that all switch boxes, distribution boxes, electric-motor cases, and any pipe or fitting that is used to protect wires, lamps, or other apparatus, should also be connected to the "ground" cable. The connection can be made in the same way, by wrapping a wire around the old wire rope, but connection to the switch boxes, etc., should be by means of large flat-headed screws, that can be well tightened down.

In another colliery in South Wales, another method has been adopted. Falls are provided against by building stone walls along the sides of the roads where cables are to be placed, the walls being of sufficient strength to support the roof under all circumstances, and to withstand the squeeze. A ledge is built in the wall, under which the cable is held upon an insulator, so that any squeeze which comes upon the wall, and which might possibly tend to grip the cable, is held off by the special shelf provided for it. In this colliery also, a "ground" wire is provided by a galvanized single strand wire, carried to the neighborhood of all the cables, and all the apparatus supplied by them. In the writer's opinion, it is doubtful if the single-strand wire would answer its purpose under all circumstances. The "ground" conductor, whatever form it may take, must have a sufficiently low resistance to allow any leakage current that passes out of any one of the cables to return to the generator without an appreciable difference of pressure arising in the "ground" conductor itself. If the "ground" conductor has a high electrical resistance, a difference of pressure may exist at certain parts of the mine between armor, switch boxes, etc., that are supposed to be "grounded" and the "ground" itself.

"Grounding" With Three-Phase Services.—"Grounding" with three-phase services is a rather different matter to that with continuous current. Star-connected generators and motors are usually employed in mines, and the neutral point of the star, the common junction of the three coils of the generator and motor, are connected to "ground." The same idea rules; viz., that if the armor of the cable becomes "alive" the circuit breaker or fuse will cut it off, and the armor itself being connected to "ground," a man cannot receive a shock from it. Provided that the armor is bonded everywhere to a good "ground" cable of low electrical resistance, this is nearly always quite correct. Peculiar cases arise, however, with three-phase services, in which "grounding" of the neutral may lead to conditions under which a shock may be obtained. Thus, if a motor stands upon stratum that offers a high electrical resistance, its case being connected to the "ground" cable, and a leakage occurs on one of the phase cables, it will be possible for a man touching the case of the motor to receive a shock. The conditions here are really inverted. The leakage from one of the phase cables has made the "ground" at a different pressure from the "ground" cable, and the case of the motor being connected to the "ground" cable, a difference of pressure exists between the case of the motor and the "ground" upon which it stands, and consequently a man touching the case of the motor bridges this difference of pressure, and may receive a shock. This will perhaps illustrate forcibly the great difficulty of the problem involved, and may lead mining engineers to understand why the present writer has always so strongly advocated insulation of every electrical apparatus about a mine before all things. If generators, motors, cables, switch boxes, etc., are insulated from the body of the mine and from every conductor about the mine, and if the insulation is maintained at a high figure, it will be very difficult indeed for anybody to get a shock. His view is, that with continuous currents, if one conductor is "grounded," and with three-phase currents, if the neutral is "grounded" you surrender half your defenses. It only needs one fault upon some part of the system to give trouble. On the other hand, if the insulation of the two sides of a continuous-current service and of all parts of a three-phase service are maintained at a high figure, it needs at least two faults to cause a breakdown, and the chances of shock are very much reduced. The writer will conclude with the remark he made earlier in the article, that the whole crux of the problem lies in the two words: insulation and care.

The government commission that has been prospecting for coal in the vicinity of Concepcion and Talcahuano, Chile, has just published its report, which states there are more than 80,000,000 cubic meters of soft coal within an area of 80,000 square meters, or about 30.7 square miles. In places the vein is 3.5 meters, or about 11.5 feet, thick. The coal is said to be of fair quality.—*United States Consular Report.*

MODERN HOLMEN COALING STATIONS

*Written for Mines and Minerals, by Clyde P. Ross**

Editor's Note.—An examination of the diagrammatic elevation in Fig. 1 will convince engineers that it is a "motion study" applicable to mining, metallurgical, and other industries.

Methods of Handling Coal from Cars Into Bins for Coaling Locomotives

The editor's request for a detailed description of this coaling station from Roberts & Schaefer, who own the Holmen patents, was turned over to Mr. Ross, and the following article is his reply:

Appreciating that most railroad and engineering journals have from time to time published articles on handling coal for locomotives, and assuming that the reader is familiar with the reasons, the writer will particularize on what is known as the Holmen system of coaling locomotives, which is the original and most widely used in service.

Recently the United States Steel Corporation has had constructed at its round house at Waukegan, Ill., a fireproof structural steel coaling station, an elevation of which is shown in Fig. 1.

The station has a storage capacity of 100 tons of coal and 8 tons of dry sand, and is designed to coal and sand locomotives on two tracks. The coal is brought to the station on the receiving track at the rear of the plant in hopper-bottom cars and dumped into a 26-foot concrete receiving hopper. The rails of this receiving track are supported over the hopper on

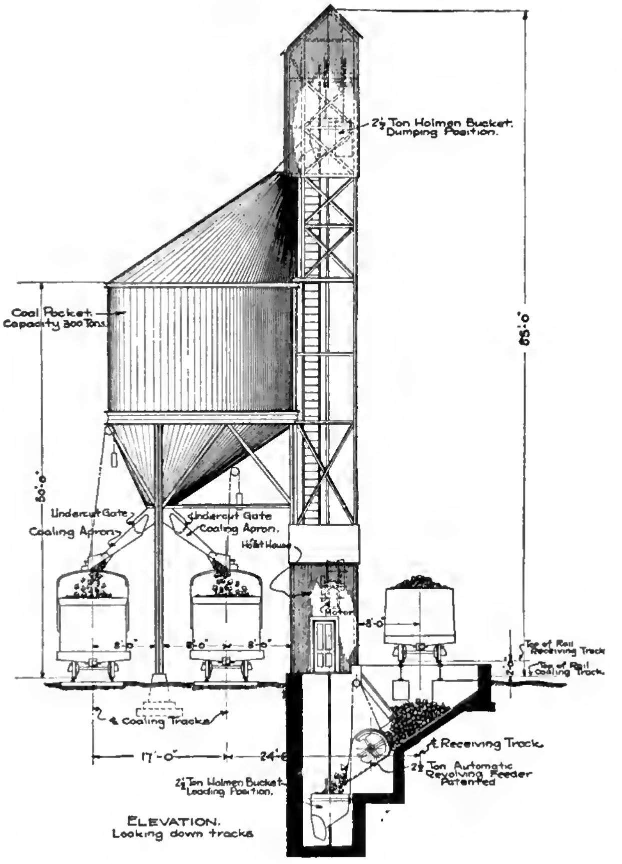


Fig. 1. Diagram of Holmen Coaling Station

24-inch 80-pound I-beam girders. The track hopper is lined with 1 inch of neat cement, sidewalk finish, with well-rounded valleys to the measuring feeder. This method of lining track hoppers has proven more satisfactory than wooden planks covered with steel, owing to the planks eventually rotting and the steel corroding.

* Contracting Engineer, Roberts & Schaefer Co., Chicago, Ill.

The coal from the receiving hopper flows by gravity into a Barrett revolving measuring feeder, which cuts out 1½ tons of coal and discharges it automatically into the elevating bucket, thus preventing overflowing the bucket and flooding the pit. This ingenious device is so designed that when in the act of discharging a measured quantity of coal into the elevating bucket, it closes up the opening in the track hopper as an undercut

Fig. 2. 300-Ton Coaling Station, Corning, N. Y.

gate. As the bucket is elevated, the feeder automatically rotates into its original position, opening the gate in the receiving hopper, and is then ready to receive another measure of coal. It remains in this position until the returning empty bucket descends and is again filled from the feeder.

The Waukegan plant is designed to elevate 40 tons of coal per hour with a single bucket balanced with counterweight. The bucket elevator is operated by a 15-horsepower, 220-volt, three-phase, 60-cycle, General Electric, alternating-current motor, equipped with solenoid brake, circuit breaker, special controller, with Hatch limit switch, a device to cut off the current and stop hoisting, provided the bucket is carried beyond the discharge point by a careless operator. This device makes the station "fool-proof." The motor is direct-connected to a reversible hoisting drum.

The round tank pocket which holds the coal is made of $\frac{5}{16}$-inch and ¼-inch steel plate, properly reinforced with a plate girder at the base to prevent distortion. The balance of the station is of structural steel, the columns being built-up angle and plate construction.

The hoist house and tower are covered with corrugated steel. The entire structure rests on concrete foundations and the bucket pit is waterproofed.

The coaling station at Corning, N. Y., shown in Fig. 2, is a good illustration of the use of reinforced concrete for structures of this kind. The foundation and entire superstructure is of concrete properly reinforced with steel, and is, therefore, of fireproof and permanent construction. The storage pockets are arranged to coal locomotives on two tracks beneath the pockets and on one track outside the pocket. The receiving track and receiving hopper are on the right side of the station in Fig. 2, enclosed with a concrete canopy roof to protect the men from the weather when unloading coal.

The storage capacity of the pocket is 300 tons of coal besides containing two dry-sand bins of 190 cubic feet each.

The receiving hopper is 40 feet long and is arranged to supply coal to two independent hoisting outfits, each consisting of a pair of Holmen balanced bucket elevators, together with a pair of automatic Barrett feeders. The hoisting capacity of each pair of elevator buckets is 125 tons per hour. The balanced buckets are connected by cable to the reversible hoists in the engine room and with the automatic revolving feeders

Fig. 3. 500-Ton Coaling Station, Delmar, Ind.

constitute practically an automatic hoisting apparatus. The driving power of each pair of buckets consists of a 20-horsepower motor. The coal is elevated about 55 feet above the railroad ties, automatically dumped, and lowered on curved chutes into the coal pockets.

The station illustrated in Fig. 3 is a 500-ton Holmen plant, built at Delmar, Ind., for the C. C. C. & St. L. Ry., by the Roberts & Shaefer Co., engineers and contractors, under the supervision of O. E. Selby, engineer of bridges and buildings of the railway company, at Cincinnati.

The station at Delmar consists of a frame superstructure with steel bin trusses supporting a 500-ton storage bin, which is elevated to a sufficient height to allow the coal to flow by gravity into the tenders, this being accomplished by the installation of standard undercut gates with hooded aprons over the two main line tracks.

At the right of the bucket tower or on the side of the structure is located a 36-foot concrete receiving hopper under the unloading track. The coal from the track hopper passes into two Barrett measuring feeders, which load two 1½-ton Holmen buckets. It is not necessary for the operator to watch the loading of the buckets, as this is accomplished mechanically. The increased efficiency of this measuring feeder is such as to bar from serious consideration a balanced bucket plant using the hand-operated slide gate. The two Holmen buckets are elevated alternately, that is, when one bucket is in the pit for the purpose of being filled, the other is at the top discharging its load of coal into the storage bin.

Each bucket is attached to a ⅞-inch crucible steel wire rope connected to a reversible steam hoist. The head-sheaves are of liberal size to insure long life and easy movement of the cable.

The buckets elevating the coal are opened at the top automatically at the proper elevation to deliver the coal to the pocket with a minimum amount of breakage and are also closed automatically after being emptied. As the bucket descends, it is immediately filled by the revolving feeder in the pit.

The operation of filling buckets, hoisting coal, and delivering it to the storage bin is under the control of one man, who merely operates the lever on the hoisting engine to put in operation the feeders and elevating machinery.

The average time required to fill and elevate a bucket of coal is usually about 1 minute and with two 2½-ton Holmen buckets an elevating capacity of 125 tons per hour can be readily attained. This machinery is also designed to elevate 40, 60, or 75 tons per hour, according to the requirements of the station, but the plant built at Delmar was designed with a 60-ton elevator.

Fig. 4 shows another coaling station recently constructed on line of the B. & O. R. R.

POWER GAS AND ITS DEVELOPMENT

The introduction of power gas has made considerable economy possible in the use of fuel resources. Blast-furnace and coke-oven gas, previously wasted, can be utilized and the replacement of steam power by gas power would diminish the annual consumption of coal for power by 9,000,000 tons, or 20 per cent., in England. Power gas can be generated from peat or other poor fuels not suitable for raising steam. There are 140,000,000 acres of peat bog in Europe. For small powers up to 30 horsepower, the use of coal gas is recommended rather than a gas plant; from 30 to 250 horsepower, a suction gas plant is recommended; and for greater power, a pressure gas plant. Recent developments are the use of the engine exhaust in place of steam in the producer, and the abolition of the gas holder in pressure gas plants, the production being controlled by fans driven by the engine.—*Chemical Trade Journal.*

POWDER-THAWING DEVICE

This is constructed by boring two 1½-inch holes in the side of a barrel, 6 inches from each end, in the same vertical line, fitting a 4- or 5-foot length of pipe into each of these holes and connecting the outer ends of the pipes with another length. A grating or rack is fixed in the barrel, on which a

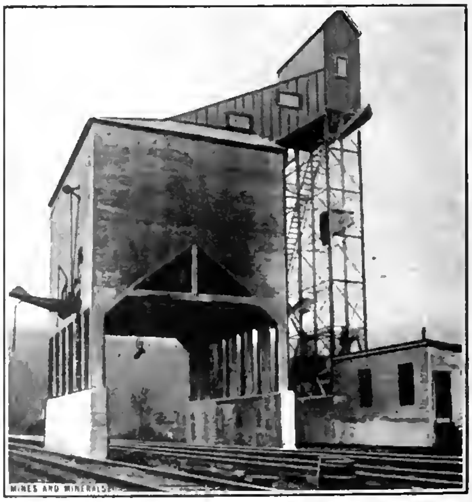


Fig. 4. Coaling Station on B. & O. R. R.

5-gallon oil can is placed containing the powder to be thawed. The open top of the can is just below the upper rim of the barrel. The barrel is filled with water to just above the upper pipe level and a fire lighted beneath the lower pipe near where the vertical riser connects it with the upper horizontal pipe, so that the heated water circulates. The powder in the can is said to thaw gradually and safely. The barrel and can are fitted with separate covers. The can should be cleaned after each operation, to remove any traces of exuded nitroglycerine. *Engineering Record.*

NOTES ON THE DELAGUA, COLO., EXPLOSION

Written for Mines and Minerals, by Geo. F. Duck, E. M.

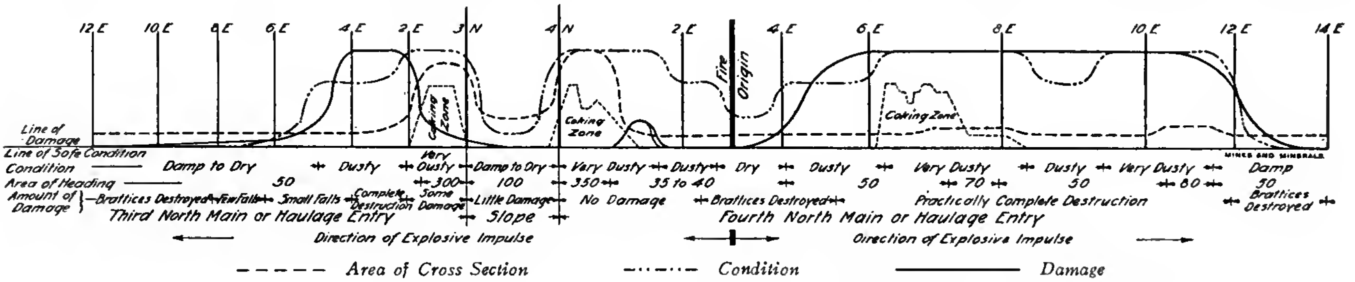
The Influence of Varying Areas and Conditions of Airways Upon the Propagation of Explosions

The explosion of the afternoon of November 8, 1910, in the No. 3 mine at Delagua, Colo., affords a striking illustration of the relation existing between the place and amount of damage done to the workings on the one hand and the condition and area of the entries concerned on the other.

In the general discussion of this accident in the January number of Mines and Minerals attention was called to the fact that the effects of the explosive wave were confined solely to the haulage roads and to but a portion of them. This limitation in the extent of the damage was due to two principal causes. One was the extremely wet condition of the entries as soon as a point was reached beyond which the abnormally dry entering air no longer absorbed the moisture artificially supplied to the mine. The explosive wave met a wet zone and stopped for want of fuel. The other, and fully as an important factor in limiting the spread of the explosion is found in that parallel entries were connected only at long intervals and not, as is usually the case, every few feet by connecting rooms driven in opposite directions. Each main heading and its cross-entries formed a panel in no way connected with the workings of an adjoining panel and an explosion originating in one block could not, as is often the case, be propagated through dusty rooms to an adjoining block. For these reasons the explosion was a simple one, and being simple, was easy to trace from its origin at the fire to the extreme limits of its spread, and this simplicity serves clearly to explain a few interesting facts concerned with dust explosions in general.

In tabulating some 300 or more observations made immediately after the accident, it became apparent, as stated at the outset, that at this mine, at least, there was a very clearly defined relationship existing between the size and dryness of the headings and the amount and location of the damage done thereto. An attempt has been made to show this relationship graphically in the accompanying drawing which is a longitudinal vertical section of the mine from the fourteenth east cross-entry off the fourth north main entry, outby the same to the main slope, up it to the third north main entry and in this to the twelfth east cross-entry. This covers all the mine where any damage was done except those portions of the main slope lying below the fourth north and above the third north main entries, respectively.

The figures and letters 14E, 12E, etc., refer to the fourteenth east, twelfth east, etc., cross-entries, and 4N and 3N indicate the place of intersection of the fourth north and third north main entries, respectively, with the main slope. The first line of the inscription below the drawing gives the condition of the roadways as observed 42 hours after the accident; the second line, the cross-section of the heading in square feet; the third line, the amount of damage done; the fourth line, the section of the mine concerned and the fifth line, the direction of the explosive wave from its point of origin at the fire.



Longitudinal Section of Delagua No. 3 Mine from 14th East Entry Off the Fourth North Entry Outby to Main Slope and Up Same to Third North Entry and Inby Same to 12th East Entry, Covering Section Damaged by Explosion

Area.—The line shown thus - - - - - indicates the change in area of the headings from point to point and is drawn to scale. It will be noted that the prevailing cross-section is about 50 square feet increasing to 70 and 80 feet at two points on the fourth north entry, at what are known as the seventh east and tenth east partings, respectively. At the intersection of the two main entries with the slope the area increased to 350 square feet at the fourth north and to 300 square feet at the third north entries owing to the network of sidings and to the fact that expansion was possible both up and down the slope.

Condition.—As stated in the January number the condition of the headings so far as dryness was concerned might be defined as very dusty, dusty, dry, damp, wet and watery in descending order from the most dangerous. It was also noted in the article mentioned that the road beds in any of the first three conditions could be classed as dangerous as the dust in them could readily be gathered up by the advancing wave and thus propagate an explosion. On the other hand a road bed in any of the last three states (from damp to watery) was essentially safe so far as furnishing dust for extending an explosion already started is concerned. If a dry condition be given a weight of one, a dusty condition a weight of two and a very dusty condition a weight of three, it is possible to draw what may be called a "condition curve" for the mine. This is shown by the line _.._.._... It is to be noted that all "safe" conditions, from damp to watery, are contained in the base line.

In justice to the Delagua mine it must be stated that the observations for condition were made from 42 to 72 hours after the accident during which time the entries had been subjected to an abnormally dry air-current. It seems probable, therefore, under usual working conditions that on the fourth north entry (for illustration) the improvement from very dusty at the slope to damp, that is to say, from highly dangerous to safe conditions, would have been reached within a much less space than was found to be the case such a length of time after the accident.

Damage.—The curve of damage or destruction as indicative of force, shown as a full line, is extremely difficult to determine, owing to the practical impossibility of finding a basis for comparison. Does it require more or less force to destroy a brattice than to knock down a set of timbers, to throw a small electric pump 3 feet than to shatter a pine door and scatter its fragments for nearly 200 feet along the entry? There appears no answer to these questions. On the fourth north entry inby the fire, the gradual increase in the amount of force as measured by the damage done could easily be traced, but outby the same it was more difficult. It was finally decided to consider the destruction as having reached a maximum when at least 90 per cent. of the timbers were destroyed and to grade the relative destruction from point to point accordingly.

Coke.—The zones of coking action are shown thus - - - - - and an attempt has been made to indicate the variation in amount of coking in each zone by fluctuations in the line deter-

mining it. One thing is apparent. Coking action is dependent solely upon the condition of the mine and is not affected by the area of the heading and bears no relation to the amount of damage done. On the fourth north entry coke was found at a place of maximum damage and also at the slope where the mine was practically unharmed. On the fourth north entry at the coking zone the area of the headings was normal, and at the slope the area of the zones was six or seven times as great as at the former place. In all cases coke was found where the mine was "very dusty," indicating that similar coking action might, if looked for, have been encountered at the tenth east entry on the fourth north entry.

The Explosive Wave Inby the Fire on the Fourth North Entry. An inspection of the drawing brings out the fact that the action of the explosive wave differed greatly inby and outby the fire and that this difference in action was due almost entirely to changes in the area of the cross-section of the workings. Inby the fire, except for a slight enlargement at the seventh east and tenth east partings, the headings maintained a practically uniform area of 50 square feet. Hence the "area curve" is essentially straight and parallel to the base line. The "condition curve," owing to the increasing dustiness of the workings, soon reached its maximum about the sixth east entry, which maximum was maintained with but one slight fall until near the twelfth east entry when, owing to the dampness of the workings, it took a sudden drop toward the base or line of "safe conditions." The "curve of damage" is also remarkably uniform. Starting from zero at the fire, this curve gradually reaches its maximum at about the sixth east entry, and maintains this until near the twelfth east entry when it also makes a sudden drop to the base, or line of no damage. It will be noted that in this portion of the affected area, the curves of condition and damage practically coincide.

The initial impulse to the explosive wave was probably given by the breaking down of the burning door, the jar consequent upon the sudden short-circuiting of the entire air-current throwing large volumes of finely divided dust into suspension. Ignited by the fire and drawn in by the momentum of the air, the dust began to distill gases until a point of explosive violence was reached at or about the sixth east entry. Owing to the dusty condition of the workings the explosion was propagated to nearly the twelfth east entry, when it stopped for want of fuel in the damp and wet headings. It is believed that, from the above, the following conclusion is warranted:

In straight headings of uniform area the violence of and consequent amount of damage done by a dust explosion is directly proportional to the condition of the workings.

The Explosive Wave Outby the Fire on the Fourth North Heading, the Main Slope and the Third North Heading.—It is apparent that the damage and condition curves for this portion of the workings are not related in the same manner as those in the section just considered. On the other hand if the first two named curves are studied conjointly with that representing the area, a relationship is at once apparent.

Starting at the fire as before, the explosive wave gained in force until at the "rock tunnel" it had sufficient energy to throw several mine cars full of rock and dirt into that heading. This is shown by the mound-like appearance of the curve of damage at a point about midway between the second east entry and the slope. Had the cross-section of the entry remained constant as it did inby the fire a point of maximum destruction would soon have been reached, and the curves of condition and of damage would have nearly coincided as before. But just after leaving the rock tunnel a network of sidings is encountered and soon thereafter a slope of large section, both up and down which the explosive wave had an opportunity to expand. At this point, then, the area of the workings suddenly increased from less than 50 square feet to 350 square feet or more, or in a ratio of not less than one to seven. Although the dust conditions were favorable to a most violent explosion and consequent great damage to the mine, this detonation either did not occur or had its force and effect practically nullified by reason of this opportunity for expansion.

Having split at the mouth of the fourth north entry a portion of the wave passed up the slope doing but very little damage in this enlarged and damp portion of the mine. At the mouth of the third north entry the wave again split, a portion continuing on up the slope to daylight and the remainder turning into the side entry. That portion passing into the third north entry gathered up a large amount of dust from the sidings and would have produced an explosion of extreme violence had it not been for the large area of the workings which permitted expansion in all directions. But just beyond these sidings there was a sudden contraction in area of from 300 square feet to 50 square feet or in the ratio of from six to one and the consequent concentration of force resulted in the total destruction of the entries at this point. Just beyond the fourth east entry the mine soon became dry, then damp, and the explosive wave was rapidly stopped, as on the fourth north entry, from lack of fuel.

The effect of a sudden change in area as influencing the amount of damage done by an explosive wave is also plainly brought out in that section of the slope just below the intersection of the fourth north entry. As stated before, a portion of the wave, which at the intersection of the slope and fourth north entry did no damage, went inward down the slope. Passing suddenly from a cross-section of 100 or more square feet to one of but 50 square feet, the concentration of force thus brought about wrecked the mine and was only stopped by a wet zone. It is believed that what has gone before warrants the conclusion that:

Under uniform mine conditions, whether of dryness or dampness, the force of and consequent amount of damage done by a dust explosion is inversely proportional to the area of the headings in the affected section of the mine. That is to say, when workings are uniformly dusty throughout, the damage done to the mine increases as the area of the heading decreases and vice versa.

Some other points affecting dust explosions were well exemplified by the Delagua accident, which in addition to the two named before may be summarized.

As it requires a certain length of time to heat dust to a point where gases begin to distill and during this interval the air-current is constantly in motion, the maximum force of a dust explosion is rarely displayed at its point of first origin.

Temperature alone is not a guide as to whether an entering air-current will or will not absorb moisture from the workings. The absolute amount of water in the air-current compared with that required to saturate it at the prevailing mine temperature is the only criterion. Consequently, while explosions of coal dust usually occur in winter because then the air for long periods is generally dryer than in summer, yet they may happen at any season of the year when (no increased watering being done underground) the air enters with a less total content of water than is demanded to saturate it at the mine temperature.

Explosions generally travel outby along the haulage road for two reasons. The haulage road is naturally more dusty than the return; and the amount and fineness of the dust, and hence the ease with which it can be thrown into the air-current, increase toward the intake. This increase in dustiness is due to the fact that the amount of coal hauled as well as the number of men and mules traveling per unit of distance increases as we approach the drift mouth, as well as to the further fact that the entering air is always dryest nearest the point of intake. There is thus a double reason for the amount and fineness of the dust increasing outby, and an explosion which feeds on dust must of necessity travel outby against the entering air-current for its food.

Coking action found is at or very near points of maximum dustiness, and its intensity does not necessarily indicate great explosive energy, merely great heat.

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

Hoisting Engineers' Examinations Held at Des Moines, Iowa, October 12

QUES. 2.—How much farther does the crankpin of an engine travel in one revolution (than the piston) when the length of stroke is 22 inches?

ANS.—The piston makes two strokes in one revolution and travels $2\times 22=44$ inches. The diameter of the crank-circle is equal to the length of the piston stroke, or 22 inches; and the circumference of this circle, or the distance the crankpin travels in one revolution is $3.1416\times 22=69.1152$ inches, which is $69.1152-44=25.1152$ inches farther than the travel of the piston in the same time.

QUES. 3.—Would you consider it best to suspend a boiler or to rest it on the walls that form the furnace?

ANS.—It is often preferable to suspend an ordinary flue or return tubular boiler rather than to allow it to rest on the furnace walls; because the walls are then relieved of the racking movement due to the alternate expansion and contraction of the boiler, which occurs to some extent even when cast-iron rollers are provided to allow for such movement. Also, there is less danger of the boiler being injured by unequal settlement of the walls.

QUES. 4.—When is a large boiler better than a small one?

ANS.—The size of a boiler is always to be determined by the work it has to perform, and due regard must be had for the future requirements. In the early development of a mine it is better to put in two or more boilers of sufficient capacity to meet every requirement for some time to come, and allow for the temporary cutting out of one of these for the purpose of repairing or cleaning. When the demand for power is variable it is better to use two or more small boilers than a single large boiler, so as to permit of cutting out some of them when the demand for steam is light.

QUES. 5.—In setting a hoisting engine where should the foundation be the heaviest? Where is the cast-iron bed of a hoisting engine most liable to crack?

ANS.—In mining practice employing a horizontal engine direct-connected or geared to a winding drum, the heaviest foundations are required under the winding drum, which must be securely anchored thereto by anchor bolts built into the foundation or made fast to anchor plates beneath the masonry. The cast-iron bed frame if improperly designed or if unequal settlement occurs may crack on the piston side of the pillow-block or journal of the crank-shaft.

QUES. 6.—Should the cogs of an engine seem to hammer what would be your conclusions?

ANS.—Hammering of the cogs of a second-motion or geared engine is often due to play between the gears, caused either by mismating, improper design, or undue wear of the cogs. It may also occur by reason of change of load or sudden application of the brake to the winding drum. A semifluid, tarry dope, manufactured specially for the purpose has been applied with good results to reduce the rattle of gears.

QUES. 7.—What size of cylinder would be required on a direct-connected hoisting engine capable of hoisting a load of 5,000 pounds from a depth of 300 feet in 11 seconds, allowing for acceleration and retardation in starting and stopping, respectively?

ANS.—Allowing, say 2 seconds for acceleration in starting and 2 seconds more for retardation in stopping, the maximum speed of hoisting would be the same as though the load were to be hoisted the entire distance in $11-\frac{1}{2}(2+2)=9$ seconds. The engine must therefore be capable of hoisting 5,000 pounds, including the weight of the rope, a distance of 300 feet, in 9 seconds, disregarding friction; or adding one-tenth to allow for friction, the load on the rope would be 5,500 pounds. This would require $\frac{5{,}500\times 300}{33{,}000}\times\frac{60}{9}=333+$ effective horsepower. Assuming an efficiency of about 85 per cent. in the engine, the indicated horsepower (I. H. P.) would be $333\div .85=$ say, 400 horsepower. Taking the mean effective pressure in the cylinder as 50 pounds per square inch, and assuming a piston speed of 600 feet per minute, the diameter of steam cylinder that would develop 400 indicated horsepower is

$$d=205\sqrt{\frac{400}{50\times 600}}=23.67, \text{ say } 24 \text{ in.}$$

If the engine makes 120 revolutions a minute the length of stroke is $(600\div 120)12\div 2=30$ inches; and the required size of the cylinder is therefore 24 in. $\times$ 30 in. under the assumed conditions. In hoisting practice it is customary to estimate on a duplex engine each cylinder of which is capable of developing the full power required.

QUES. 8.—In case of accident to one of a pair of engines at a critical moment, how would you proceed to cut out the crippled engine in order to use the other as a single engine?

ANS.—Shut off the steam from the disabled side by means of the stop valve. Then loosen the wedges behind the crankpin brasses and slide back the brasses so as to allow the connecting-rod to be removed from the crankpin and dropped to the floor. The valve rod need not be disconnected.

QUES. 9.—(*a*) Why are steam domes placed on boilers? (*b*) What purpose does the air chamber on a pump serve?

ANS.—(*a*) The chief purpose of the steam dome on a boiler is to obtain dryer steam in the engine by taking the steam from a point in the boiler high above the water where the water will not be entrapped by the steam as it flows into the steam pipe. The dome also increases the steam space of the boiler. (*b*) The air chamber on a pump serves to maintain a more uniformly constant flow of water, by means of the alternate compression and expansion of the air in the air chamber that takes place with each stroke of the pump. The compression of the air confined in the air chamber maintains a certain pressure between the successive strokes of the piston, which keeps the water moving, and avoids the great loss of energy that would otherwise occur from shock due to the water coming to rest between strokes.

QUES. 10.—(*a*) Explain the term suction. (*b*) What is the theoretical height of suction of water? (*c*) What is the height of suction practically?

ANS.—(*a*) Strictly speaking, there is no such thing as suction. The term is used, however, to describe what takes place when a vacuum is formed behind the piston of a pump. The pressure of the outside atmosphere acting on the water forces it into the vacuous space formed by the movement of the piston; and the pump is said to suck or draw the water. (*b*) The theoretical height to which water will thus rise into a perfect vacuum, or the theoretical suction, depends wholly upon the atmospheric pressure. This height in feet is equal to the pressure in pounds per square inch divided by .434; or multiply the height of the barometric reading in inches by the specific gravity of mercury (13.6), and divide the result by 12. Thus, at sea level the normal atmospheric pressure is 14.7 pounds per square foot, corresponding to about 30 inches barometric pressure. The theoretical height of suction, at sea level, is therefore $14.7\div .434=$ say, 34 feet; or $30\times 13.6\div 12=$ say, 34 feet. (*c*) The practical height of suction depends on the efficiency of the pump and pipe system. It may be taken in feet, ordinarily, as, at least, eight-tenths of the barometer reading in inches; thus, at sea level, $.8\times 30=24$ feet. It may be greater or less than this, depending on the pump.

QUES. 11.—How many volts and amperes of current would be required to burn five 108-volt, 2-ampere lamps, in series?

ANS.—When the lamps are connected in series a current of 2 amperes will be sufficient; but the voltage required will be $5\times 108=540$ volts.

QUES. 12.—How many volts and amperes of current will be required for five 108-volt, 2-ampere lamps connected in parallel?

ANS.—When these lamps are connected in parallel a current of $5\times2=10$ amperes will be necessary, while the voltage required is only the voltage of a single lamp, or 108 volts.

QUES. 13.—(*a*) Give the electrical unit of power. (*b*) Give its relation to the horsepower.

ANS.—(*a*) The watt is the electrical unit of power. (*b*) It requires 746 watts to equal 1 horsepower.

QUES. 14.—Find the indicated horsepower of an engine having a piston 12 inches in diameter and a stroke of 15 inches when the crank makes 140 revolutions per minute and the mean effective steam pressure is 50 pounds per square inch.

ANS.—$\text{I. H. P.}=\frac{50(.7854\times12^2)15\times2(140)}{33{,}000\times12}=\text{say, } 60 \text{ H. P.}$

QUES. 15.—Name some of the important parts of a hoisting engine.

ANS.—The important parts of the engine proper are the cylinder, piston and rod, cross-head, guides, connecting-rod, crank-shaft, crank-arm and crankpin, valve and valve rod, and eccentric. The answer to the question may also be made to include the winding drum, drum shaft of a geared engine, drum gear and pinions, drum brake, indicator, steam throttle and stop valves, lubricating cups, etc., etc.

QUES. 16.—For what use is a rope made of fine wires better adapted than one made of heavy wires?

ANS.—Lighter wires will stand less wear but are much more flexible than heavier wires. For this reason, ropes made of the lighter wires are better adapted to the purposes of hoisting, where the bending strains are heavy and the wear light; while those made of heavier wires are more often employed in haulage, where the bending strains are less and the wear of the rope greater. Where lighter wires are used a greater number of them are employed. Thus, for hoisting, six-strand, 19-wire ropes are quite generally employed; while six-strand, seven-wire ropes are more often used for haulage.

QUES. 17.—Explain the action of a sight-feed lubricator. Draw a sketch.

ANS.—A sight-feed lubricator is shown in section, in Fig. 1. The oil is contained in the large glass cup *L*, which is filled through the hole in the top, closed by the slide cover *A*. The needle valve *P* is raised or lowered by raising or lowering the cam-lever *C*, and the movement of the valve, which regulates the feed, is adjusted by the nut *D*, held in place by the lock-spring *E*. The glass cylinder *R* permits the oil to be seen as it drops through the small opening, shown as closed by the needle valve *P*. The cam-lever *C* is in its closed position; when this lever is thrown straight up the valve is open.

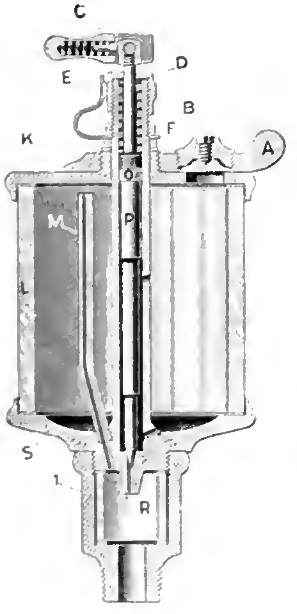


FIG. 1

QUES. 18.—What is the principle on which the injector depends?

ANS.—An injector is a device by means of which a jet of steam issuing from a nozzle, under high pressure, is brought into contact with water that enters a vacuum chamber, under atmospheric pressure. The steam imparts its velocity to the water by which at the same time it is condensed, forming a jet of water that forces open a check-valve by reason of its great momentum, and enters the boiler.

QUES. 19.—Explain the action of a pulsometer pump.

ANS.—This question is Ques. 18 of the mine foremen's examination and is fully answered as Ques. 26, page 541, MINES AND MINERALS, April, 1911.

QUES. 20.—What is the horsepower of a boiler 60 inches in diameter, 18 feet long, with fifty-seven 4-inch flues?

ANS.—The circumference of a 4-inch flue is $3.1416\times4=12.5664$ inches. Multiplying this by 57, the number of flues, and adding to that result the semicircumference of the boiler in inches (assuming one-half of the boiler shell is exposed to the flame); then dividing by 12 and multiplying the last result by the length of the boiler in feet, gives

$$\frac{57\times12.5664+3.1416\times60}{12}\times18=1{,}357 \text{ sq. ft.}$$

To this must be added, say one-half the two end areas of the boiler, less $2\times57=114$ times the sectional area of a 4-inch flue. Thus, $.7854\times60^2-114\ (.7854\times4^2)=.7854\times1776=$ say, 1,395 square inches, or 9.68 square feet. The total heating surface is therefore, say, 1,367 square feet. For a horizontal tubular boiler the rated horsepower is estimated from its heating surface as varying from 14 to 18 square feet of heating surface per horsepower. Taking the average rating of 16 square feet per horsepower, the power of this boiler is $1{,}367\div16=85$ H. P.

NEW INVENTIONS

Complete specifications and drawings for any of the following patents can be obtained from the COMMISSIONER OF PATENTS, WASHINGTON, D. C., at the following rates.

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
An entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED MARCH 7 TO MARCH 28, 1911, INCLUSIVE

No. 985,909. Coking and gas-generating oven, Heinrich Koppers, Essen-on-the-Ruhr, Germany.

No. 986,389 Concentrating device for mineral ores, Herbert T. Herr, Denver, Colo.

No. 986,271. Process of refining and agglomerating ores and the like, Hugo Dicke, Frankfort-on-the-Main, Germany.

No. 986,813. Ore-cooling apparatus, James Belcher Etherington, Winthrop, Mass.

No. 986,608. Roller ore-crusher, Lamartine C. Trent, Reno, Nev.

No. 986,709. Furnace for roasting ores, Nicholas L. Heinze, La Salle, Ill.

No. 987,642. Coal-washing jigger, Thomas H. O'Brien, Dawson, N. Mex.

No. 987,179. Jigger, Wilhelm Seltner, Schlan, Austria-Hungary.

No. 987,209. Ore-concentrator, Henry E. Wood, Denver, Colo.

No. 987,428. Apparatus for the treatment of ores and for the electrolytic deposition of gold and silver and other metals from solutions containing said metals, Frederick Capel Brown, Komata, New Zealand.

No. 987,156. Treatment of sulphide ores, James A. McLarty, Toronto, Ontario, Can.

No. 987,941. Coke oven, Louis Bansart, Johmont, near Haine, St. Pierre, Belgium.

No. 987,993. Coke oven, William J. Kearns, Dunbar Township, Fayette County, Pa.

No. 988,029. Hammer drill, William Prellwitz, Easton, Pa.

No. 987,744. Miner's lamp, Paul Rennert, Hagen, Germany.

No. 987,866. Ore concentrator, Henry Earle, Denver, Colo.

No. 987,850. Process of treating ores, Isaac A. Braddock, Haddonfield, N. J.

No. 987,832. Pipe Fishing Tool, E. Seibert, Didsbury, Alberta, Canada.

No. 988,293. Pulverizing Mill, J. R. Moffitt, Denver, Colo.

No. 988,230. Producing synthetic sapphires, A. V. L. Verneuil, Paris, France, Assignor to L. Heller & Son, New York, N. Y.

No. 988,255. Screening and Picking Apparatus, Joseph Dodds, Rutherglen, Glasgow, Scotland.

Mines and Minerals

Vol. XXXI—No. 11. SCRANTON, PA.—JUNE, 1911—DENVER, COLO. Price, 25 Cents

BUFFALO-SUSQUEHANNA SAGAMORE MINE

*Written for Mines and Minerals, by R. Dawson Norris Hall**

Surface Plant at a Modern Bituminous Coal Mine, Capacity of Tipple 1000 Tons an Hour

The Sagamore mine, operated by the Buffalo and Susquehanna Coal and Coke Co., lies as shown in Fig. 2, on both sides of the line dividing Indiana and Armstrong counties, Pa., most of the acreage, the town and plant being on the Armstrong County side of that line. The mine is reached by the main line of the B. & S. R. R., that railroad being broken by a gap between Sykes and Juneau, which is temporarily filled by the use of the Buffalo, Rochester & Pittsburg Railroad between these points. The portion of the B. & S. R. R., between Juneau and Sagamore, proceeds by Canoe Creek, Little Mahoning, and Plum Creek, a distance of 21 miles, cutting off a triangular strip of Indiana County in its northwest angle.

The only available way of reaching the coal by drifting exhibited itself on Little Plum Creek, the coal around the tipple running everywhere in an unbroken field from the center of operation. It was possible therefore and advisable to erect at this point a large plant capable of handling the entire field, for at no other point was it convenient of access.

The method so often advocated, and with so much justice, of small scattered units of lower capacity served by a central power plant and so arranged as to avoid the congestion too often found in a larger plant, did suggest itself to the management. But as the crop on Plum Creek extends barely 9,000 feet, there seemed inadequate room for two complete tipple outfits with all the trackage now found so necessary, above and below the tipple. It was therefore decided to build the two tipples proposed into one structure, side by side. The coal also crops in Big Plum Creek, but though drainage at this point was a trifle more convenient for a portion of the territory, here the plant would have been at the very confines of the field, increase of tonnage would have been slower and long haulages would have resulted with accompanying additional equipment. Moreover an economy in railroad construction was effected, for otherwise an extension of the road to Big Plum Creek would have been necessary. Every care has been taken to avoid congestion by having adequate accommodation and provision for a large capacity. The usual problem presented is between several plants or only one along a river front, or between many shafts or a single shaft sunk to a completely buried seam; and in these cases a number of small plants is preferable to one big one. But all through central and northwestern Indiana County, it is possible to avoid the expense and inconvenience of sinking and working shafts by concentrating at favorable points in the field, and the result is that this section has been exploited in no other manner. Sagamore is only one of many large plants in this section; small operations being seldom found.

The area comprises roughly 10,000 acres of excellent coal. The bed mined is the Upper Freeport and has an average thickness of 4½ feet.

The preparation of an agricultural region, several miles from any railroad, for the operation of mines in their preliminary stage, prior to the arrival of the projected means of transportation, involves the construction of many temporary structures and installations. Almost every large mine has to be so constructed today that the railroad built to remove its coal may earn a dividend as soon as the track is completely laid. Moreover a modern mine requires too lengthy development to admit of the inception of its operations being delayed.

To this end, Sagamore was provided with two portable sawmills and four temporary power houses, each of the latter accommodating two mines and each equipped with two boilers of locomotive type of 50 horsepower each, and one Sullivan air compressor. These compressors were afterwards distributed at other plants of the same company. A large amount of stock coal accumulated between March 22, 1905, the day when ground was broken, and November 29 of the same year, when part of the stock was loaded into cars from a temporary tipple.

Little Plum Creek, having a grade of 26 feet to the mile, like all streams on a gentle grade, meandered freely from side to side of the valley and for half a mile this troublesome stream was provided with a new and more efficient channel to permit

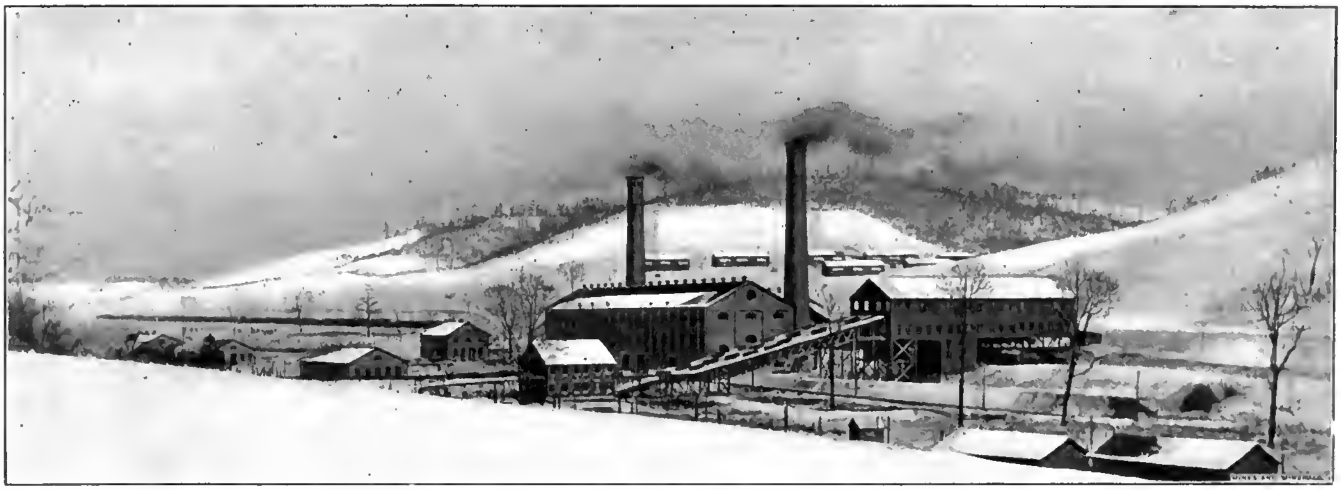


Fig. 1. Sagamore Plant, Buffalo and Susquehanna Coal and Coke Co.

*Mining Engineer, DuBois, Pa

of the erection of permanent buildings and railroad tracks in and around the ground formerly made useless by its irksome ubiquity.

Above the projected power house, a dam confined its waters for use in the boilers, but now the water from what were Artesian wells, but which now require to be pumped, is used in preference to the creek water.

There is nothing much demanding consideration in the underground projection of the Sagamore mines because the roof breaks freely and the cover is not heavy, so no elaborate or expensive treatment of the roof problems was necessary. The roof is generally of sandstone but occasionally slate appears. The bottom is of fireclay. There has never been the slightest premonition of any squeeze or creep. The room and pillar advancing method is everywhere followed. The rooms are made 24 feet wide, the pillars 16, and are, when conditions permit, 300 feet long. All chain pillars in both butt and face headings are 48 feet thick, while the wing pillars flanking the face headings on either side are kept 100 feet wide. The headings are driven 12 feet wide and occasionally the rock from these is loaded into gobbing holes instead of being brought out for disposal. The fact that the operation consists of eight separate mines with wide headings ventilated by eight distinct fans, makes it necessary to provide only a few headings in parallel system to furnish sufficient ventilation for each unit. The three-heading system is that commonly adopted, but four headings in parallel are sometimes used. Despite this fact, a sufficient current of air is furnished with a low velocity of air-current. The fans are all 10'×4' Clifford-Capell, non-reversible blowing fans, belt driven by 100-horsepower motors capable of producing 100,000 cubic feet of air per minute against a 4-inch water gauge.

All the coal is mined by machines; there is no hand-pick work. Some of the men both cut and load their own coal, and it is this fact which reduces the output per machine, which would be very much greater if steady cutting were provided for each machine. There are 72 Sullivan machines and these cut not only the rooms and headings but also all the pillars in the mines.

The track gauge is 3½ feet, and mine cars have a capacity of 2 tons. The cars are hauled by 6 Jeffrey and 6 General Electric locomotives ranging in weight from 12 to 14 tons, and each capable of pulling 100-car trains.

The floor of the mine rolls in gentle undulations, but as an anticlinal makes its appearance at the railroad, there is on the whole an uphill grade against the loads throughout the mine which does not average over 1 per cent. The tipple, shown in Fig. 1, is a large steel structure of the car-haul type, that if run at full capacity is amply competent to handle 1,000 tons per hour, although at present the tonnage does not exceed 4,000 tons per day. The cars are brought up the incline by a moving chain, one by one in a steady stream, weighed and dumped. They then pass on over the dump and reach a kick-back, returning thence to a retarder which lowers them to the empty track. This is elevated above the loaded track, by which the cars approached the tipple, so that the grade favors the haulage plant, both approaching and leaving the foot of the hoisting incline. To handle so large a tonnage as is contemplated, three Phillips cross-over dumps are installed.

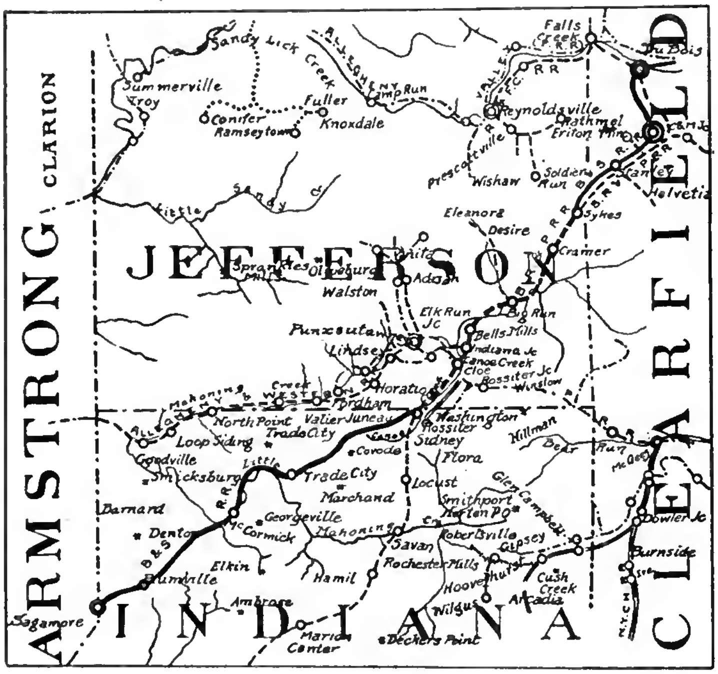


Fig. 2. Map Showing Location of Sagamore

Under the tipple, Fig. 3, are four tracks with two miles of aggregate storage for cars both above and below the dump. A "scissors" cross-over, Fig. 4 (*a*), connects all the empty tracks together conveniently near the tipple so that any car can be shifted to any track with ease and dispatch. This could have been arranged by the construction of a "double ladder" as shown in Fig. 4 (*b*), but there are objections to such an arrangement, as it takes up so much more room, since only one track can be made really straight and therefore all cars but those on that track must round the curves which constitute the ladder and because, with it, the cars have to be kept so far from the tipple The plan which involves the "scissors" cross-over is the only one which provides for placing cars on whatever track is desired without unnecessary congestion, and loss of room, together with minimum resistance to the motion of the cars and minimum risk of derailment. But above all, this method of arrangement saves many unnecessary steps to the car dropper.

The grade on the railroad loading tracks is 2 per cent. for 50 feet above and below the tipple. At this point each car in its transit has to be handled separately and must be stopped and started several times. But with such an abundance of grade there is no danger of the output of the mine being suspended by the stalling of a stiff car. Below the tipple, the grades run from 1½ per cent. down to 1¼ per cent., but above, where empty cars have to be handled, the grade is 1.6 per cent.

To avoid breakage of coal, the tipple was provided from the first with shaking chutes. This reduced both the height of the structure and the destructive momentum due to the well-known impossibility of regulating the fall so as to suit coal of every size and shape, subject to differing frictional retardations due to its location in the pans and to changes of temperature and weather.

A new method of saving coal from disintegration whilst being loaded is being installed at the tipple in accordance with the designs prepared in the office of the chief engineer. The purpose is to forward and lower the pan when the coal is low in the car and draw it back and raise it when the section of the car being filled is nearly full. The pan dumps lengthwise of the tracks. A basket could be used to lower the coal but that device has to be lowered and raised for every mine car unloaded. On the other hand this pan permits a continuous flow of coal even when the elevation and position of the point of delivery is being adjusted.

The first idea was to regulate the point of discharge by a simple elevation of the end of the pan to permit of the passage of the highest steel gondolas under it. But such a pan creates difficulty inasmuch as raising one end of the pan and leaving the other stationary causes the slope of the chute to be so reduced that the coal will cease to flow. It was accordingly

decided to design a pan which could be raised at the will of the car trimmer and which nevertheless would retain the same or approximately the same pitch. This was accomplished by providing a track on which the pan could travel and installing a long air cylinder operated by compressed air to furnish the power to move the pan back and forth whilst the end was supported by a hanger of unvarying length.

The power house is a massive piece of construction as the illustration shows; the engine and boiler rooms are set parallel

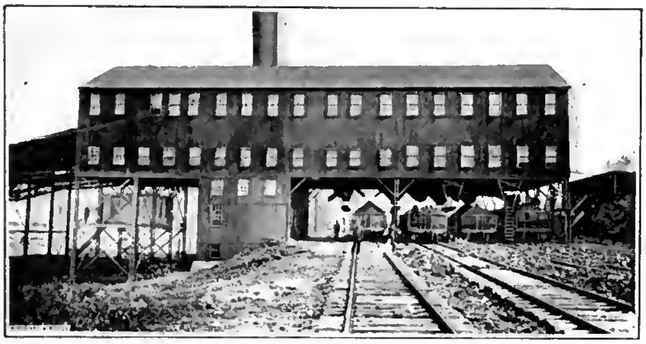


FIG. 3. LOADING TRACKS AND TIPPLE, SAGAMORE

to each other but side by side, separated for cleanliness and fire isolation by a party wall 36 feet high, the full height of the building. The power house is provided throughout with a concrete floor reinforced with iron, the lower part or basement being used for the piping and for the delivery of ashes from the boilers. The roof is supported by steel trusses and covered with slate. The engine room measures inside 67 ft. × 324 ft., and the boiler room 39 ft. × 260 ft. In the latter, as shown in Fig. 5, are five batteries of Rust vertical water-tube boilers of 350 horsepower each, making 3,500 horsepower in all. These are fired by Murphy automatic stokers. The fuel is supplied by a belt conveyer from the tipple; this runs from a bin under the screen to an overhead storage bin and the coal is distributed by a traveling automatic tripper. One man tends to the fires, dumping the ashes into hoppers beneath the furnaces whilst another man takes care of the water supply of the boilers. These two men do all the work of the steam raising. Judging by other plants, at least 8 men are saved by the use of automatic stokers, which is a considerable saving when the cost of a year's stoking is considered. This saving would not be so apparent were the plant of less capacity and it must also be remembered that by using this automatic feed, the boilers can be made to give satisfaction using the finest and most unmerchantable slack.

The ashes from the boilers, dropped into concrete hoppers, are sprinkled with water and hauled away from the basement in special ash cars by an electric locomotive and dumped from a staging into railroad hopper cars. To effect this, a special form of dump has been provided. The trestle will hold several ash cars, and as many as desired are pushed by the motor up an easy incline to the top of the trestle. They are permitted one by one to drop back to the head of the incline. There, singly, they are stopped on a revolving dumping table, turned a quarter circle and dumped into a railroad gondola or hopper standing below. The tilting of the dump is performed by a small cylinder pivoted at the lower end. Thus speedily and cheaply the ashes are removed from the plant and delivered to a place where they can be made to fill a useful purpose.

The water for the boilers is supplied from six wells and is pumped into two Stillwell feedwater heaters. Like all boiler rooms with automatic stokers, it is cleanly in the extreme and it is possible to keep the engine room in perfect condition despite the fact that the two are ranged side by side. The power house is flanked at each end by a brick stack 150 feet high, lined with firebrick and of 12-foot inside diameter, supported on concrete foundations. An arched concrete underground flue connects the boilers with the stacks.

In the engine room are three C. & G. Cooper cross-compound engines with 20- and 34-inch cylinders and 42-inch stroke, making 100 revolutions per minute, directly connected to Crocker-Wheeler dynamos of 500 kilowatt capacity generating a direct current of 500 volts. The Ridgway Dynamo and Engine Co. supplied the dynamo for furnishing light. It has a capacity of 100 kilowatts and furnishes an alternating current of 1,100 volts, being a 60-cycle generator coupled directly to a McEwen engine.

There are also four Allis-Chalmers cross-compound air compressors. The high-pressure steam cylinders have a diameter of 26 inches, the air cylinders are of 26-inch and 44-inch diameter, respectively, and the stroke is 48 inches. Each compressor produces 5,000 cubic feet of air per minute at a pressure of 100 pounds to the square inch with the horsepower rating of 1,000.

Along the full length of the engine room and hung 26 feet above the floor, travels a standard erecting and dismantling electric crane of 22 tons capacity, manufactured by the Northern Engineering Co.

A standard gauge railroad track, connected with the B. & S. R. R., enters one end of the engine room, and space is provided for one car to stand inside the power house for convenience in loading any injured machinery or for unloading any additions to the equipment.

All piping in the engine room is beneath the concrete floor. The pipes from the header over the boilers drop through the floor and are passed through the concrete wall of the engine room basement, holes being provided large enough to permit of all possible side motion resulting from changes of temperature in the steam lines. This provision keeps the engine room clear, with the head room entirely unobstructed. In both sides of the power house, there is abundance of room. On the boiler side, there is accommodation for four more batteries of boilers, and on the engine side for another large compressor and two generators, the size of those already installed.

Four receivers stationed outside the power house receive the air from the four compressors. Air leaves these at 80 pounds per square inch pressure and is conveyed by pipe 14 inches in diameter, branching eastward and westward toward the mines. This dimension is reduced at each mine opening, the pipes off-bearing to the drifts being uniformly 8 inches in diameter.

The locomotive barn, car-repair shop, and machine shops, are all one story brick buildings, 13 feet high with frame roof trusses, covered with slate, and foundations of concrete. The first measures 40 ft. × 102 ft., and the other two 40 ft. × 100 ft.

The locomotive barn has in front one pair of double-swinging doors for each track, separated by door posts only.

There is room for 24 motors though only 12 are now in use. Twelve separate tracks with 7-foot centers enter the building

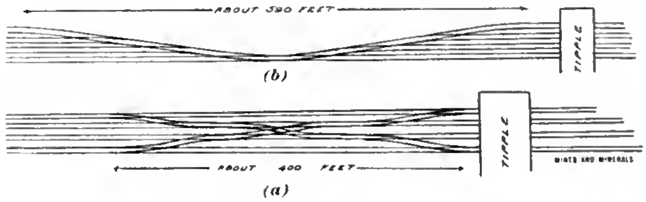


FIG. 4. "SCISSORS" AND DOUBLE-LADDER CROSS-OVERS

and these being of 80-pound rails need only occasional support, which is afforded by 12″ × 12″ brick piers set 8 feet apart and rising 4 feet above the basement floor on which they stand. The tracks are open and the space between the tracks floored, as shown in Fig. 6. The building is heated by steam and lighted by electricity.

The importance of an adequate and well-heated locomotive barn is considerable. In the first place it provides a place for

a close inspection of the locomotive after the day's work is done. Resting as the locomotives do in this Sagamore barn on heavy rails supported by pillars, not by walls, it is possible to travel all around, under as well as over them, tightening loose bolts and making complete inspection. For this inspection portable lights having long leading wires are provided. A locomotive should not be permitted to become unduly chilled for when it enters the saturated air of a drift mine, it immediately becomes

Fig. 5. Boiler Plant, Sagamore

covered with condensed moisture with bad results. The difficulties encountered by the electrical firms in providing adequate insulation have been not a little increased by a lack of consideration on the part of the mine owners as to the need to such machinery.

Fourteen feet of the east end of the barn is partitioned off and used as a repair shop. An 8-inch I beam running lengthwise of this shop carries a small traveling chain hoist for use in dismantling locomotives. A concreted pit under the track serves to remove worn wheels from the locomotive and to provide for the emplacement of new wheels. For this purpose the locomotive is jacked up, the rails over the pit removed, the axle and wheels lowered to a small truck beneath and hauled outside.

The car-repair shop has two mine tracks running lengthwise of the building. In addition to the usual power grindstone and emery wheel, an automatic cut-off track saw and a large drop hammer are installed. These tools are all operated by an electric motor. Special attention may be drawn to the fact that the hammer, in addition to its most usual uses, shapes all the machine picks used at this plant. By the use of special dies, bar steel is hammered out in the rough, a very little hand-shaping followed by tempering resulting in a finished pick. All the general blacksmithing is done in this building, and five concrete forges are provided. One large forge in the center of the building is used whenever a large open space around the fire is desirable. The other four are set near the walls and these are in continuous use. The floor of this shop is of cinders and the lighting is by electricity.

The machine shop has a concrete floor with $2'' \times 4''$ sills laid flush with the top and over these are nailed $\frac{7}{8}$-inch tongued and grooved yellow pine flooring. The mine track enters one end and runs through the building. The equipment consists of an 8-inch pipe machine, a large drill press, a planer and a shaper, all operated by an electric motor.

The supply house is most elaborate. Supplies are not allowed to go to waste or to be lost sight of, so as to be unavailable when required, nor is there any looseness and uncertainty as to the necessary provision in case of a breakdown. Where no storehouse control is maintained there is always a large loss in keeping small duplicate parts on hand. The need for them is felt only at long intervals, and when it appears the personnel may be changed, the duplicates may be covered up or may have been used in some other place or even with careless management destroyed. So that not only is the delay sustained, but good material, purchased for the occasion, is made unavailable or valueless. All the bins, shelves and racks are carefully stenciled with the size and nature of their contents. The storekeeper not only keeps a vermin and dirt-proof depository, but also an account of material by an accurate card system which records the nature of distribution and the stock on hand. A two-story brick building measuring 100 ft. $\times$ 40 ft. $\times$ 30 ft. houses the supplies. The first floor contains racks and bins, in the center of the building and along the walls, for all the heavier mine supplies. On this floor, the storekeeper has his office and shelves are provided for lighter parts. A mine track, extending 20 feet into the building, provides for the transfer of heavy material.

The bar iron is stored on a large vertical rack within the building. The lower floor is of concrete. A hand-power elevator of half a ton capacity connects the floors. The second floor is arranged with bins and racks mainly for supplies of lighter weight, but the bar iron rack extends through to this upper story. The railroad supply track passes the unloading platform on a level with this floor. Here also electric lights and steam heat are provided. The oil is kept in a separate brick building on the opposite side of the supply track.

The town is laid out in separate sections called wards, on ground sloping in all directions separated by small streams affording good drainage and isolation in case of serious fire. The streets are 40 feet wide and fronts of houses 20 feet back of street line. The building lots are 50 ft. $\times$ 130 ft. with a rear alley 12 feet wide.

The houses are in four styles, built on hollow block tile foundations, studded and covered with double half **V** and siding, underlaid with heavy building paper then lathed and plastered and roofed with slate. The water supply is from three Artesian wells near the creek and pumped through a 6-inch cast-iron pipe to a double concrete reservoir holding 80,000 gallons on

Fig. 6. Locomotive Barn, Sagamore

each side, located at the highest point in the town. This 6-inch pipe serves as a main for the town system with 4-inch cast-iron lines leading from it through each street, set $3\frac{1}{2}$ feet underground. Thus the piping system can be supplied direct from the pumps or by gravity from the reservoir. From the 4-inch lines $\frac{3}{4}$-inch branches are laid to domestic hydrants at the sidewalk, spaced 100 feet apart, and located on alternate sides of the street. At all street intersections and along the streets at

intervals not exceeding 400 feet fire hydrants are placed. The elevation of the reservoir is higher than the highest house. Each ward has an organized fire brigade and is provided with a hose house containing a hose cart and 150 feet of hose.

The public school, located in a natural grove, is large and modern in every respect, containing four class rooms upstairs and four downstairs, all 24 ft. × 30 ft. 6 in. with a cloak room adjoining each class room. A drinking fountain is provided at each floor, and toilet rooms are found in the basement. The building is of brick cased, built on stone foundations. It is roofed with slate and heated by steam. There is a hotel with 31 bedrooms, in keeping with other improvements, a fine brick office (with two office rooms, a drafting room and three bedrooms and a two-story vault) and other conveniences are all interesting parts of this equipment.

Nor in these days, when the care of the injured is a paramount consideration, should reference be omitted to the doctors' office arranged with a waiting room and two offices, one for each physician. There is provided also an operating room with a concrete floor, a central drain for efficient cleansing of the room, every facility for cleanly operating, and a sterilizing cabinet. This departure may be described as a connecting link between first aid and the hospital. Here minor injuries can be adequately handled and professional aid in the case of major injuries can be supplied before the long distressing trip to the distant hospital commences. Sagamore, it must be remembered, is on a different railroad, and 30 miles from the nearest hospital; viz., that founded by Mrs. Adrian Iselin, at Punxsutawney.

The store of the Keystone Store Co. is a large two-story building, measuring 66 ft. × 110 ft.

The main details of this comprehensive plant were laid out under the immediate direction of Chas. D. Oldknow and Jas. Harvey. The former is chief engineer, whose previous affiliation was with the Temple Iron Co., of Scranton. The latter, who has made mining a life study, is the very capable general superintendent. H. A. Molder is the directing head at Sagamore, and J. A. Caseley is the auditor. The steel work and trusses and the mechanical equipment of the tipple and power house were detailed and constructed by the contracting engineers, Heyl and Patterson, of Pittsburg.

COAL NOTES

Steel vs. Wooden Cars.—An English engineer, basing his opinion upon experience, says: "In coal seams lying horizontally, or only slightly inclined, steel mine cars are best, and last a long time without any considerable cost for repairs; however, where the coal seam is inclined, wood cars are preferable, by reason of their necessary frequent renewals or repairs caused by the breakages general in these seams."

British Columbia Labor Troubles.—At this writing the labor troubles in the Northwest are unsettled and operators are preparing for a period of idleness. The railroads are contracting for coal in the United States. Large shipments will be made over the Great Lakes into Canada. The Great Northern is seeking to place contracts for coal in the states of the Northwest. Under these circumstances it will not be long before the operators and miners settle their differences by arbitration.

New Washington Mine Commission.—An act was recently passed creating a commission to investigate the operation of coal mines in the state of Washington; to revise the coal-mining laws of the state and to report their conclusions to the next legislature. Washington has had but one mine inspector, but the last legislature authorized an assistant inspector to be appointed by the governor.

Coal Beds Under Scranton.—In all there are 11 coal beds under Scranton, known by names as follows, beginning with the highest:

No. 1. The 8-foot coal bed. It only exists in two small areas or islands under the highest part of Hyde Park hill. No. 2, 5-foot bed; No. 3, 4-foot bed. These are in the hill top on the West Side only, from Dodge mine to Marvin mine above the level of the Lackawanna River. No. 4, the Diamond bed, No. 5, the Rock bed. These are on the West Side of the Lackawanna River only, under Bellevue, Hyde Park, Providence, and, parts of Keyser Valley. No. 6, the Big or 14-foot bed, No. 7 New County seam. These extend under the whole West Side and also become the surface vein on the East Side at the National Colliery near the south line of the city, also in the central city and hill section nearly to the Moses Taylor hospital. No. 8, Clark bed, No. 9, Dunmore No. 1, No. 10, Dunmore No. 2, No. 11, Dunmore No. 3. The latter four beds are under the whole city from Nay Aug Park to West Mountain.

The average thickness of the 8-foot bed is 7.6 feet; of the 5-foot bed 4.4 feet; of the 4-foot bed 3.6 feet; of the Diamond bed 9.4 feet; of the Rock bed 5.2 feet; of the Big bed 12.2 feet; of the New County seams 6 feet; of the Clark bed 6.7 feet; of Dunmore No. 1, 3 feet; Dunmore No. 2, 4 feet; and of Dunmore No. 3, 3.3 feet. The lower Dunmore at Price-Pancoast, called No. 4 or China bed.

Coke Ovens in Great Britain.—The *Iron and Coal Trades Review*, of London, states that at the close of 1909 the number of beehive coke ovens in Great Britain, was in England 16,346, in Wales 173, and in Scotland 874, making a total of 17,393. As there were at the time 6,789 retort ovens in use, the entire number of ovens of all classes was 24,182.

Natural Gas Supply of Ontario.—The county of Welland and adjoining county of Halimand furnish most of the natural gas in Ontario. There are about 500 producing wells, some of which furnish 1,500,000 cubic feet daily. The depth of the wells ranges from 550 to 3,000 feet. Prices for gas are 15 to 45 cents per 1,000 feet, according to quantity used. Gas is piped to Hamilton, Niagara Falls, Ontario, and as far west as Brantford and Paris. Farmers and country villages are supplied at retail prices, while iron works, smelting works, and other manufactories in Welland are supplied at wholesale rates. It is estimated that over 30,000,000 feet are produced daily in this district.

India's Coal Output.—According to the principal address delivered before the Mining and Geological Society of India with reference to the coal industry it was said that after the serious setback of the last two years there seems every prospect of better times. In 1909, 2,544,610 tons of coal were exported, while during 1910 the shipments rose to 3,167,481 tons. This is the highest figure yet attained, being higher even than that of 1908, when 3,095,748 tons were shipped. Regarding the quality of Indian coal, it is alleged that the fuel from the best seams compares favorably with that produced in Japan, Australia, or South Africa, which are the keenest competitors; and with low costs of production Indian coal should be able to hold its own in the eastern market.

Moistening the Air.—President Kuhn, of the Pittsburg-Westmoreland Coal Co., has devised a simple method for lessening the dangers of explosions, particularly dust explosions, on cold winter days, when the danger from this source is greatest. When the temperature is low the moisture in a mine is quickly absorbed by the ventilating current on account of the dryness of the air, and to overcome this feature Mr. Kuhn has exhaust steam turned into the fan houses at all his mines following a sudden drop of temperature. This keeps the air-currents moist and prevents the mines from becoming filled with dry dust. Other companies in Western Pennsylvania have adopted the plan of late and many mining men are of the opinion that it minimizes the danger of dust explosions.

Erasmus Haworth, State Geologist of Kansas, has issued a special warning to all coal operators and miners in which he says: "Do not presume your mine does not make gas, all coal mines do." An abstract of his "warning" which appears on another page, will be found interesting reading.

THE CARE OF MINE MULES*

Written for Mines and Minerals

Importance of Care in Choosing Animals for Underground Work and of Proper Treatment and Feed

Outside, away from the immediate vicinity of a coal mine, did you ever see a dead mule? No less an authority than Frank Amos, of the Fairmont Coal Co., made the statement "that the average life of a mine mule was 3½ years, and unless conditions were changed to prolong life, the use at the present cost of the animal was unprofitable."

When an animal of this kind lives indefinitely on the farm it seems incredible that his life should be shortened to 3½ years in the mine. There are mules today in anthracite mines that have been working 20 years, yet it is probable that the average life of such animals in all anthracite mines does not exceed 5 years. Most mules come to an untimely end through accidents; however, a large number of accidents can be avoided if judicious choice, proper training and proper care be given the animals. Those animals which attract attention on the surface because of their animation, cavorting, and appearance, are treated kindly, are properly fed, watered, and well groomed, consequently the adage "A horse is what you make it" is as applicable to underground animals as those which live and work above ground.

In purchasing stock for underground haulage, activity, eyesight, feet, temperament, strength, and wind are considerations, but if the animal lacks intelligence he has no place in the mine.

It is natural for the farmer or teamster who has an occasional animal to sell to dwell on its good points, neglecting the bad. The dealer's talk may therefore be eliminated beyond the soundness of the animal, and the acceptance or rejection simmered down to the use of one's judgment. In most instances it is better to deal with those who make a business of furnishing mules to mining companies, and if it is possible, to go to the stock yards and pick out the animals rather than trust to the dealer's judgment. This suggestion is made because the dealer scarcely knows more about the animals than the purchaser, and does not know the conditions under which the animals must work. Again the dealer, anxious to retain the trade of the purchaser, will send a nice looking animal which is probably everything a farm mule should be for soundness and strength, yet lacking in the most important necessary points for mine haulage; namely, activity and brains.

Many animals which act and look right on the surface are most unsatisfactory underground, for which reason the suggestion is made that after the animals are picked out an agreement should be made with the dealer that in case any of them do not act rightly in the mine they may be exchanged. If mine stock can be purchased near the mine it can be trained as soon as received. If, however, it has been subjected to a long car journey, 3 or 4 days should elapse before an attempt is made to take it in the mine. In the meantime the animals should be placed in the paddock to allow them time to get over their transportation legs, rest up, and get acclimated. At the same time they should be examined to see that they have not contracted influenza, pink-eye, or some other disease on the journey.

During the first 2 or 3 days the animals are put to work in the mines they should be in the hands of capable handlers and trainers, because being usually green and soft, they may be readily scared and made mean. It is stated that any habits incurred by mules during this training period are difficult to eradicate, and in almost every instance mules that are properly handled at the beginning of their underground careers are the most tractable. By nagging and abuse it is possible to spoil the temper of an animal as well as a child, and frequently a mule will resent this treatment to the best of his ability; therefore, nearly every coal-mining state will report deaths due to mine mules.

While a mule's heels are not to be trusted, nevertheless the humane driver and his mule become good chums. If, after a 3-day training period the mule does not appear active on his feet and to use judgment, he should be taken from the mine, as he is unsuited to the work and cannot be depended upon to look out for himself when occasion demands. It is believed that if this suggestion is followed many mules' lives will be saved. Next to animal intelligence as a means of saving mule life, is its care. Animals that work in mines should have clean comfortable quarters, with pure water and good wholesome food; nor is this all, they should have their feet and legs washed regularly and their hocks dried and their bodies should be as regularly combed. At large operations it is customary to keep a veterinary, who examines the animals that are well and treats those that become sick. To those who are unable to incur such expense, it may be stated that stock well fed, groomed, and housed, will, if in good health, enter into the spirt of hustling with as much zest as the drivers; but if abused, neglected, or feeling unwell there is danger of their losing interest in the work and becoming injured. A mule that is not feeling well should not be worked, and a sick mule should be taken from the mine. If it be possible the mules should be taken from the mines every Saturday and returned Monday morning before daylight; that is, if it is not the custom to hoist them out every day.

Fig. 1. Underground Stable, Western Pennsylvania

Few mining men not directly connected with large shaft operations realize the magnitude or the initial cost of a modern underground stable. Shaft operations, once confined to the anthracite collieries in the United States, have become so extensive in the bituminous fields that in most cases the old practice of lowering and hoisting the live stock has been abandoned, and the horses and mules are provided with quarters even more comfortable and commodious than those of the live stock that are stabled outside the mines. The underground stable, shown in Fig. 1, is reproduced from a flash-light exposure made 330 feet below the surface in one of the best constructed underground stables in the bituminous coal region of Western Pennsylvania. The stable is 18 feet wide, 12 feet high, and long enough to accommodate 50 head of stock. It is constructed of red brick, and the walls and arched roof are frequently whitewashed. It is built immediately in the 8-foot seam of coal, and is lighted with incandescent lamps, ventilated by a current of fresh air which does not traverse any other portion of the mine, and is

* Literature in Mines and Minerals: "Pit Stock and Management," J. W. Byers, Vol. XXIII, p. 295. "Care and Protection of Mules in Mines; Rules issued by D. & H. Coal Co.," Vol. XXIII, p. 568. "Nutriotone," Vol. XXIII, p. 272. "Bickmore Gall Cure," Vol. XXIV, p. 491. "Care of Mine Mules," Dr. J. C. Newhard, Vol. XXVIII, p. 56. "Care of Mules' Feet," Vol. XXX, p. 435.

supplied with fresh water brought down from the surface through a pipe. It contains a hospital stall for sick or crippled animals. While it is customary to have a sanitarium for mules underground, it is unquestionably better to remove the sick or injured animal to a hospital outside the mine. Disease and wounds always do better where there is fresh air and sunshine.

The stable ventilation should be watched, as the mules are subjected to more or less impure air all day, and while fresh air is good for the animals, nevertheless, they should not be subjected to drafts, particularly after coming from work when they are warm. The stable boss takes pride in keeping his animals in good condition; therefore, it is discouraging when a mine foreman will permit drivers to abuse them. When a mule comes to the stable in the evening showing whip marks and bruises the stable boss should not allow this driver to have another animal until the case has been investigated. There is no reason for abusing an animal with a whip, particularly when pulling at its best, and it is an incompetent foreman that will permit such treatment. It is the duty of the stable boss to report the abuse of the mules, and it is the duty of a mine boss to see the mules once in a while as they go out in the morning and return to the stable at night.

Mule Barn in Ross Seam, Truesdale Mine

According to two authorities the animals should be fed as follows: First, hay, next water, and then grain. Animals should eat hay at least half an hour before being given grain. If the water is given last it washes the food into the intestines before it is acted upon by the gastric juices. If the hay is given after the grain it carries the grain with it, for the hay is principally digested in the intestines, while the grain is acted upon by the stomach for the most part.

Corn is richer in fat than oats; therefore, for strength, feed corn, and for speed, feed oats. For an illustration, race horses are fed oats, and the experienced teamster will favor the feeding of corn.

Dr. I. C. Newhard, Chief Veterinarian of the Philadelphia & Reading Coal and Iron Co., experimented with various feeds and found that two-thirds crushed oats and one-third cracked corn the most reliable. "A handful of coarse ground pure salt should be fed to each mule twice a week." Dr. Frank Amos, who is in West Virginia, suggests a coarse-crushed feed, about two-thirds corn and one-third oats.

Mine stock will consume about 12 pounds per head per day of this feed and about 15 pounds of hay. If a horse or a mule has not cleaned up its former feed the troughs should be cleaned and less put in the next time, until it is ascertained just how much it takes to keep them. The animal should have about all it will eat, but it is better to give not quite enough than too much. Too much grain will cause acute indigestion, paralyze the walls of the stomach, and usually results in death. A stable boss will make a great mistake by feeding too much and allowing food to stand before the animals all the time. While this method will increase flesh for a short period the animals eventually break down through their digestive organs being destroyed.

Grain should not be placed in the animals' troughs ready for them when they come in from work, and it is better for them to be without grain at noontime than to be without water, but by all means give them three feeds a day. Plenty of water will keep the digestive organs in good condition, while large quantities of grain and no water will destroy them.

To give a feed of good bran once a week will aid the conditioning of stock, keep the bowels open, and reduce fever, which is caused by strong grain.

The most common troubles which result from high feeding are heaves and affected wind. Horses sometimes die from the effects of a disease known as azoturia, which is caused by idleness and feeding strong grain. Laminitis is more common in mules from long standing on hard floors for several days and being fed the same amount as when at work. This disease can be avoided by reducing the feed and giving some exercise. Time and attention given to this simple treatment by a stable man is profitable, for it is always the largest and best stock which become affected with this disease, and there are few that recover from it.

So far no veterinarian has recommended the use of cut feed with chop as a regular diet, although this feeding is followed in many places.

The P. &. R. C. and I. Co. feed their stock twice daily. Doctors Hogg, Phipps, and Newhard state that "fresh water should be provided in iron, terra-cotta, or concrete troughs in front of the mangers, where the animal has access to it at all times. Where our stables are equipped with a constant supply of fresh water, colic and acute indigestion are unknown."

Frequently a mule is said to be a kicker and dangerous to handle, but after investigation it is found that the animal has been worked with a collar upside down, or the hames back on his shoulders, and that the stable man or mine foreman did not see the animal go out of the barn and knew nothing of it until it came back from work, with sore places on it. An animal working under these conditions should be admired for trying to defend itself, but the results are it must be sold at a great loss or kept at some other work until well. A stable boss or mine foreman should not allow any of his stock to go out of the barn until he sees that they have the harness that belongs to them, and that it fits properly. To obtain good results from animals the harness must be made to feel comfortable and strong on them.

Part of Mule Barn in Clark Vein, Continental Mine

Good judgment should be used in fitting the bridle, collar, hames, and harness. The collar should be neither too large nor too small, and the hames should fit so they will have the right pressure against the shoulders and the traces should be of equal length. If these matters are given close attention many accidents will be avoided, the stock will be more valuable, and help to keep down expenses in various ways.

It is particularly necessary to give the shoeing of stock close attention, for a good foot is the foundation of a good horse. The stable man should examine the feet carefully to see that the shoes are all tight and properly set before he lets the animal go to work. Then again, if the shoe stays on too long it will cause the hoof to expand over the shoe and produce sore heels and corns. When a shoe comes off and is not promptly replaced, the animal's hoof may be so torn that sufficient nails cannot be inserted to hold the shoe, and should this occur several times successively the animal is of no account for mine work because his feet will not stand the strains. When stock is condemned on this account, it is often said they were of no account when purchased.

There are many mules crippled and made unfit for service by ignorant and inexperienced blacksmiths, who do not know the first principle of proper shoeing, and are unable therefore to give proper attention to the animal's feet. Oftentimes when a blacksmith goes to set a shoe he will just rasp the dirt off of the hoof and then place the new shoe on the exact spot where the old one came off. This treatment in time allows the hoof to grow so out of proportion that it is almost impossible for stock to travel at all. Some blacksmiths do not know enough to pare down the hoof so the shoes will rest on the outer horn, which must be done in order to keep the hoofs as nearly natural as possible. Often blacksmiths drive nails through the sensitive tissues of the hoof, which always lames the animal and is often the cause of stock being so badly crippled in a short time that they cannot be longer used in mine work.

The animals' feet should be kept thoroughly clean, and when they commence gettting hard or become feverish, they should be made to stand in clean, not acid, water at least 2 hours per day until the feet are softened and regain their natural condition.

Quite a number of horses and mules need not be condemned if these comparatively simple matters are given attention.

Having disposed of the subject of crippled stock from the effects of bad shoeing, attention is next directed to making good roads, and this in the judgment of some is the most important question on the inside of the mines. The stock is compelled to travel in mines with very little light, and it is reasonable to suppose that with smooth roads and cars kept well oiled, more can be accomplished with fewer accidents, and a better general condition of the stock maintained. If there are rocks and lumps of slate in holes between the ties, the stable bosses have a good excuse for not being able to keep their animals' legs in good condition. They cannot heal bruised knees caused from stumbling over the ties, nor bruised heels and skinned ankles in one night's time. Often there is a mule brought out on a truck dead, the driver saying he could not get the brake to work, or that the bits on the mining machines caught them and almost tore their legs off. These reports are frequent ones and are not always to be believed if they come from the driver alone, for they make reports of this kind because they have become common; a driver who thus reports should not be allowed a horse or mule until the case has been thoroughly investigated, and the cause remedied, so the same thing will not again occur.

The Fairmont Coal Co.'s records for 1905 show that 26 per cent. of their stock either died, was killed, or had to be disposed of at practically nothing, on account of being crippled and worn out. There is probably no part of the company's business in which the loss is so great, and one-half of this is brought about by carelessness and neglect. There is probably no other business that requires the use of stock in which the loss is so great, and this in face of the fact that the facilities furnished, with the exception possibly of good roads inside the mines, are the best that can be had.

The daily grooming of mine stock is important. It has a tendency to keep the body in better health and causes the mules to be more alert.

Doctor Newhard has introduced a vacuum grooming machine with excellent results at the St. Nicholas colliery stables.

Not more than three mules should be in a string team, as four or five mules cannot be handled successfully by any driver.

Oftentimes bad teeth are the cause of an animal being sluggish and in poor condition; therefore, competent veterinarians should look after the teeth of mine mules once every 2 years.

The maintaining of live stock is no little item, and in cases there is an average of 5 per cent. of the total number of animals standing in the barns all the time unfit for service on account of having been crippled. The feed for this stock, besides other expenses and the loss of their work, costs one company over $6,000 per year.

Where mine stock is given good attention, the upkeep is reduced to a minimum, more work is obtained, and the animals are more valuable. As the methods suggested are reasonable and simple, it is to be hoped that they will be followed by all persons interested in prolonging the lives of mine stock.

OBITUARY

SAMUEL CALVIN

Prof. Samuel Calvin, M. A., Ph. D., LL. D., F. G. S. A., head of the Department of Geology of the State University of Iowa, died at his home in Iowa City, April 17, from dilation of the heart. He was born in Wintonshire, Scotland, February 2, 1840, and came to this country in 1852. He became Professor of Geology at the State University of Iowa in 1874.

Professor Calvin was one of the foremost scientists of the United States, and was recognized in certain fields of geological activity as the world's authority. He was the editor and associate editor of the *American Geologist* from 1888 to 1905, retiring on account of other work. He was a fellow of the American Association for the Advancement of Science, being elected president in 1894. He was a fellow in the Geological Association of America, of which he was president in 1908. He was made State Geologist of Iowa in 1892 and held this position at the time of his death. President Roosevelt chose him as one of the members of the White House conference on the conservation of the natural resources of the United States in 1908. He was a member of the Government Advisory Board on Fuels and Structural Materials, and has written extensively on these subjects. He published quite extensively on the Pleistocene geological period in Iowa, and is considered the authority on this subject. He is made famous the world over for his work on Paleontology. To all Iowa students he is known as "The Grand Old Man of Iowa."

The electrical horsepower consumed by direct-current motor is found from the following formula:

$$\frac{V \times A}{746}$$

Where V = volts; A = amperes; 746 = number of watts in 1 horsepower.

Example: A direct-current motor is using 250 volts, 100 amperes, what is the horsepower consumed? Substituting in above:

$$\frac{250 \times 100}{746} = 33.5 \text{ horsepower.}$$

THE COAL MINES OF DAWSON, N. MEX.

*By Jo. E. Sheridan, Silver City, N. Mex.**

A Description of the Plant of the Stag Cañon Fuel Co. and the Methods of Management and Operation.

The Dawson coal mines are owned and operated by the Stag Cañon Fuel Co., of which Dr. James Douglas is president and E. L. Carpenter general manager. The mines are in the southern end of the Raton coal field, which extends north from Colfax County, N. Mex., into Colorado and embraces the coal mines of the Trinidad section.

The coal measures, known as the Laramie series of the Cretaceous system, have a thickness of about 800 feet in the vicinity of Dawson.

The coal makes an excellent coke. There are but few dikes in the southern portion of the field and little or no faulting along the dikes. There is but little disturbance of the strata throughout the Raton coal field in as far as it extends into New Mexico. table land has been eroded, exposing the green shales below the coal measures, and leaving a bold escarpment along the entire side, whereon each stratum and coal seam is distinctly identified. The Vermejo River and a few small cañons or gulches intersect the land in such manner as to expose a crop line which aggregates a length of about 40 miles. From these exposures the coal seam may be developed by as many openings as are necessary to supply the demand for the product. At present five openings are in operation, known as mines Nos. 1, 2, 4, 5, and 6. Mines Nos. 3 and 5 were connected by entries more than a mile long, between Rail cañon and the Vermejo River. Mines Nos. 1 and 2, located in Rail cañon, have entries driven into the coal for more than a mile; the coal at the faces shows a thickness of 8 feet 4 inches, and is apparently cleaner than that near the outcrop. All of the mines are opened by drifts, which are rendered practicable owing to the continuous outcrop of the coal and the easy and constant dip of the seam, from N 10° W to N 30° W.

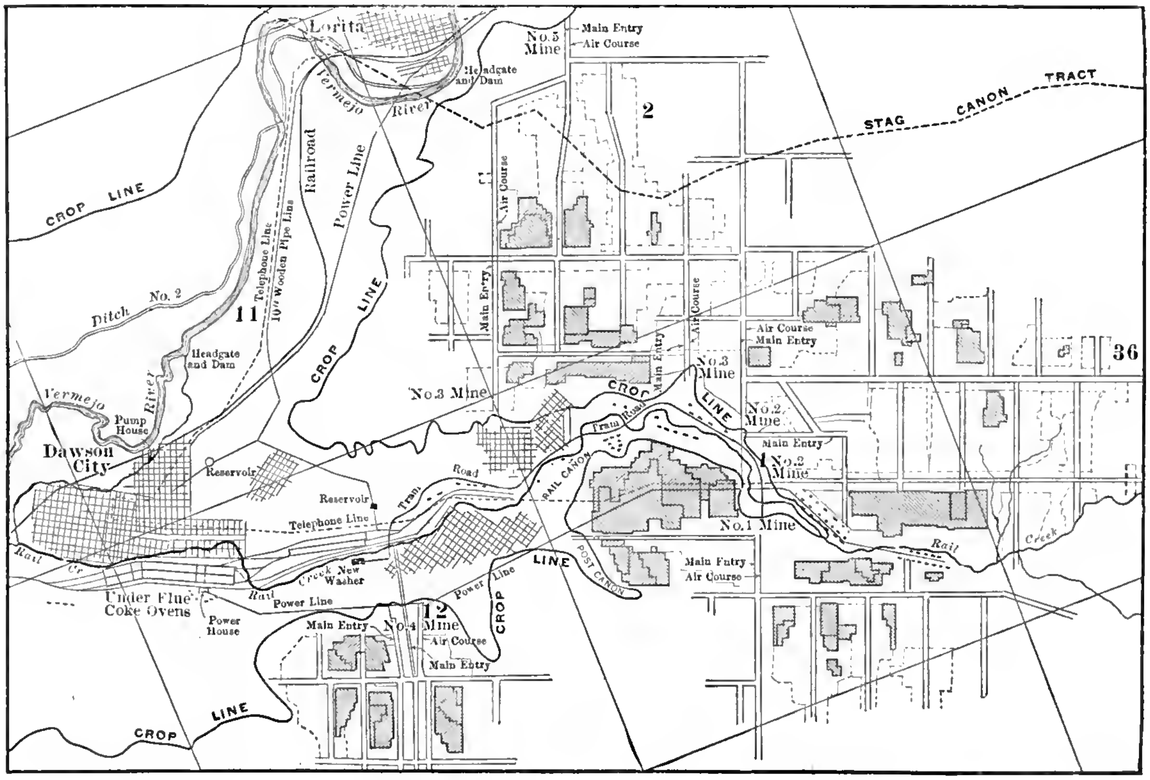


Fig. 1. Working Plan of the Stag Canon Fuel Co.'s Mines, Nos. 1, 2, 3, 4, 5, and 6, Colfax County, N. M.

There are two workable seams in the coal measures, and two or three smaller coal seams, ranging from 1 to 2.5 feet in thickness. The Dawson mines are located upon the lower of the two workable seams, which is known as the Raton or Blossburg seam.

The Stag Cañon Fuel Co. owns about 38,300 acres of land underlain by this great coal seam. In the Dawson mines the thickness of the developed seam varies from 6 to 11 feet, with an average of at least 7 feet. Computing the tonnage which may be recovered upon the usual basis of 100 tons per inch of thickness per acre, there are 321,720,000 tons of coal in the property, and deducting 20 per cent., or 64,344,000 tons, for eroded gulches and insurance against unforseen losses, such as mine fires, squeezes, and other unfortunate conditions, there remains 257,376,000 tons to be recovered.

The topography of the field favors the economical and rapid development of the coal. The eastern projection of the elevated

* Territorial Mine Inspector. Transactions of the American Institute of Mining Engineers

The system of mining is by triple main entries, double cross-entries, room and pillar, and robbing on retreat, when the district becomes exhausted. The width of main entries and cross-entries and air-courses is 9 feet; the height of air-courses, 6 feet 6 inches; the height of roads, 6 feet; room necks are 20 feet long; average width of rooms, 24 feet; average length of rooms, 350 feet; distance between room centers, 50 feet. The coal is hauled by mules from the rooms to the partings within the mine, whence it is brought to the outside yards by electric locomotives, of which there are 10, of the Jeffrey, Westinghouse, and Goodman types. A system of electric signal lights is used, a red light hanging beside the regular mine light. As the motor enters each block a red light is turned on automatically to give warning that a car is coming on that block. Mines Nos. 1 and 2 are ventilated by two Vulcan fans, 24 ft. × 8 ft., exhausting, but reversible. These fans are driven by two 50-horsepower alternating-current induction motors of slip ring, variable-speed type. There are also auxiliary, direct-current 50-horsepower motors, which can be run independently in case

of emergency. Each fan, operating at 60 revolutions per minute, and a pressure of 1.2 inches water gauge, produces an intake ventilating current of about 80,000 cubic feet per minute. Mines Nos. 4 and 5 are ventilated by two Cole 15-foot diameter straight-vane fans.

The following data, pertaining to the operation of mines Nos. 1, 2, 4, and 5, are of interest: Average number of miners on the pay roll, 700; average number in the mines each day, 620; number of company men constantly employed underground, including drivers, trappers, timber men, fire bosses, motor men, and pit bosses, 115; the total air-intake averages 260,558 cubic feet per minute; 59 mules are used for gathering the coal from rooms to the partings; and allowing 600 cubic feet of air per minute for each mule, or 35,400 cubic feet for 59 mules, there remains for the use of the 735 men underground 225,158 cubic feet of air per minute, or 306 cubic feet per minute for each man employed. The water gauge varies from .8 inch in No. 4 mine, with the shortest pull, to 1.2 inches at No. 2 mine, the longest pull. The air measurement is given in the aggregate, for brevity, but each mine has its proportionate share for persons underground, which amounts to three times the quantity required under the United States law governing the operation of mines in the Territory.

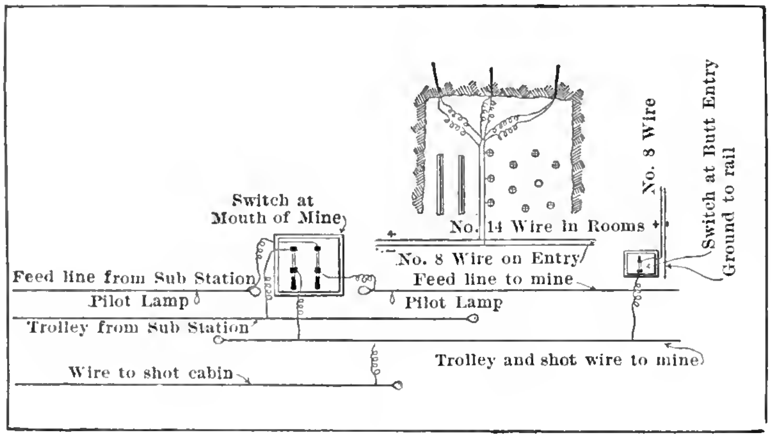


Fig. 2. Diagram of Shot-Firing System in Dawson Mine

About April 1, 1909, an air-shaft will be sunk from the surface at a point 1 mile north from the mouth of mine No. 2. This shaft will be 12 ft. × 12 ft. in the clear, and 250 feet in depth to the intersection of the main return air-course of mines Nos. 2 and 5. A fan of large capacity will be installed at the top of the shaft, exhausting through the shaft, the present openings to be used as intakes.

From mines Nos. 1 and 2 the coal is conveyed to the tipple in mine cars over a tramway 6,600 feet long, which has a rise of 112 feet from the tipple to the mines. Six locomotives haul these cars, as follows: Two 28-ton Porters, one 20-ton Vulcan, one 18-ton Lima, and two 6-ton Porters. The tipple contains a double Phillips dumper, with two chutes for loading railroad cars; the tipple equipment also includes stationary and shaking screens, for sizing coal for various purposes, also a moving slate-picking table.

The coal from mine No. 4, which is located immediately opposite the tipple of mines Nos. 1 and 2, is delivered over a steel Phillips tipple abutting the tipple of mines Nos. 1 and 2. At mines Nos. 5 and 6, the coal is screened as it is dumped on to railroad cars, the slack being hauled to the slack bin, whence it is elevated to a belt traveling to the washery storage bins.

A complete telephone system, having stations at the most convenient points within the mine, affords communication with every important place in the camp, and through the central station with Santa Fe, Albuquerque, Denver, and other cities.

The mines are sprinkled by water cars to lay the coal dust, which is removed from the roadways, as far as practicable, and taken out of the mine. Extra fire bosses have recently been employed at each of the mines to instruct the men in regard to timbering and to see that every precaution is taken to guard against accident from careless work by the miners.

The following rules and regulations have been adopted by the Stag Cañon Fuel Co. for the government and operation of its mines, and distributed to the employes in convenient pamphlet form under date of August 3, 1908:

1. It shall be the duty of each and every employe of this company to inform himself in reference to his duties under the mining laws of this Territory and to comply strictly therewith.

2. No person in a state of intoxication shall be allowed on any of the works, or allowed to enter any of the mines, under penalty of prosecution for trespass under the law.

3. No person or persons shall be allowed to enter any mine except he be a regular employe of that mine, or unless he has a permit from the mine foreman or superintendent.

4. Persons seeking employment shall procure it outside of mine. No boy under twelve (12) years of age shall be permitted to work in any mine.

5. If any person rides upon or in the mine cars going in or out of the mine or on the tram road, he does so at his own risk.

6. All persons, except those duly authorized, are forbidden to meddle or tamper in any way with any electric lights, switches, signal wires, or shooting wires in or about the mines.

7. No person or persons shall go into abandoned parts of any mine unless permission be granted by the mine foreman.

8. All persons before entering the mine must deposit a check at check-house, and get the same when they come out of the mine.

9. The fire boss shall make, before any person is allowed to enter the mine, a careful inspection, with a safety lamp, of

Fig. 3. General View of Washery and Tipples at Stag Canon Fuel Co.'s Mines, Dawson, N. M.

every working place in the mine, marking the day of the month on the face of the coal in each working place where it can be readily seen. If dangerous gases are found in any working place he will mark on a cap piece or shovel two large crosses with the day of the month between them, thus: X27X, and will place these marks so that it will be impossible for any one to pass them without seeing them.

If a quantity of gas is found, which, in the opinion of the fire boss, would endanger the operation of the mine, he is authorized to close the entire mine or any part of it he thinks endangered. The fire boss must always be on the safe side. The fire boss must not allow gas to be moved where men are working in the return air from it.

After complete examination of the mine has been made the fire boss shall come out of the mine and make a report in Report Book of all dangerous conditions found, which report must be read by the mine foreman before any men are allowed to enter the mine. The fire boss shall remain at mouth of mine, or some convenient place, until all the men have entered the mine, instructing each man as to the condition of his working place.

The fire boss must make an inspection at least once a week of all old or abandoned parts of the mine and report conditions of same in Report Book.

10. The mine foremen shall familiarize themselves with the mining laws of the Territory, and shall comply with the requirements thereof by discharging every duty imposed upon them by law and by the rules of the corporation.

11. They shall visit each working place at least once every week and direct the miners and all other employes in their work, and see that their instructions are complied with. They shall direct the miners to securely prop their working places and see that breakthroughs are driven at proper distances. They shall see that the ventilation of the mine is kept in good condition and that all dangerous conditions are removed as soon as possible. They shall have absolute authority over all underground employes, and shall see that all the rules and regulations are carefully carried out.

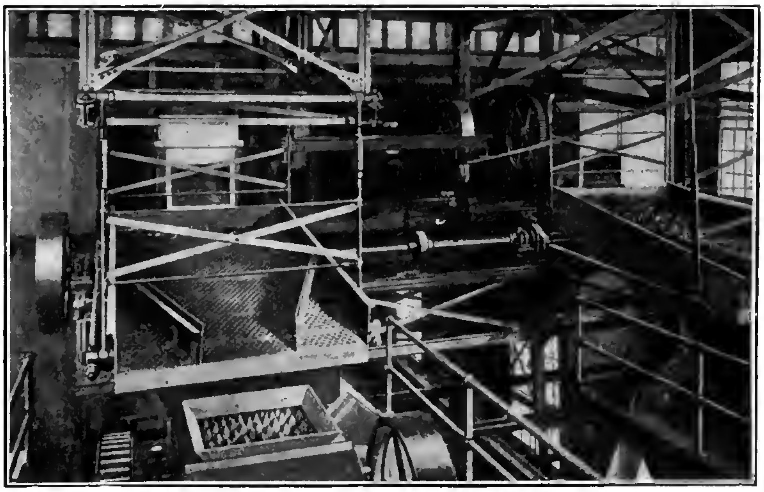


FIG. 4. INTERIOR OF CRUSHER HOUSE, SHOWING SCREENS AND ROLLS

12. All employes shall use every precaution to prevent accidents in or about the mine; they shall not work in an unsafe place when timber would remedy the danger. If timber is not at hand they must stop work and report the fact to the mine foreman. The miner shall each day, before beginning work, examine his working place and take down all dangerous rock, or otherwise make it safe by properly timbering, and shall carefully sprag the coal when undermining.

13. No miner or other employe shall be permitted to burn kerosene, black strap, or machine oil in his lamp.

14. It shall be the duty of every miner to ascertain from the fire boss the condition of his working place before entering the mine.

15. It shall be the duty of the wireman to see that all the employes are out of the mine and the power cut off the mine before he enters the mine to connect up shooting circuits, and to see that all shooting circuits are disconnected from power lines after shots have been fired; also to see that shooting lines are kept up in good shape and that miners are furnished wire for extensions, and to see that all wire is removed from pillars and abandoned places.

He shall make daily report in Record Book of the cutting out and cutting in of shooting circuits.

SHOOTING REGULATIONS

The following regulations for drilling and charging shot holes, mining and cutting the coal, will hereafter be in effect at Dawson mines, and must be strictly carried out by all parties:

1. The mining or cutting must extend at least 6 inches beyond back of holes in all cases.

2. All holes must be at least $2\frac{1}{2}$ feet in length; no shorter holes will be fired.

3. All coal dust must be extracted from holes before they are charged.

4. No holes must be charged with more than five (5) sticks of powder.

5. Standing holes, or parts of standing holes, must not be recharged.

6. The hole in a tight corner must be at least 1 foot from rib at back end of hole.

7. In solid faces, holes must not be more than six (6) feet apart horizontally, and not less than two such holes shall be fired.

8. The object of these rules is to prevent and remove the danger from blown-out or windy shots, and it shall be the duty of the shot inspectors, in addition to the above rules, to refuse to shoot any holes which, in their judgment, may be dangerous, whether the circumstances are fully covered by the rules or not.

9. When giant powder is used in mines not more than fifteen (15) sticks must be taken in the mine for any one working place for any one shift, and in no place must there be more than twenty (20) sticks at any one time.

10. No giant powder must be taken in the mine in a frozen condition, and any attempt to thaw it out in the mine is strictly prohibited. Miners must have their powder supplied to them at the proper temperature to be exploded. Miners are prohibited from accepting, and powder men forbidden from giving out, powder in a frozen condition, and shot inspectors are hereby made responsible for the strict carrying out of this rule.

11. Giant caps must not be kept in the mine; the shot inspectors will give them out to the men, one for each shot, as they are needed, and personally supervise the placing of them in the hole with the powder. Under no condition must they be kept with the giant powder.

12. The powder man will not give giant powder to any person not supplied with a canvas bag in which to carry it.

13. Mine foremen, shot inspectors, powder men, and all others connected with the handling of giant powder going into the mine, must personally see that the above rules are carried out, as far as their supervision in the matter extends.

14. No intemperate man or habitual smoker must be employed as powder man, and, when on duty at the powder magazine, the powder man must not have on or about his person, in the magazine, any pipe, tobacco in any form, or matches, nor any tools or materials from which a spark might be emitted or a light created.

15. When powder is being given out to the miners no one

but the powder man must be inside the magazine, and no person must be allowed around the door of the magazine with a light or while smoking.

16. The presence of women, children, or any person under 18 years of age in or around the magazine is prohibited at all times; also their employment in handling powder, and no powder shall be given out to them.

The shooting is done by electricity after all the men are checked out of the mine. As the men enter the mine they are required to deposit a metal check at the shot-firing house outside, near the entrance to the mine. These checks are placed on a check-board and returned to the men as they come from the mine. A record of the working place of each check number is kept in the shot-firing house, and in case any check is uncalled for, the shot firer makes a search for the man until he is found. No shots are fired until it is known positively that no one is in the mine. The method of placing the shots is shown in Fig. 2.

To insure safety against accidental discharge of the shots by electricity, there are two or more locked switch boxes in each mine, with throw-off switches, one at the mouth of the mine and at one or more stations inside the mine. After inspecting the inside connections with the shots to be fired, the shot firer en route from the mine makes connection at each of the switches mentioned. He then goes to the shot-firing cabin to turn on the electric current, but before doing so he turns on an electric signal light in a red globe, to warn all persons to remain away from the vicinity of the mouth of the mine; so that should an explosion occur within the mine, no one outside could be injured by flying debris. The shot-firing system has proved a success; the safety of the men from disastrous dust explosions due to blown-out shots is assured; miners make better wages, and the production of coal is proportionately greater per man employed. A record is kept of the number of shots fired, showing less than 2 per cent. of missed shots. The missed shots are left for the next day's shooting, and are either reprimed or a new hole drilled to perform the work intended for the original shot. Very little firedamp has been encountered thus far in the mines; but a supply of safety lamps is kept ready for use.

Fig. 5. Hutches from Four Stewart Jigs and Discharge Pipes for Refuse

A Babcock two-cylinder chemical fire engine is kept on a side track, under cover, ready for instant use; also portable chemical fire extinguishers, and helmets of various types to supply means of respiration in any vitiated atmosphere. Hose reels, each carrying 500 feet of best grade of fire-hose, are kept at stations throughout the camp, and a man is employed to inspect daily the hose and fire-fighting appliances.

An organized first-aid corps has had regular practice and competitive drills during the past year, for which the company contributed appropriate prizes and medals for the most efficient team work.

A large building is being erected for a rescue station in which the first-aid corps and others may practice and exercise while wearing the helmets in a chamber filled with vitiated gases. An instructor watches the men, and on showing any signs of exhaustion they will be quickly removed and the gases dispelled from the chamber by suitable outlets. After sufficient experimental work to demonstrate which type of helmet is best adapted to the needs of the mines, a supply will be purchased for use in cases of emergency.

The rescue station is designed after plans of the one in use at the mine of the Dominion Coal Co., in Nova Scotia, modified to some extent. In the upper story of this building there will be a technical library on coal mining, and a "School of Mines" will be conducted by a competent instructor. The superintendents, pit bosses, fire bosses, and others occupying responsible positions in the mines will be required to pass an examination, and if not proficient in the technical and theoretical studies pertaining to their respective positions, as well as in the practical application of these studies, they will be given 6 months in which to perfect themselves. If after this time, they are still deficient, they will be reduced in rank or discharged. It is the aim of the company to introduce and maintain such an excellent standard that a certificate to a graduate of the Dawson School of Mines will be recognized as a guarantee of competency.

The powder magazines at the mines, built of stone, iron, and cement, are absolutely fireproof. The heat is supplied by electric radiators, which maintain a constant temperature within the magazine; the electric stove or radiator and all wires are at a considerable distance from the stored powder, and out of reach of anything combustible or explosive.

The main building of the coal-washing plant is 112 feet long, 70 feet wide, and 70 feet high, and the laboratory and crusher building, are absolutely fireproof, being built throughout of reinforced concrete and structural steel.

Starting at the tipple, the undersize coal from the Nos. 1 and 2 tipple screens is delivered on a 28-inch cross-belt conveyer, running at right angles to the main belt, and carried to a 36-inch belt conveyer. Another 28-inch belt conveyer delivers the slack from the screens of No. 4 tipple to the same 36-inch belt conveyer, and an elevator carries the slack from mine No. 5 slack bin to join the undersize from the other mines on the 36-inch belt conveyer, which conveys the whole to the two 1,000-ton storage tanks, each 40 feet in diameter and 40 feet high. These storage tanks guarantee a constant supply to the crusher house and washery, so that they are not dependent upon the work of the tipples.

Under the storage tanks are two 28-inch parallel belts, upon which the slack coal is delivered from the storage tanks, through eight rocker-gate, adjustable automatic feeders, and conveyed by these belts to the crusher house, where it drops from the belts upon two $6' \times 12'$ shaking screens, about 1.5 inches slope to the foot, .5-inch plate, with 1.5-inch round perforations. The oversize is delivered to two 32-inch toothed rolls, making 125 revolutions per minute and having 100 tons per hour capacity, shown in Fig. 4, which reduce the material to 1.25-inch size to correspond to the sizing of the shaking screen above. The two 28-inch belts and the screens and

rolls are driven by an 85-horsepower General Electric motor. The product from the screens and rolls is deposited upon a 30-inch belt conveyer, which carries it to the dust-proof room on the third floor of the washery. As this belt with its load of slack leaves the crusher house en route to dust-proof room, each 25-foot section is automatically weighed and recorded by a Blake-Dennison automatic and continuous weighing machine. Thus the data of results are based upon accurate figures. This belt is 278 feet long, center to center, 76 feet 8 inches rise, has a capacity of 250 tons per hour, and is driven by a 50-horsepower Western Electric motor.

In the dust-proof room water is added by two 5-inch centrifugal pumps to the crushed coal, and the whole is carried in launders to eight jigs of the Stewart type, two double jigs on each side of jig floor. The jig and water-supply tanks are of steel plate, concrete lined. The pumps which supply water to these jigs are driven by two 50-horsepower Western Electric motors.

From the dust-proof room onward the washery plant is built in two units on the east and west sections of the building, and operated independently or together, so that an accident on one side offers no hindrance to the continued operation of the other half of the plant.

The hutches of the jigs, Fig. 5, taper downward, and are connected with two Luhrig elevators by 8-inch pipes. These elevators discharge the refuse into launders, which deliver it to two refuse trommels 4 ft.×8 ft. All trommels have $\frac{5}{16}$-inch perforations, $\frac{3}{16}$-inch plate, 1.5-inch slope to the foot, and are operated at a speed of 17 revolutions per minute.

The oversize from the refuse trommels passes to rewash jigs of the Stewart type; the undersize is rewashed in four Luhrig jigs, two on each side; the recovery from these jigs joins the washed coal from the primary Stewart jigs, and is conveyed by launders under the jig floor to four dewatering trommels, two on each side, the oversize from which is spouted into two 60-inch Steadman disintegrators, operated at 325 revolutions per minute, where it is crushed to desired size for coke ovens. The east and west side sections of the jigs are each driven by an 85-horsepower General Electric motor.

The undersize from the trommels is recovered from settling tanks beneath by perforated-bucket elevators running 15 feet per minute; and, together with the washed coal from the Stewart and Luhrig jigs, is delivered to a conveyer belt traveling 287 feet 3 inches, to seven 300-ton cylindrical steel storage tanks, each 20 feet in diameter, 40 feet high, and distributed by two drag conveyers operating above the bins, whence it is taken by electric larries to the coke ovens. The rejected material from the various washings and rewashings is picked up by elevators and discharged into the waste tank at the south end of the washery building, whence it is taken by electric trolley cars to the waste dump.

The recovery from the oversize from the refuse trommels carried to Stewart rewash jigs is a product equal in fuel value to the unwashed mine product, and is used as nut coal for domestic or steam purposes. This material is carried by belt conveyer to a circular steel storage bin.

Twenty-seven electric motors, having an aggregate capacity of 1,159 horsepower, are operated in conveying the coal from the tipple and through the crusher house and washery until delivered in the washed-coal storage bins. All motors on the alternation current are three-phase, 25-cycle, 220-volts.

An adjunct common to the mine tipple of mines Nos. 1 and 2 and to the washery is the "run-of-mine" crusher situated at the tipple. The crusher is a McCully gyratory, with a capacity of 200 tons per hour. Should there be any temporary cessation of orders for screened coal for commercial purposes, the whole product of these mines could be crushed and conveyed to the storage bins to be washed and made into coke.

The washery has proved an eminent success. Even in the experimental stage the fuel value of the waste was as low as 8 per cent., and the average loss of fuel values in the waste from the washery now and hereafter will probably be below 5 per cent. The capacity of the plant is 2,500 tons per day of 10 hours, but as there are not a sufficient number of coke ovens erected to utilize this tonnage, the plant has never exceeded 8 hours in constant operation. The washery is located in Rail cañon, at a common center to the greatest area of the coal lands of the company.

A complete laboratory is in a two-story concrete and iron fireproof building, 38 ft.×26 ft. 6 in., opening into the main washery building. The lower story is used for grinding and preparing for analysis samples of coal, coke, bone, and waste; the upper story contains the laboratory proper, which is fully equipped with every modern appliance necessary for the work at hand.

All of the machinery for handling the unwashed coal, jigs, and other appliances used in the washing, as well as machinery for handling the washed coal, was manufactured by the Jeffrey Mfg. Co.

FIG. 6. DEWATERING TROMMELS

OBSTACLES TO DEVELOPMENT OF ALASKAN COALS

Two influences have held back the development of the Bering River and Matanuska fields. One has been the advances made in the California oil districts, and the other the unfortunate conditions existing in regard to the laws under which Alaska coal lands can be taken up.

During the decade ending with 1908 the annual output of the California oil fields increased from about 2,500,000 to nearly 45,000,000 barrels. As probably 80 per cent. of this petroleum is used for fuel in the Pacific coast states, it has to a corresponding extent decreased the demand for coal.

A far more serious handicap has been the coal-land laws. Though laws intended to enable the individual to obtain title to coal lands have been on the statute books for the last decade, not a single acre of land has yet gone to patent. It is therefore not surprising that progress has been checked in the coal fields and that many who would undertake their development have become discouraged.—*United States Geological Survey.*

THE COKEDALE, COLO., EXPLOSION

Written for Mines and Minerals, by Geo. F. Duck

Conditions Existing in the Mine After the Explosion and the Conclusions to Be Drawn from Them

A few minutes before 9 P. M., February 9, 1911, a quantity of 40-per-cent. dynamite left behind by the shot firers near the face of the twenty-first room on the first west B entry of the Cokedale mine of the Carbon Coal and Coke Co., 8 miles west of Trinidad, Colo., exploded through the agency of a blown-out shot.

In this as in other similar accidents in which large loss of life and destruction of property resulted, care must be taken to distinguish between the initial and secondary or propagating causes. It would seem that the various initial causes have been exhausted in the recent explosions in Colorado. At Primero on January 31, 1910, the explosion, so far as known, was due to marsh gas; at Starkville, on October 8 of the same year, to an electric arc brought about by the short circuiting of the trolley wires; at Delagua, on November 8, to an underground fire; and at Cokedale, on February 9, 1911, to the explosion of dynamite.

In each of the above cases the destruction of life and property by the initial cause and at the focus of the explosion was either nil or relatively insignificant. In no instance would the destruction have extended beyond the point of origin had it not been for the presence of a propagating agent. This agent is found in all these cases in a coal dust high in volatile matter (about 31 per cent.), and of marked and even unusually pronounced coking character.

The Cokedale mine was supposed to be eminently safe and had always been so reported by the state inspectoral staff. It was generally well watered, the ventilation was good, the shot firing system was approved, and inspection was regular and thorough. On the other hand, the mine must have been dusty or the force of the exploding dynamite could not have extended beyond the twenty-first room on the first west B entry.

Two views have been advanced to account for the dust in the Cokedale mine at the time of the explosion. One holds that there was no standing dust generally distributed throughout the workings upon ribs, roof, floor, and timbers, but that all the dust concerned in propagating the explosion of the dynamite was due to that made immediately before and at the time of said explosion by the breaking into minute particles of the very soft coal through abnormally and unusually rapid shot firing.

The second view is the general one; namely, that there was standing dust in the mine and that that actually made by the rapid shooting of the coal played an insignificant part in extending the destructive effects of the exploding dynamite.

Before a decision as to the correctness of either view is permissible, a more detailed study of the conditions prevailing at the time of and immediately prior to the accident must be made.

The surface plant at Cokedale, remarkable in the universal use of concrete, particularly in coke-oven construction, has been described in Mines and Minerals.

The coal seam, which outcrops at the tipple site at the level of the railroad track, varies between 4 feet 6 inches and 8 feet 6 inches in thickness, with a fairly well sustained average of 6 feet. It is a member of the Laramie series of the Cretaceous period and is said to be some 200 feet above the bed worked at Starkville, about 4 miles to the east. The seam is universally divided into two members by a parting slate from 2 inches to 8 inches and more in thickness. The upper member of the seam is soft, friable, and of a texture and structure similar to the Connellsville coal of Pennsylvania The lower member is harder and more blocky. The coal averages about 31 per cent. in volatile matter, and possesses marked coking qualities. The entire output of the mine, amounting to 1,300 tons daily, is washed and subsequently coked in the plant of 350 ovens, the resultant product being shipped to the works of the American Smelting and Refining Co., of which the Carbon Coal and Coke Co. is the fuel end.

At the time of the explosion the average number of men employed underground was 275, and outside, 170.

The mine is divided into panels, known as the A, B, and C blocks, by a series of north and south entries from which west entries are turned at right angles. It will be noted that the panels are all connected by entries or rooms. The entries are usually 6 ft. × 10 ft. with a 25-foot pillar to the parallel entry. The rooms are 300 feet long, 20 to 25 feet wide, with a 25-foot pillar to the next room. Gathering is done by mules, and haulage to the foot of the slope is performed by two small electric motors operated under 240 volts pressure. The slope, which, with the trestle approach to the tipple, is about 600 feet long, is equipped with a chain hoist of 2,200 tons capacity. The entry timbering cannot be too highly commended and is very unusual for a bituminous coal mine.

Because the south entries were driven to their limit before the room entries were turned and because these room entries were also driven to their limit before the first room was turned, the naming and numbering of the entries and rooms is the reverse of that usually followed. The entry most distant from the foot of the slope is numbered one with a letter affixed to denote the block in which it is driven, thus, first west B entry. A portion of the mine beyond the first west B entry is called Japan, by reason of the number of natives of that country working therein.

The men entered at the manway, or fifth east entry, where a careful system of checking in was maintained. By this means, 15 minutes after the accident it was possible to prepare a typewritten list of the men in the mine, together with their age, distinctive marks, next of kin, working place, etc.

The method of mining in rooms is unusual, and may be likened to the bench and heading system familiar in tunnel work, but laid out horizontally in place of vertically. The advantage of this system is that, except in the heading or narrow portion of the room, two free faces are always available to the action of a shot, undercutting being therefore unnecessary. On the other hand, unless carefully watched it is obvious that it lends itself to blown-out shots through misplaced holes having too great a thickness of coal to work against. It is practically impossible to maintain the steps at right angles and with a face of just 6 feet.

In the benches, the holes are placed in vertical pairs near the rib, four being needed to square up the face. In the heading, which varies from 6 to 8 feet in width, the coal is rarely undercut the full height of the upper member of the seam. That is to say, it is top-cut near the roof and not undercut at the floor. When cut the full width of the heading one center shot is relied upon to bring the coal down; when undercut partially, one bottom shot near the floor and under the cutting, combined with a vertical pair near the rib are used, and when not cut at all, which is the usual practice, the customary cut and side shots are used. For the reasons given immediately above, as many as seven shots are not infrequently fired in one room at the same instant.

Ventilation is produced by a 15′×7′ Capell fan which, at 80 revolutions and at .9-inch water gauge was circulating some 55,000 to 60,000 cubic feet of air a minute. The fan is set in reinforced concrete above and to one side of a shaft connected with the third north entry or main return air-course. The return underground, Fig. 1, is driven to the crop where it is closed by double explosion doors. The three points of intake, the direction of the air throughout the workings (indicated by arrows), as well as the location of brattices, regulators, doors, over and undercasts, are shown on the map. An arrow with the letter *E* affixed indicates the direction of advance of the explosive wave.

Two fire bosses are employed who inspect the working

places not more than 3 hours before the men enter and who examine the abandoned portions of the mine once a week. They state that methane is of rare occurrence, although sometimes found at the faces of the advanced workings in the C block. As opposed to this, the analyses by Mr. John B. Ekeley, Professor of Chemistry at the Colorado State University, Boulder, and published in the biennial report of the State Coal Mine Inspector, which appeared April 1, are of interest, as showing and eighth west C entries which served as the return air-course. An electrical shot firing system was used up to within a short time of the accident. At the time of the explosion the mine was divided as nearly as possible into four shot-firing zones, to each of which was assigned a head shot firer and an assistant. In addition there was a boss shot firer whose duties seem indefinite and of no immediate value, as he naturally could be with but one gang at a time. About 5:45 P. M., after

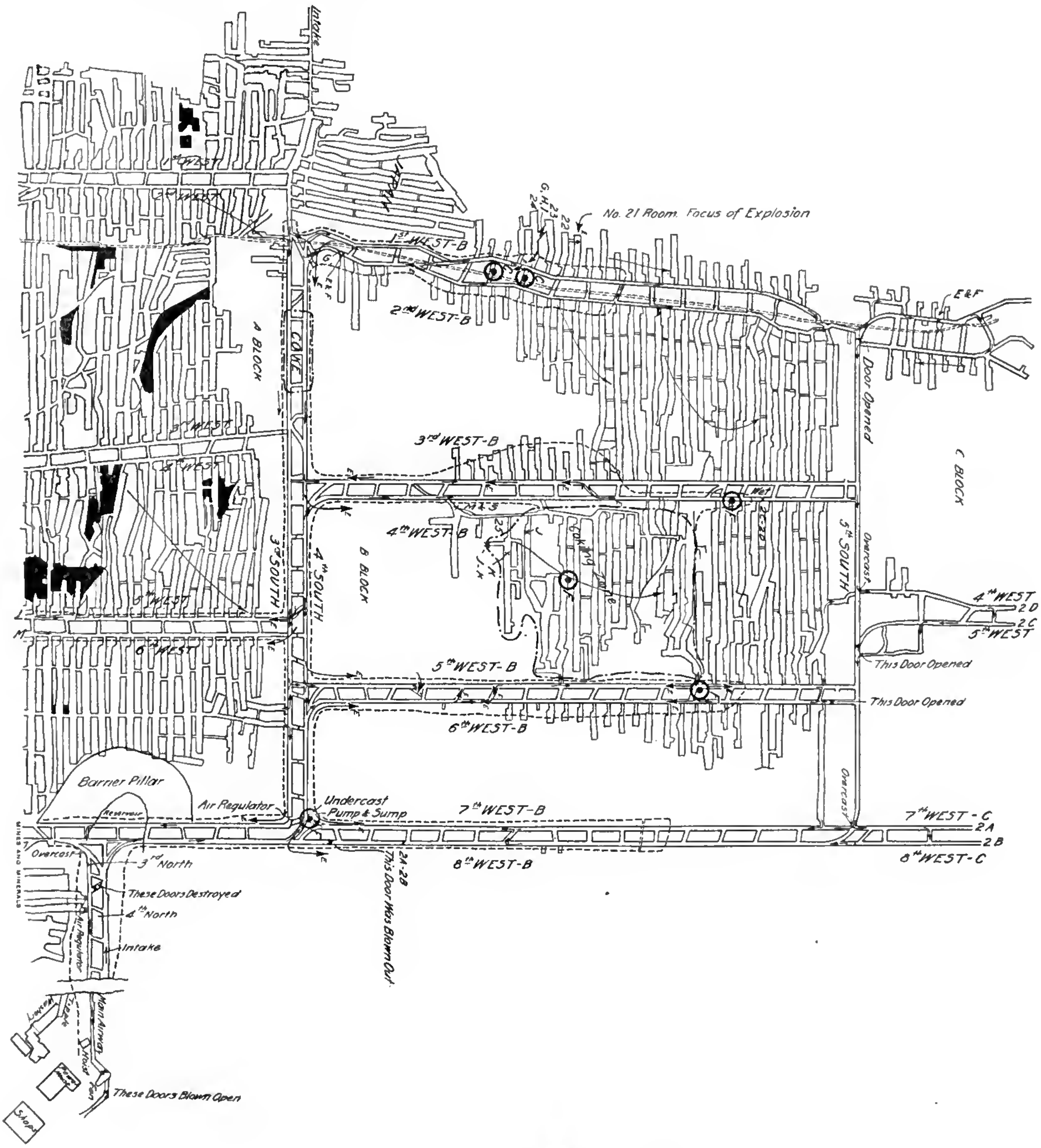


Fig. 1. Map of Part of Mine at Cokedale

the more general presence of gas than suspected. Professor Ekeley found .7 per cent. of CH_4 in the pillars in room 10 on the second west B entry; .8 per cent. at the face of the fifth west C entry, and .4 per cent. in the main return airway in the undercast at the fourth south entry. However, it appears improbable that gas figures either in the origin or propagation of the explosion.

It will be noted that the air-current swept practically all over the mine; that brattices between parallel entries were not usual and that all the air gathered at the in-by end of the seventh the check-board showed that all the men were out of the mine, the shot firing crew met at the small powder magazine on the hill above the manway and received an amount of powder believed to be sufficient for the night's work. This powder, the night of the accident amounting to 648 pounds, was placed in eight or more canvas sacks and loaded into a powder car drawn by a mule. After hanging red lanterns at the mouths of the manway and the main slope the crew entered the mine and at the cap house at the third south entry secured the necessary detonating caps. From a nearby underground telephone,

word was received from the powder house that all current was shut off from the mine. The powder car was then driven into the workings, each shot firer and his assistant removing therefrom at the proper cross-entry the amount of explosive required by them.

When a room was reached the sacks were supposed to be left on the entry, the shot firers taking as much powder to the face in their pockets as they believed would be necessary to charge the holes in the room. At the face, the shot firers were presumed to examine the holes and to reject such as were dirty or tight. After charging and tamping with clay, the procedure varied. In some instances a series of rooms were prepared and the shots fired, room by room, as rapidly as the men could pass from one place to another, while in other cases the room was shot as soon as the holes were ready and then the next room prepared, etc. In no case were sacks of powder to be taken to the face. After all the shots were fired the men left the mine, supposedly about 1 o'clock in the morning, 2 hours before the fire bosses appeared. However, it transpires that the shot firers were rarely in the mine after 11 at night. As it naturally required considerable time to secure the powder and caps, to enter the mine, and to get to the working places and charge the holes, it seems probable that little firing was done before 7. The sustained maximum rate probably was reached about 7:30 and continued to 9:30 or 10 o'clock. In these 2 to 2½ hours, over 600 pounds of high explosive were fired at the rate of 2 to 2½ pounds per minute. As the firing was going on in four sections of the mine simultaneously, a very large quantity of dust was thrown into the air-current at many points. The open condition of the workings combined with the large-volumed air-current permitted the dust stirred up by the rapid shot firing to be disseminated all over the mines.

In order to keep down the dust, water lines were laid throughout the mine, except in the first and second west B entries, which had but recently been reopened after abandonment for some months. At various points on the intake were placed a total of 12 sprays, but these were not always in operation. A water car was also in use. A man known as a sprinkler was employed to wash down the ribs and roof by a hose attached to the main water lines by connections at convenient distances apart. At the inquest this man testified that he washed the places as often as necessary and usually was able to make his rounds once every 4 days. The rooms were not watered. However, as the mine made considerable water, the entries were generally wet and in some places the floor was covered with standing water. The fifth south entry for a large portion of its length is driven through rock shot down to bring up the grade and is naturally wet. The entries in the C block are also naturally wet and the same may be said of the last 500 to 750 feet of all the eight west B entries. The humidity of the return air was determined regularly and ranged from 85 to 95, and even more, per cent.

On the night of the accident, in addition to the eight shot firers and their boss, there were in the mine eight entry men employed two each in the fourth, fifth, seventh, and eighth west C entries, a total of 17 men, of whom two of the shot firers escaped.

The shot firing crew met at the powder house at the customary hour, 5:45 P. M., and there received 648 pounds of high explosive. Of this 166 pounds were two permissibles, which were being tried out, and the remainder consisted of dynamite. After the accident a sack containing one-half the supply of permissible powder was found untouched and the records showed that the remainder of this kind of powder had, with the highest degree of probability, been used up shortly before the explosion. Further, investigation has shown that at the time of the explosion 183 holes had been fired, consuming at the usual rate of 1.4 pounds per hole, 256.2 pounds of dynamite, and this had all been fired in about 2 hours, or at the rate of 2.13 pounds per minute.

At about 8:55 P. M. the two shot firers, who had been accompanied by the boss shot firer, prepared the holes in the twenty-first room on the first west B entry. What happened next is largely conjecture. It would seem from all the evidence at hand that these men, in violation of the rules, carried a sack of powder to a point about 20 feet from the face of this room. How much powder was in the sack is uncertain, but one, well qualified, testified at the inquest it could not have contained less than 20 pounds, and may have held as much as 50 or more pounds of high explosive. It further appears that after the holes were ready the head and boss shot firer started to leave the room, one of them picking up the wooden tamping bar and both overlooking the sack of powder which was in the center of the track. The assistant shot firer, presuming that the other two had taken the powder as was the custom, lit the fuses and went through the breakthrough into No. 22 room, down it to the entry, and was with the others in the cross-cut between the twenty-third and twenty-fourth rooms when the explosion took place. The location of the men at the time is determined by the finding of the tamping bar in the cross-cut mentioned.

Just why the dynamite exploded is uncertain. About 20 feet back from the face, and toward which is directly pointing a shot which blew out both at the back end and the mouth, is a hole in the center of the track of some 6 square feet in area and something over 2 feet in depth in the solid floor. All the timber in the room was blown back from the face and some of it lodged in the mouth of the room and some was thrown out into the entry. The ties were ground to powder and the rails twisted and bent. Sections of the rails were torn out from the web and flanges and thrown considerable distances, and all the evidence points to this having been the focus of an explosion of extreme violence. There were other foci of violence *F*, as shown on the map, in the twentieth room on the fourth west B entry, at the mouth of the eighth room on the same entry, and at the mouth of the tenth room on the fifth west B entry. Aside from the fact that none of these places shows even approximately the violence displayed in the twenty-first room on the first west B entry, there is not known to have been any powder near them or men working in their vicinity. They must be abandoned as points of origin in favor of the one named at the outset. At the origin of the explosion the hole named may have been dirty and the flame from it may have ignited the dynamite, or its detonation may have been caused by a lump of coal thrown against it at high velocity, or possibly by a piece of roof falling thereupon.

The limits within which evidences of heat, as determined by coking action, and of force, as determined either by the scouring of the ribs or by the destruction and displacement of timbers, brattices, mine cars, and the like, are shown by a dotted line surrounding the area so affected.

Evidences of heat and destructive action are naturally not uniformly distributed throughout this area. The centers of force are shown on the map marked with the letter *F*, and with a single exception are where the explosive wave passed from a larger to a much smaller area of workings, thus gathering in intensity by reason of compression. This exception was met in the twentieth room on the fourth west B entry. At this point forces apparently met from opposite directions and, the room being wide, set up a whirling or rotary motion sufficiently violent to throw out the gob some 4 feet deep in a circle 15 to 20 feet in diameter.

At the mouth of the eighth room on the fourth west B entry, and also at that of the tenth room on the fifth west B entry, mine cars were completely destroyed, the wheels being torn off, axles bent, and frame irons curved out of shape. At the undercast at the fourth south and seventh west entries, where was placed a large electric pump, the violence was severe. The pump house of 2-inch lumber was blown in and down into the sump. The pump was badly damaged, the pipe connections,

plunger shaft, and outside gear being broken. The 3-inch floor and a heavy wooden brattice were blown down into the undercast and two 6"×6" supports to the pump were broken. The undercast being built of heavy reinforced concrete was cracked slightly and showed some leakage where it intersected the coal. The rails over the undercast were raised about 4 inches.

At the foot of the slope the destruction was more complete than elsewhere. The timbers at and near the mouth were thrown out and some broken in two and hurled outside 100 feet or more. Cars on the slope were smashed and thrown outside and the loaded trip at the foot was badly shattered. The wooden floor of the chain haul was ripped out and a 12"×14" Oregon pine timber supporting the sheaves at the bottom was broken in two. Two double steel doors set in concrete in breakthroughs between the main slope and the return airway were blown out toward the fan. The concrete was entirely blown out and the doors, badly bent and twisted, were carried 75 feet.

At the fan no damage was done. The explosion doors were blown open and immediately shut automatically and the light roof on the top of the air-shaft was lifted off and at once fell back into place.

Beyond the limits marked on the map there were no evidences of violence other than the blowing open, without damage, of a door or two. Aside from the two mentioned at the foot of the slope, the few brattices in the mine, which were of concrete were unharmed.

The coking action was as variable in intensity as was the force. On the map the zones of greatest heat unless plainly marked are indicated by the letter *C*. In these zones coke up to 2 inches and more in thickness was very general. On the third and fourth south entries between the second west and third west entries was one extensive heat zone. Here, in addition to coke 2 and even 3 inches deep on the caps, the posts were found to be scorched and the solid rib was coked to a depth of ½ inch in places.

The most extensive area of coking action was found in the rooms on the fourth west B entry. In some of the rooms in this section coke was found on the ribs from roof to floor and from mouth to face, as well as on the roof and floor, and on the timbers, and in one instance was nearly 6 inches thick. In some of these rooms there was an additional deposit of carbon as lampblack in long festoons or whiskers upon the coke, indicating that the oxygen in these rooms had been completely consumed.

The course of the explosion, owing to the open and connected character of the workings was difficult to trace, but it is believed that its direction, as shown by the arrows on the map, has been fairly well determined. It will be noted that in some instances the wave came through some rooms out an entry to be met by an entering wave from the fourth south. Where these waves met there was generally a neutralization of force or else a whirling action through the cross-cuts. In any case the advance of the wave may be plainly traced from the twenty-first room on the first west B entry to daylight.

The effects of the heat wave displayed the usual vagaries. The mule hitched to the powder car which was standing near the mouth of the first west B entry was burned, as well as was a sack containing dynamite in the car itself, but no explosion occurred. In one room a coil of fuse would be found burned and in the adjoining room under apparently identical conditions the fuse would be untouched.

Strictly speaking, the explosion at Cokedale was rather an inflammation or burning of coal dust ignited by an explosion of dynamite rather than a true explosion in the accepted meaning of the latter word. There was most intense heat and but little damage throughout the workings. The writer is forced to the conclusion that the wide area affected by the explosion, aside from the dust generally present, was due to the large amount of oxygen available for burning whatever dust was present and the universal connection of the workings.

Without any exception whatsoever, where the heat action ceased, the stoppage was due to want of fuel in entries, which were not only damp but actually wet and not uncommonly shoe-top deep with mud. In one instance the flame leaped over a wet zone about 150 feet wide but was stopped shortly beyond. In some places the presence of rock in the track dust helped to stop the spread of the inflammation, and in others the same end was accomplished by rock in the ribs and roof where the latter had been shot down in a local swamp.

On the map are given certain combinations of letters and figures as *2A*, *E*, and *F*, *2D*, etc. The letters indicate the place where a man or men were working at the time of the accident; the figures, the number in the place in question. A letter without affixed number means that but one man was working there or that, if more than one was employed in the place, the bodies were not found together. It will be noted that two men were working in the seventh and eighth west C entries, denoted by *2A* and *2B*, respectively. After the explosion these men started to run out and were a trifle over 1,800 feet from the face when overcome by gas. The four men in the fourth and fifth west C entries (*2C* and *2D*) were found 1,300 feet away on the fourth west B. The two shot firers (*E* and *F*) who were in the first west C entry were able to go over 1,900 feet, their bodies being found on the first west B entry near its intersection with the fourth south entry, with those of the two shot firers and the boss shot firer (*G*, *H*, and *I*) who were in the breakthrough between the twenty-third and twenty-fourth room on the same entry at the time of the explosion. The two shot firers (*J* and *K*) were working in the newly started twenty-fifth room on the fourth west B entry in the hottest part of the coking zone, and were only able to go 75 to 100 feet before being overcome with the gas.

The evidence at the inquest showed, although no analyses of the blood were made, that all the men died from CO poisoning. Their bodies were flaccid and faces peaceful with none of the signs of strangulation apparent in those dying by means of CO_2.

Of the 15 miners and shot firers who succumbed to the effects of gas, the lives of 10, those working in the C block, might have been saved. It will be noted that the explosion did not come within 500 feet of the C block at any point. Also the two doors on the fifth south entry used to force the air up the fourth and fifth west and first and second west entries, respectively, were blown open. The poisonous gases, therefore, did not enter the C block at all, and had the men remained in their working places and bratticed themselves off (there was an abundance of brattice cloth and lumber at several places on the fifth south) they could have for all practical purposes remained in absolute safety indefinitely.

The lives of the three men on the first west B and of the two on the fourth west B entry (*G*, *H*, *I*, and *J*, *K*) could not have been saved, as they were either inby a coking zone or immediately in it.

The two survivors were at the time of the explosion near the intersection of the sixth west B and second south entries at the point *L*, *M*, on their way to shoot a few pillars in the A block. Both testified that they first felt a rush of cool air in the face which blew out their lights and which was instantly followed by a hot blast from inside which knocked them down The hot air had a distinct smell of burning coal. This preliminary inrush of cold air is interesting and is analagous to the experience of two men near the outlet of the Lick Branch, W. Va., mine, at the time of the second explosion there, to which my attention has been called by Mr. John Verner, of Chariton, Iowa.

The rescue cars of the Colorado Fuel and Iron Co., stationed at Trinidad, and that of the Victor American Fuel Co., from Hastings, were on the ground shortly after midnight and their large force of trained men and excellent equipment rendered invaluable service, as the rescue crew of the mine itself, being

composed solely of shot firers, were either dead or, as in the case of the two survivors, incapacitated for further exertion.

As stated before, but little damage was done to the mine and the temporary stopping of the destroyed brattices at the foot of the slope, of the door at the eighth west B, etc., soon restored the normal circulation. The bodies were all discovered in a few hours except two, which were not found and brought out until Saturday noon.

An extremely unfortunate feature of this accident was the death of E. A. Sutton, assistant superintendent at Cokedale, and of Robert Meeks, tracklayer at Starkville, who ventured beyond the rescue station at the fourth south and sixth west B entries, and who were overcome by *CO* gas in the fourth west B entry at a point about 300 feet in and marked by the letters *M S* on the map.

It remains then to consider the two theories of dust origin as commented upon at the outset in this article. Was the dust which propagated this explosion that which was made during the few minutes prior to the accident by the abnormally rapid shooting of friable coal, or was it that which had accumulated in the workings as the result of days or weeks of operation?

Draw a line between the twenty-first room on the first B entry and the twenty-fifth room on the fourth west B entry. Whatever dust was created by shot firing on the west side of this line did not figure in the explosion, because the air-current was drawing it away from the initial point, and also the explosion did not travel inby from its point of origin. Hence, only the dust created by shot firing on the outby, or east, side of this line could have served as a propagating agent. But there were no shots fired on this outby, or east, side the night in question. Had some been fired in Japan the conclusion would not be affected for the inflammation did not enter Japan for lack of fuel. Hence we must dismiss the idea that the explosion of dynamite in the twenty-first room on the first west B entry can be connected with the next nearest place of shot firing, the twenty-fifth room on the fourth west B entry, by means of newly created dust.

By way of the route of the explosion it is a full half mile between the above points, and in this distance there is one zone of most intense heat and two smaller ones. The first and second west B entries were undoubtedly dry and dusty, being without water lines and were but newly opened up. It is probable that the third and fourth south had not been watered for 3 or 4 days, the sprinkler being then about to reach them on his rounds. It is an easy matter to connect up these points on the supposition of standing dust and impossible to do so on the theory of newly created dust.

Similarly, it does not seem that the intense heat and enormous deposits of coke in the coking zone on the fourth west B entry which covers an area of over 4 acres were due to any thing but standing dust. It is impossible to say how long it would have taken at Cokedale to clear the atmosphere of dust after a shot was fired. The slowly moving air-current was favorable to the speedy deposition of the dust and 10 minutes after firing a shot may have seen it all deposited. As the section under consideration was but one of four shot-firing zones, the rate of fire therein would be one-fourth of the rate for the entire mine. That is, if the average for the mine demanded the firing of 2 pounds of dynamite a minute, this section calls for the explosion of 2 pounds in 4 minutes, or a total of 5 pounds in our assumed 10 minutes. To any one who has seen the deposits of coke in this zone, some of them covering an area of nearly 500 square feet, and varying from ½ to 1 inch, and rising to 4 inches in places in thickness, combined with the festoons of soot in some of the rooms, the view that this heat action was due to the limited amount of dust possible through the explosion of but 5 pounds of dynamite, is untenable. Nothing but standing dust can account for the phenomena.

And finally in this connection, had there been any large amount of dust in suspension in the air in this section it would have, by means of the air-current, been drawn into the C block workings. That this was not the case is proved by the fact that the explosion on all the west B entries stopped within 600 feet of this block. Had the C block been dusty it would have inflamed.

This theory has been touched upon at length, as it is a most important one, as pointed out in this journal when commenting upon an editorial in the *Mining Journal*, of London, England, treating of the Hulton colliery disaster. Intensity of working, abnormally rapid shot firing and the like, unquestionably favor the propagation of explosions by means of dust created through the rapidity of production. But this was not the case at Cokedale.

It does not seem that the Cokedale accident teaches any new lessons, but it does serve to emphasize some old ones. It shows that open and connected workings combined with an air-current of large volume favor the propagation by means of dust of an explosion otherwise started, by disseminating the agent of this propagation over a large extent of workings and by furnishing a large volume of oxygen (air) for its combustion. It shows that, by reason of the above, to limit the spread of an explosion, the workings should be, as far as possible, not connected, and should be ventilated by separate splits with only as much air as the health of the men demands. It shows that material damage to the mine is done where the wave passes from a cross-section of larger to one of a smaller area by reason of the compression thus set up. It shows that zones of intense heat action are not necessarily connected with zones of violence. It shows that, in the presence of intense heat action, very wet zones up to 150 feet in length may be crossed by the inflammation, and it shows that very wet zones, particularly if in rock or where the floor dust is heavily impregnated with rock, and of sufficient length, will invariably stop the advance of a wave of inflammation.

This accident clearly demonstrates that rescue crews should never consist of less than four men each; that they should be carefully trained, be drawn from men of diversified lines of work, and be equipped with the latest and best apparatus.

It will be admitted that if oxygen (air) be absolutely excluded from a mine there can be no inflammation; if there be no dust, there can be no inflammation regardless of the amount of oxygen (air), and that, regardless of the amount of oxygen (air) and dust present, if this dust be in the condition of watery mud, if the mine be absolutely saturated with water, there can likewise be no inflammation.

When the relationship between these three, oxygen (air), dust, and moisture (water), which must exist in order to render impossible the propagation by means of dust of an explosion otherwise started, has been determined, accidents such as those described herein will be impossible.

LIBRARY OF THE ENGINEERING SOCIETIES

Attention is called to the facilities offered by the Library of the Engineering Societies, at 29 West Thirty-ninth Street, New York City. The library, formed from the combined libraries of the American Institute of Electrical Engineers, the American Society of Mechanical Engineers, and the American Institute of Mining Engineers, contains over 40,000 volumes on engineering subjects. It is open for reference to the general public, without charge, every day and evening, except Sundays.

Those residing in New York City and vicinity are invited to utilize the library in their researches on technological subjects. The library is prepared to furnish references on engineering subjects to persons at a distance, and also furnish transcripts, translations, and photographic reproductions of diagrams and maps. For such work, if extensive, a moderate charge is made. Correspondence is welcomed; telegraphic and telephonic inquiries will receive especial attention.

PUMPING AND SIPHONING HOT WATER

Written for Mines and Minerals, by J. T. Beard

A common difficulty experienced in mining and steam-engineering practice is that of handling hot water by means of pumps and siphons. In the drainage of mines it frequently happens that the exhaust steam of the pumps or underground haulage engines is conducted into the sump where its condensation raises the temperature of the mine water in the sump to a degree that seriously interferes with the action of the pump in raising this water to the surface. Under these conditions a pump or a siphon may be rendered utterly helpless to draw the water from the sump. In many cases, judging from the frequent inquiries made, the cause of the trouble is not known to the pump runner or the mine foreman in charge, who may spend several hours in a fruitless endeavor to locate and remedy the evil. The difficulty and annoyance is greatly increased at

Difficulties and Their Causes. Effect of Temperature of Water and Elevation Above Sea Level

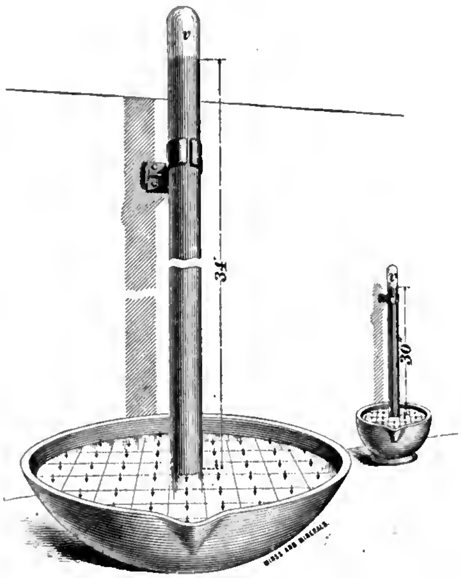


Fig. 1

high altitudes, and is greater as the temperature of the water increases.

In order to clearly understand the difficulty and comprehend its cause, it is necessary to call to mind that in the case of both the pump and the siphon, it is the pressure of the atmosphere acting on the surface of the water in the sump, or supply basin, that forces this water to enter and rise in the suction pipe a height dependent on two factors: (1) the pressure of the atmosphere or barometric pressure; (2) the degree of vacuum within the pipe. A vacuum gauge on the pump or at the crown of a siphon will show the net result or the lifting power in such case, and determine the height of suction.

Turning now from the pump and the siphon, consider, for a moment, the principle of the mercurial barometer. The atmospheric pressure acting on the surface of the liquid in the basin (Fig. 1) supports a column of liquid of such height that its weight is equal to the pressure that sustains it, area for area, as long as a perfect vacuum exists in the tube, at v, above the column, so that there is no pressure at that point. For example, the normal atmospheric pressure, at sea level, is 14.7 pounds per square inch; and since 1 cubic inch of mercury weighs .49 pound, the height of mercury column the pressure of the atmosphere will support at sea level, or the normal barometric height at sea level, is $14.7 \div .49 = 30$ inches. In like manner, since 1 cubic inch of water, at say 62° F. (normal), weighs .03608 pound, the height of water column the pressure of the atmosphere will support, at sea level (temperature of water 62° F.), assuming a perfect vacuum, is $14.7 \div (.03608 \times 12) = 33.95$, or practically 34 feet.

In the case of water, however, there cannot be a perfect vacuum above the column of liquid since this space v (Fig. 1) is always saturated with water vapor, the vaporization taking place instantly, to the point of saturation. This water vapor exerts a pressure that increases rapidly with the temperature and reduces the height of the water column supported by the atmosphere. At 62° F. (normal) the tension of water vapor is .2731 pound per square inch, which is equivalent to .5561 inch of mercury column, or .63 foot of water column. This is no inconsiderable reduction in the suction lift of a pump or siphon even at normal temperature and sea level. At higher temperatures of the water and at greater altitudes the reduction may and often does incapacitate the pump or the siphon, as the case may be. Thus, at sea level, the theoretical water column (34 feet), supported by the atmosphere under normal conditions, is reduced by the vaporization of the water to 33.3 feet when the temperature of the water is 60° F.; 30.3 feet at a temperature of 120° F., and to 17.1 feet if the water is heated to 180° F., as is often the case with feedwater before it is pumped into the boiler.

The foregoing refers only to the static water column supported by the atmosphere, and shows the effect of the vaporization of the water in the reduction of the height of water column supported. In the operation of a pump or siphon there are other losses that greatly reduce the vertical height the atmospheric pressure will force the water in the suction pipe, or what is called the suction head. Briefly summarized these losses arise from the following causes: leakage of valves and packings, and slippage of water past the piston; air absorbed or entrapped in the water, and which escapes under the reduced pressure of the vacuous space above the suction column; friction of the water in the suction pipe and resistance of valves. These losses vary greatly according to the character of the apparatus and the conditions that create them. In common practice such losses may be assumed as averaging 25 per cent. of the theoretical head, after making due allowance for the reduction of the suction head by reason of the vaporization of the water.

Table 1. Safe Suction Head for Pumps at Different Elevations and Various Temperatures of Water

Elevation Above Sea Level Feet	Atmospheric Pressure Pounds Per Square Inch	Barometric Pressure Inches Mercury	Temperature of Water Raised Degrees Fahrenheit				
			60	90	120	150	180
			Safe Suction Head for Pump (Feet)				
10,000	10.107	20.582	17.0	16.4	14.7	11.3	4.7
5,000	12.224	24.890	20.5	19.9	18.2	14.8	8.2
4,000	12.689	25.837	21.5	20.9	19.2	15.8	9.2
3,000	13.169	26.813	22.4	21.8	20.2	16.8	10.2
2,000	13.665	27.824	23.2	22.6	21.0	17.6	11.0
1,000	14.174	28.861	24.1	23.5	21.9	18.5	11.9
Sea level	14.696	29.925	25.0	24.4	22.8	19.4	12.8
Tension of water vapor at various temperatures..........		Lbs. per Sq. In.	.255	.693	1.683	3.706	7.500
		Inches mercury	.518	1.410	3.427	7.547	15.272

Suppose, for example, at an elevation of 2,000 feet above sea level, a pump is to be installed to pump water having a temperature that may reach 120° F., and it is required to ascertain the height it would be safe to set the pump above the low-water mark in the sump or basin. The normal atmospheric pressure at this elevation is 13.665 pounds per square inch. Deducting from this the tension of water vapor at 120° F. gives for the effective pressure that acts to force the water up the suction pipe to the pump $13.665 - 1.683 = 11.982$ pounds per

square inch. A cubic inch of water at 120° F. weighs .0357 pound; and this pressure therefore corresponds to a water column of $11.982 \div (.0357 \times 12) = 27.9$ feet. Now, deducting 25 per cent. to allow for frictional and other losses, the safe suction head, in this case, is $27.9(1 - .25) = 21$ feet.

On this basis Table 1 has been prepared and is offered here as suggestive of the variation in the safe suction head

Fig. 1. Three-Piece Steel Set Used by Susquehanna Coal Co.

for pumps and siphons, at different elevations above sea level, and for various temperatures of the water raised.

The values given for the safe suction head in the above table refer to pumps and siphons where the suction pipe is vertical, or nearly so. When the suction pipe is much inclined, as is sometimes the case in pumps, but more often and quite generally the case in siphons, the length of the pipe and consequently its frictional resistance is increased and varies inversely as the sine of the angle of inclination of the pipe, or the angle the pipe makes with the horizontal plane. The valve resistances, leakage, or slippage, and other losses are not changed.

To ascertain how the inclination of the pipe affects the suction head it may be assumed that of the 25 per cent. of losses in vertical suction, 10 per cent. is due to pipe friction. If this be approximately the case under ordinary conditions of installation, the suction-head values given in the table, and which apply only to practically vertical pipes, must be modified by multiplying by the expression $1 - \frac{10}{75}\left(\frac{1}{\sin a - 1}\right)$, where a is the angle of inclination of the pipe. The limiting angle of inclination is found by making this expression equal to zero, when $a = 6° 45'$, in which case the length of the suction pipe is 8.5 times the vertical lift. For example, assuming the above conditions, the suction pipe of a siphon, at sea level, and using water at 60° F., would cease to deliver any water at the crown when the length of the suction pipe was $8.5 \times 25 =$ say, 212 feet. The water would run slower and slower as this limit was more nearly approached; and it would be necessary to reduce the size of the long leg (discharge pipe), or to throttle the discharge, to prevent the siphon from emptying itself.

The values given in the table for the safe suction head under different conditions, are only intended to be suggestive of practice under ordinary conditions. A good pump will often exceed the suction head given here by 1 or 2 feet; or a poor pump may not stand up to the values given. Bends and elbows in the suction pipe greatly reduce the head The principle of suction, however, in its application to pumping and siphoning, is clearly shown; and the effect of elevation above sea level and raising the temperature of the water is made plain. The aim of the writer has been to show, in a practical manner, the limitations of these operations, and to give sufficient data for the calculation of the safe suction head in any given case.

INTERLOCKING STEEL MINE SUPPORTS

Written for Mines and Minerals

In an interesting article on "Steel Supports in Coal Mines,"* R. B. Woodworth, engineer for the Carnegie Steel Co., says: "For the legs and props, where loads have to be carried in compression, the highly economical integral section was not rolled in the United States 4 years ago. The **H** beam, which is the closest practical approximation in the form of a rolled shape to the section of the largest compressive resistance in proportion to its weight, has been found the section best fitted for use both as single props and for legs of the three-piece gangway support, and its introduction can be said to mark an era in the history of American mine timbering as it did when introduced in England by the Darlington Iron and Steel Co., in 1885. The use of this section has been extended to include the framing of mine shafts. When used for wall plates of a mine shaft the **H** beam has the advantage of a large bearing surface as compared with beams or channels of the same depth, and for buntons and compartment separators the advantage of a large compressive strength combined with a comparatively high moment of resistance against bending."

When R. Van A. Norris introduced the three-piece set in the Susquehanna Coal Co.'s mines at Nanticoke, Pa., he made use of channels for legs and **I** beams for collars. The collar rested on a pin that separated the two steel channels that formed the leg. This is shown in Fig. 1. Later on, 85 sets of the three-piece supports shown in Fig. 2 were introduced in the Allport Coal Co.'s mine at Barnesboro, Pa. In the latter sets the collars are riveted to the legs. Dr. Wesley Waite got the notion that he could improve on this kind of mine support and invented the knock-down interlocking three-piece set, shown in Fig. 3 knocked down and in Fig. 4 set up. As shown in Fig. 3, the collar a and the legs b of **H**-shaped iron or steel are slotted in such a manner that the legs will interlock with the collar.

The slots made in both ends of the collar do not extend quite to the web, in order that it may not be weakened and

Fig. 2. Steel Gangway Supports, Allport Coal Co., Barnesboro, Pa.

further that some play may be allowed between the members so that a natural adjustment may occur when pressure comes on the set. The legs are slotted so as to remove the web and permit the web of the collar to rest on the flange of the leg. If desired, the legs and collar can be matched on the floor and raised as a whole into place, or the legs can be stood and the collar placed on them. In Fig. 4 the legs are given a batter, but they can be stood upright, in which latter case a key keeps

* Mines and Minerals, Vol. XXXI, No. 7, page 387.

the legs from lateral motion and so vertical. The slots in the collar are made to conform with the angle at which the legs are to stand.

In the Waite patented interlocking mine-support system, the legs and collar act as an integral piece of steel to the section and impossible of shearing at any angle. To destroy the jointing the whole section must be crushed. An advantage claimed for this construction, in addition to the ease with which it may be adjusted to conform with requirements in setting up, is that each set is so adjusted that it will take up strain throughout many sections on account of the loose joints. Where steel is riveted this strain comes directly on the parts affected and it cannot yield. In wooden supports there is a certain yielding which is advantageous. The two pieces of iron under the collar in Fig. 3 are hooks for spacing the bents and keeping them from moving out of place when pressure comes on the sets unevenly. Where the legs are battered inwards, as in Fig. 4, downward pressure tends to tighten the joints; or if there is lateral pressure on either or both sides of the set both the legs and collar resist it as a unit on account of the interlocking.

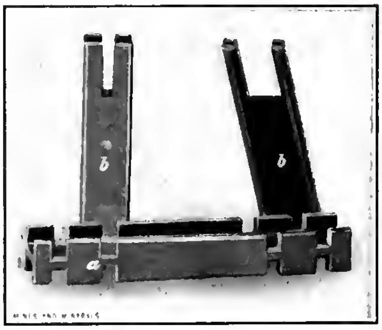


Fig. 3. Knockdown Three Piece Steel Mine Support

"SHEELITE" IN NOVA SCOTIA

An interesting event is the recent discovery of "sheelite" in Halifax County, Nova Scotia. Sheelite is one of the minerals containing tungsten. It is of no known use in itself, excepting as an ore from which tungsten may be extracted. The mineral is chemically a tungstate of calcium. As an ingredient in the chemical side of steel making it is quite important. At present the world's annual output, coming mostly from Sweden, is placed at 4,000 tons. If present indications are correct, the recent discovery in Nova Scotia will not only yield sufficient for the steel plants in the Province, but will have an effect on the markets of the world. The ore is reported to yield 60 per cent. of tungsten acid to the ton of sheelite. At present 25 men are at work on the preliminary experiments. It is believed that Mr. Hiram Donkin, Deputy Commissioner of Mines, Halifax, Nova Scotia, will give interested parties all reasonable information.—*From Consul John E. Kehl, Sydney.*

REPORT OF COAL MINE INSPECTOR OF COLORADO

Written for Mines and Minerals, by Geo. F. Duck

The recently issued report of James Dalrymple, State Inspector of Coal Mines for Colorado, for the years 1909 and 1910, covers a most interesting period in the history of coal mining in that state; a period during which the fatalities reached the enormous total of 21.6 per 1,000 employes. So far as our records show, but twice in the history of coal mining in the United States has this rate been exceeded: In Utah in 1900, when it was 138.96 per 1,000, and in Tennessee in 1902, when it was 25.80 per 1,000. In British Columbia in 1901 the rate was 25.67, and in 1902, 34.65, and in Nova Scotia in 1890, 25.17. The abnormally high death rate in Colorado during 1910 is due to four major accidents which cost 220 lives: Those at Primero on January 31, 76; Starkville on October 8, 55; Delagua on November 8, 79; and at Leyden on December 14, 10. These four accidents alone caused a death rate of 14.89 per 1,000. The difference between the total fatality rate for the state, 21.6 per 1,000, and that due to these four accidents, 14.89, is 6.71, or essentially the same as the rate of 6.76 per 1,000 for the year 1909, when there was, strictly speaking, no major accident, the largest being that at Toller on July 6, when nine lives were lost.

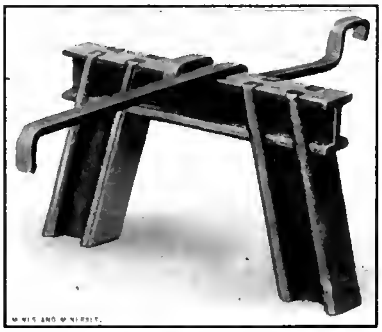


Fig. 4. Steel Mine Support Assembled

In addition to very interesting reports on the Toller, Primero, and Delagua accidents, by John D. Jones, the recently retired Chief Inspector, there are others of moment by James Dalrymple, the present chief, on the Leyden fire, and by Frank N. Oberding, deputy inspector, on the accident at Starkville. There is also a very timely article by Prof. John B. Ekeley, of the University of Colorado, Boulder, on the coal dusts and mine gases of the state, with analyses of numerous samples from various mines.

The following interesting statistics are taken verbatim from the report:

General Statistics

	1909	1910
Number of mines in operation	167.00	155.00
Number of new mines opened up	15.00	12.00
Tons of lignite coal produced	2,173,877.00	1,654,955.00
Tons of semibituminous coal produced	855,762.00	1,014,588.00
Tons of bituminous coal produced	7,613,332.00	9,284,758.00
Tons of anthracite coal produced	59,519.00	80,586.00
Tons of unclassified coal produced, estimated	70,000.00	70,000.00
Total number of tons of coal produced	10,772,490.00	12,104,887.00
Increase in number of tons as compared with 1909		1,332,397.00
Tons of coal mined by hand	9,033,057.00	10,563,651.00
Tons of coal mined by machinery	1,739,433.00	1,541,236.00
Total number of mining machines used	208.00	222.00
Total number of tons of coke produced	1,076,593.00	1,190,901.00
Total number of coke ovens	3,240.00	3,164.00
Number of employes in and about the mines	13,156.00	14,768.00
Number of employes at the coke ovens	1,089.00	1,090.00
Number of non-fatal accidents	116.00	146.00
Number of fatal accidents in the mines	89.00	319 00
Tons of coal mined for each life lost	121,039.10	37,946.30
Tons of coal mined for each non-fatal accident	92,886.30	82,910.40
Number of employes for each life lost	147.80	46.20
Number of killed per thousand employed	6.76	21.60
Number of employes for each non-fatal accident	113.40	101.20

Accident Statistics

	1909	1910	1909	1910
Total number of accidents	79	96		
Total number of men killed	89	319		
Total number of men injured	116	146		
Serious injuries			79	115
Minor injuries			37	31
Total number of wives made widows	50	163		
Number of children left fatherless	71	303		
Causes	Killed		Injured	
Gas explosions	10		13	9
Dust explosions (including mixtures of dust and gas)	3	210		
Falls of roof, coal, rock, and draw slate	63	73	70	79
Powder explosions			2	2
Crushed by trip cars	7	18	25	28
Shaft accidents	2	1		2
Miscellaneous	2	3	6	26
Electrocuted	1			
Suffocated	1	14		
Total	89	319	116	146

BRITISH COAL MINING NOTES

Written for Mines and Minerals

Indian Coal Mining Methods.—In the principal coal fields of India the customary method of coal working is pillar and stall, or bord and pillar. Referring specially to the mines in Bengal in the last annual report issued, Mr. J. R. R. Wilson, acting as chief inspector, says that from the mouth of the incline, or the limits of the shaft pillar, the area is gradually cut up into pillars, the output depending upon the amount of coal extracted in the galleries. This operation continues until the boundary of the royalty is reached, and then the work of extracting the pillars is commenced.

Sometimes it happens that the demand for coal is so large that an effort is made to take out the larger percentage of coal in the seam by means of the galleries, and the pillars that are then left are barely of sufficient strength to resist the weight of superincumbent strata; consequently, when the boundary of the property is reached the mine has to be abandoned and the coal in the pillars is lost. Further than this, gallery driving yields a far greater percentage of small coal than pillar cutting, the work is more arduous, the cost per ton is higher, and the output per person employed is considerably less, so that for long after the mine has started it is worked at the greatest disadvantage from every point of view.

It is further stated that several seams in India are liable to spontaneous combustion, and the danger from fire is always present after pillar cutting or goafing has started. The small pillars of coal left behind as temporary supports are crushed into heaps of dust by the weight of the roof, and spontaneous combustion invariably follows. When this happens it is impossible to get near to the heated coal to shovel it out, or to quench it with water, on account of the fallen stone in the goaf, and the only effective remedy is to isolate the area where the fire occurs from the rest of the workings. With a network of galleries opening out of the goaf on all sides this is often a gigantic undertaking, for every outlet will have to be hermetically sealed by walls to prevent the spread of the fire. Unless a fire is checked or extinguished immediately after its discovery in the goaf the chances of successfully coping with it are very small.

The assumption of Mr. Wilson is that the adoption of any system that would facilitate the separation of parts of the mine from the rest and at the same time obviate the aforementioned difficulties must be of immense practical value. It is recalled that in the early days of active coal mining in England these difficulties presented themselves, and in 1809 Mr. Buddley, a famous North of England colliery manager, overcame them to a very great extent by the introduction of the panel system, a system which effectually separated adjoining districts of the mine by leaving solid ribs of coal of varying thicknesses between them. The workings are laid out in districts or panels in this system of a size to suit the local condition, and when one panel is cut up into pillars these are immediately extracted. Only three or four roads are made into each panel for the purposes of ventilation and haulage, so that in the event of a fire or a sudden inrush of water the panel could be quickly dammed off. Mr. Wilson concludes that the introduction of paneling into Indian methods of mining coal cannot fail to be beneficial in every way. A minimum of risk in case of fire or sudden inrushes of water, the production of better coal, and a quick return on the capital expended, are amongst the chief advantages to be derived.

Stone Dust Zones.—The Official Record of the British Coal Dust Experiments, issued on behalf of the Mining Association of Great Britain, is an important and a valuable document. In carrying out the experiments at Altofts, in Yorkshire, attention was given to details, and more than 800 coal owners, colliery officials, inspectors of mines, and scientists from all parts of the United Kingdom, from India, and South Africa, and from France, Germany, and America, witnessed demonstrations. So much that is general has been conveyed to the rank and file of mines regarding the value of stone dust as a means of preventing the ignition of coal dust, that an excerpt may be acceptable, giving the suggestions of the committee for distributing stone dust in the pits.

They say the effect of stone dust as a "zone" in the path of a coal-dust explosion would seem to lie in its power of offering resistance to the projection of the flame of the explosion. That is to say, its action is mainly mechanical. The cloud of incombustible particles in the air immediately in front of the explosion presents a denser atmosphere, offering greater resistance and prevents the flame of the explosion from penetrating so far as it would in dust-free air. At the same time it diminishes the danger of the flame of the explosion spreading through the cloud of unburnt coal dust driven in front, since it mixes with it and in this way raises its ignition point. For stone dust to act in this way, however, it is essential that it must be fine enough to be raised as a cloud in the path of the explosion, and it has yet to be proved whether its action would be sufficiently rapid when dealing with an explosion that has traveled a longer distance and has attained its maximum velocity of propagation. The value of stone dust would appear to lie more in its use as a diluent, thus preventing an ignition, than in any specific action it may have in stopping an explosion that has once started. It would, therefore, seem advisable not to employ zones of any description, whether dustless, watered, or stone dust, in spite of the good results that have been given by the last named when dealing with an explosion that has traveled 275 feet. The better principle would appear to be to treat with stone dust all places where coal dust can accumulate, and in this way guard against the primary ignition of coal dust; for it is a far easier matter to prevent an explosion ever occurring than to stop it after it has traveled some distance, and it is, without doubt, preferable to exclude all possibility of the formation of carbon monoxide by the combustion of even a few hundred pounds of coal dust.

Some Interesting Coal Committee Points.—Dipping into the pages of the Record of the Coal Dust Experiments some highly interesting points may be culled. Amongst other things we are told:

The fact that coal dust, in the complete absence of firedamp, is explosive when raised as a cloud in air and ignited, has, in the opinion of all who have witnessed the experiments, been definitely established.

The development of the flame of the explosion after it issues from the downcast end of the gallery has been studied by means of cinematograph records. The existence of a "pioneering cloud" in front of the explosion has been established, and evidence has been obtained that the true flame of the explosion has a length of from 60 to 80 feet, or possibly less. Lengths of flame outside the gallery of 150 feet and upwards that have been recorded are shown to be due to the subsequent burning of the cloud of coal dust that issues in advance of the flame.

It would appear that the presence of a cloud of incombustible dust in the path of a coal-dust explosion that has traveled 275 feet, checks the continued propagation of the explosion. The experiments in which stone dust has been intimately mixed with coal dust also tend to show that as the percentage of incombustible dust is increased it becomes increasingly difficult either to originate an explosion in the mixture or to cause an explosion to be propagated. It is further shown that the use of stone dust might strike effectually at the root of the danger by controlling one of the factors that are essential for the occurrence of a coal-dust explosion; namely, the inflammability of the dust.

The problem of the mode of propagation of coal-dust explosions is a very complicated one. Of the facts that have been established, the most important are the increase in the pressure developed with increased distance of travel of the explosion, the marked influence of the presence of obstructions in causing

the explosion to be propagated with greater violence, and the possibility of propagating an explosion through a cloud of wood-charcoal dust and air.

Colliery Rescue Brigades.—The British Government, as represented by the Home Office, would appear to be determined upon securing a better system of rescue organization, and a Departmental Committee has reported upon the desirability of the formation of rescue brigades at the collieries, each brigade to consist of not less than five persons employed at the mine, carefully selected for their underground knowledge, coolness, and powers of endurance. The first stipulation is that no person, unless authorized by the manager, or an official appointed for the purpose, shall be allowed to enter a mine after an explosion or the occurrence of a fire for the purpose of engaging in rescue work. Then brigades have to be formed as follows: One brigade where there are less than 250 men employed below; two brigades where the number is between 250 and 500; three between 500 and 800; four above 800. Small mines employing less than 100 miners will be deemed to have complied with the provision upon acquiring the privilege of calling for a brigade from a central rescue station. The order requires that there shall be provided at each mine two sets of portable breathing apparatus for each brigade, capable of enabling the wearer to spend at least an hour in bad air; two electric hand lamps for each brigade; a safety lamp for every member of the brigade. Two or more small birds or mice are to be maintained at every mine for testing for carbon monoxide, and tracings of the workings must be kept up to not more than 3 months previously, the plans to be in suitable form for use by the brigades. At the Central Rescue Stations there must be not less than 15 complete sets of breathing apparatus, properly maintained; 20 electric hand lamps; four sets oxygen reviving apparatus; ambulance boxes, together with antiseptic solution and fresh drinking water; cages of birds and mice; whilst a motor car must be kept in constant readiness.

Maypole and Hulton Disasters.—In the course of an address before the Lancashire branch of the National Association of Colliery Managers, Mr. G. H. Winstanley, lecturer in mining at the Manchester University, recalled that only once in the period of 22 years ended August, 1908, did Lancashire figure in the list of principal colliery explosions. Then the Maypole disaster dispelled any delusion that Lancashire had done with explosions, whilst a few weeks ago, at the Hulton collieries, there occurred a greater explosion than ever previously recorded in the history of the county, and only twice in the history of the world. Mr. Winstanley asked the members to consider the facts. At Maypole, at the time of the explosion, no electricity had been installed. There was no shot firing in the working shift, a good type of safety lamp was in use, excellent airways, and a fan capable of producing more than half a million cubic feet of air per minute. Never had he seen a mine that looked less likely to be the scene of an explosion. As to the Hulton collieries, it was there that the hydraulic wedge, as a substitute for blasting, had its birth. There was little or no shot firing in the true sense of the word, or none in the working shifts. The collieries had earned the reputation of being up to date and replete with all manner of modern improvements. Like Maypole, it had been somewhat of a "show" place. Both places were well conducted and carefully managed. The fact they had to face was that apparently no matter what care might be exercised and what precautions adopted to prevent disaster, they might, any one of them, have a similar experience.

Pointing out that they could not have an explosion without ignition, Mr. Winstanley urged upon managers to "try and fight ignition." Preventive measures might be broadly divided into three groups: (1) Direct treatment of the gas or dust, so as to render it incapable of ignition; (2) the establishment of zones, a sort of forlorn hope to prevent the spread of an explosion already started; (3) precautions to prevent ignition. Water was of all methods the least effective in dealing with the coal dust difficulty. In regard to shot firing, if the strict letter of the law was observed at all times, together with such additional precautions as circumstances might suggest, the risks from this cause might also be looked on as negligible. The match danger was not to be overlooked. They read that within 2 months of that most terrible disaster at Hulton a box of matches was found hidden in a tub at the bottom. He was afraid there was nothing for it but to search the workman before he descended, not with a view to punishment after the offense had been committed, but to prevent the commission of the offense. He urged the importance of managers impressing upon the workmen that they should take a share of responsibility in maintaining the safety in the mine. Bring the workman up in the belief that individual neglect or carelessness might be the cause of a disaster and great loss of life, and that freedom from such calamities was to be attained by a firm determination on the part of every one in the mine, that so far as he is concerned, he will do nothing that might endanger its safety.

PERSONALS

Wm. L. Saunders, B. S., delivered the class day address to the students of the Michigan College of Mines, April 20, 1911. Mr. Saunders is much in demand as a lecturer, because he says something worth while in his speeches.

Prof. Louis D. Huntoon, of the Sheffield Scientific School, Yale University, has resigned the chair of metallurgy at that institution and will open an office in New York City as consulting mining engineer, after the close of the college year.

D. D. Davis, son of W. G. Davis, district superintendent of D., L. & W. coal operations, has been appointed to succeed the late J. E. Evans as superintendent of the United States Bureau of Mines Rescue Car at Wilkes-Barre, Pa.

James Ashworth, manager of Crow's Nest Pass Coal Co., Fernie, B. C., has resigned. He will be succeeded by W. R. Wilson, former general manager of the company. Under Mr. Ashworth's management the Coal Creek and Michel mines have been placed on a sound basis.

J. J. Rutledge, W. S. Rountree, J. D. Weldon, J. W. Key, J. W. Newby, A. R. Brown, William Routenlusch, H. H. Hamilton, G. T. Gambill, Doctor Caraway, and Harry McCrorie, had a narrow escape from death while investigating the cause of the Banner, Ala., mine explosion on April 9. The party were overcome by afterdamp, but were recovered and returned to inspection in a few hours.

Henry R. Cobleigh has resigned as mechanical editor of *The Iron Age*, which position he has held for the last 7 years, to take charge of the advertising and publicity of the International Steam Pump Co., No. 115 Broadway, New York City. He entered upon his new duties May 1.

J. C. Anderson, of Westinghouse, Church, Kerr & Co., Pittsburg, has resigned to take a position as mechanical engineer in the sales department of the Pressed Steel Car Co., New York.

C. T. Van Winkle, former superintendent of the Magna plant, Utah Copper Co., is now consulting engineer for the Duffy estate, of Rochester, N. Y.

W. R. Holbrook, formerly with the Balaklala Consolidated Copper Co., Coram, Cal., is now chief chemist for the Mexican-American Steel and Rail Co., at Guaymas, Mexico.

Daniel Ryan, of Terre Haute, Ind., has been elected president of the Hocking Valley Products Co. the title under which the reorganization, of the Columbus and Hickory Coal and Iron Co. has been effected.

W. J. Jenkins has been elected vice-president and general manager of the Western Coal and Mining Co., of St. Louis, to succeed B. F. Bush. This company operates mines in Illinois, Kansas, Arkansas, and Oklahoma.

SOUTHERN KANSAS COAL DISTRICT

Written for Mines and Minerals, by Lucius L. Wittich

Geology—Extent of Development. Methods Used in Sizing and Cleaning the Coal

More than 65 per cent. of the coal production of Kansas comes from the Southern Kansas district, embracing the counties of Cherokee and Crawford, in Kansas, and Barton County, in Missouri. Comparatively little of the production comes from the latter county, the heaviest producing areas being north and west of the city of Pittsburg, in Crawford County, and near Weir City and Scammon, in Cherokee County. Numerous smaller mining camps are scattered over the Southern Kansas area, among them being Arma, Frontenac, Minden, Mineral, Crowburg, and others. While estimates of the production from this district for 1910 have not been made, it is thought the output will be materially

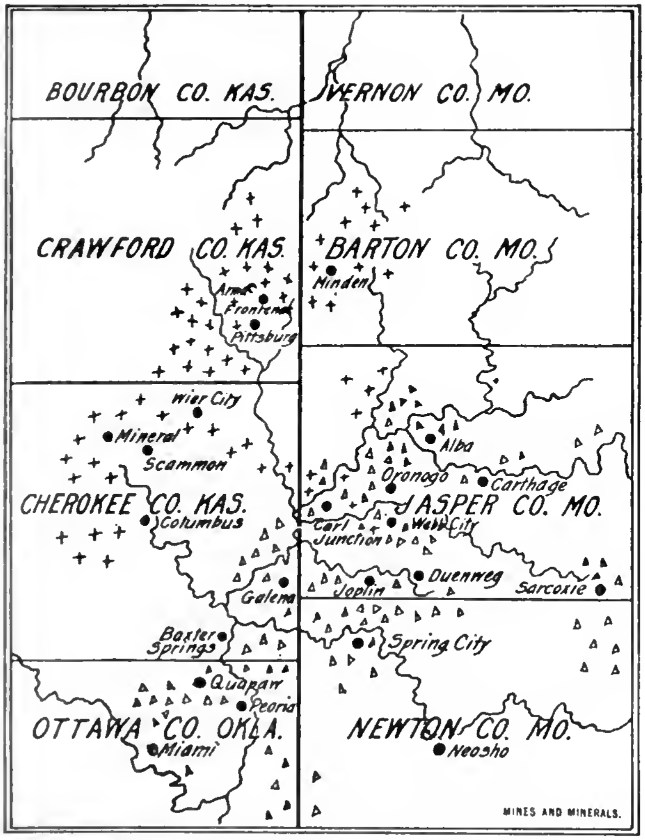


FIG. 1. COAL AND ORE ZONES OF KANSAS AND MISSOURI
Crosses, Coal Areas; Triangles, Mineral Regions

less than that of 1909, when the shipments aggregated 6,986,478 tons, with a valuation of $10,083,384. The miners' strike, which began April 1 and continued until September 15, caused a heavy curtailment of the production.

This coal district adjoins the Missouri, Kansas, and Oklahoma zinc and lead district, to which a considerable tonnage of the Kansas coal is shipped for consumption at the metal mines. In places the coal fields extend over into the ore belt, as is shown by Fig. 1, the crosses indicating the coal area while the triangles show the developed mineral regions. From this map it will be noted that the coal measures extend to within a few miles of Joplin, Mo., while, also, it will be noted that the ore deposits extend northward in Kansas, overlapping the well-defined coal belt. From several tracts in Jasper County, Mo., coal, lead, and zinc are produced from the same mine; the coal, although of medium quality as a rule, occurring in sufficient quantities to supply not only the fuel for the mine from which it is produced, but also furnishing cheap fuel for many surrounding ore mines, where the operators have been less fortunate in having their own coal beds from which to secure their supply.*

The Southern Kansas district is distinctive as the producer of an unusually good quality of coal in which the percentage of carbon is high; the Missouri, Kansas, and Oklahoma district is noted for the production of more than 55 per cent. of the zinc ore of the United States that is converted into primary spelter; yet, closely related as it would seem these two big districts might be, they are as widely different as day and night; the hypothetical line that marks the division between the coal belt and the ore belt that outlines the western fringe of the Mississippian series of rocks before it dips beneath the Pennsylvanian, also marks the division between a district that is purely American and one that is pronouncedly cosmopolitan. Foreign labor in the ore belt is the exception; the advent of foreigners is discouraged by the native employes, and, in a number of instances, when efforts have been made by mine operators to introduce foreign help, violence has resulted; for the American toilers look upon the zinc and lead zone as their own, and accordingly resent intrusion.

To go from the heart of the distinctly American ore mining community to the heart of the Kansas coal fields is a trip of barely an hour by luxurious electric palace cars, by train, or by automobile, but the brevity of the journey only emphasizes the astounding transformation. It is like journeying from New York to Italy in 60 minutes; it is a trans-Atlantic expedition in tabloid form, with all the unmistakable foreign atmosphere in evidence when the coal fields are reached. The tidy cottage of the zinc and lead miner is replaced with company settlements of severe box houses, close to one another, stringing across the prairies in painfully regular rows; even the landscape seems different; strange plants adorn the door yards; unfamiliar vines creep over the porches; odd-looking potted shrubbery graces the windows.

In the coal district the workman buys his own powder, at so much per keg, from the company that employs him or direct from the powder company; in the wash plants of the coal district, where a jigging process not unlike that employed in the ore mines, is used, the waste is the heavy portion that responds to the gravity treatment, while the marketable product is that which washes away.

While Crawford County is the heaviest producer of coal at present, Cherokee County formerly held this distinction. Early discoveries of coal were made in Cherokee County in 1867, the first production coming from mines near the state line east of Columbus. The later discoveries to the north resulted in a much superior article of coal being produced, as the early miners found a rather inferior grade of fuel, the depth of the deposits ranging from 40 to 80 feet, at which depth many of the mines of the extreme southern part of the south field are still being worked, while the mines north of Pittsburg are down to a depth of 200 feet. Leading coal companies operating in the Southern Kansas district are the J. R. Crowe Mining Co., the Sherridan Coal Co., the Fidelity Coal Mining Co., the Central Coal and Coke Co., the Columbus Coal Co., the Western Coal Mining Co., the Hamilton Coal Mining Co., the Mayer Coal Co., Clemens & Son, and other smaller concerns.

When the coal fields of this district were first prospected the general run of coal appeared to be confined to the immediate vicinity of Columbus and Scammon, but later developments showed the coal to extend from the southwest to the northeast, passing through Crawford County and up into Barton County, Mo. So numerous are coal mines now through the northeastern part of Cherokee County and the eastern part of Crawford County that it would be no easy matter to select a spot from which some coal mine could not be picked out with the naked eye.

* The editor has a piece of sphalerite attached to slate that was found in a Missouri coal bed.

The early part of 1910 saw operators pushing their production in anticipation of the labor strike which drew thousands of men from their jobs, leaving the coal producers helpless to meet the demands; hence, the surplus stocks of coal at the mines were enormous when the general walk-out finally was declared. Big mining companies and power companies of the Joplin district laid in enough fuel to carry them for months; railroads depending on the Southern Kansas coal in their traffic across the plains carried coal by the hundreds of thousands of tons to terminal points, where the fuel was heaped in gigantic piles to be used when the occasion demanded. But all supplies were well exhausted when regular production was resumed late in September.

The result of the strike was a victory for the workmen, whose wages were raised generally throughout the Middle Western States, Kansas being included. Miners now receive 75 cents per ton and furnish their own powder. A state law prohibits the use of dynamite in the coal mines, but this has proven to be a law that is violated. For turning a room off the entry the miner is allowed $2.37 additional to his fixed charge per ton for coal. Eight hours constitute a working day for all employes save the engineers, who work 9 hours. The drivers, cagers, track layers, and roustabouts receive $2.70 a day; top men receive $2.14 a day; shot firers receive $3.17 to $8.50 a day, the element of danger determining the amount. In especially dry, dusty mines where the shot firer momentarily is in peril of being converted into a blistered mass, the maximum wage will be paid, while the minimum amount will be his income in the safest mines of the district, where the air is moist and where there is little dust in circulation.

About 10,000 miners are employed in the Southern Kansas coal district, and of this number fully 40 per cent. are Italians, 20 per cent. negroes, and 40 per cent. Americans—all the miscellaneous clans of half-breed foreigners being included with the Americans. There are Scotch, Irish, Germans, Poles, Austrians, Swedes, Slavs, and representatives from almost every other nationality. For the most part the foreigners are jovial, generous, in some instances, to a remarkable degree, and excellent workers as a whole. Almost to a man they are thrifty, for almost to a man they have one crowning ambition: to return to the old country with their pockets bulging with American gold. Last spring when the strike went into effect the coal field representatives of the various steamship lines did a thriving business, and almost every train, for weeks after the strike was declared, carried homeward-bound travelers, eager to return to their native lands. Autumn witnessed a return of the excursionists to America, many of them bringing recruits to be initiated into the ways of the Americans, others bringing relatives. Intermarriage is common among the foreigners of various nationalities, and so it is that an oncoming generation is made up largely of an indiscriminate blending of the races.

FIG. 2. WASHERIES AT DUNKIRK, KANS.

The Southern Kansas coal district is a portion of what is known as the central bituminous coal fields, embracing the states of Missouri, Kansas, Arkansas, and Oklahoma. Around this area extends a zone, barren of coal, which depends to a large extent upon the coal production from the central fields. The states of Colorado, Dakota, New Mexico, and Wyoming, on the west; Ohio, Illinois, Indiana, Kentucky, and Pennsylvania on the east; and Alabama and Tennessee on the southeast, are the coal producing states surrounding the central fields.

The bituminous coal deposits of Southern Kansas yield the bulk of the fuel mined in that district, and in the state for that matter, although some semianthracite and lignite coals are found, the two latter types being little mined. These occur farther to the west.

The coal deposits, which are found outcropping at points along the Missouri-Kansas line dip gradually to the north and west. West of Pittsburg, Kans., open-cut mining is practiced on a limited scale, this type of coal mining having grown less in importance during the past few years, as many of the workable deposits have been exhausted. The line of outcrop of the coal deposits is from the southwest to the northeast. In the ravines the outcropping is especially noticeable, although on the higher elevations an occasional outcrop will be observed.

The thickness of the veins worked ranges from 2 feet 8 inches to 3 feet 6 inches; the roof, as a rule, being of shale. Some gas is encountered but not in sufficient quantities to cause inconvenience. Through the enforcement of a recent state law, all entries now are supposed to be 6 feet in height, and as the law is enforced rather satisfactorily, the old style low entry is rapidly disappearing. Much of the coal is shot from the solid, although efforts to abolish this method are being made.

Coal from Cherokee and Crawford County, and also from Bourbon County, to the north, comes from the Cherokee shales, the chief coal-bearing formation of the state, and one which extends westward, gradually attaining depth, but still containing coal, which eventually may be mined, although much deeper operations will be necessary to produce it. Erasmus Haworth, head of the Geological Survey of Kansas, says: "The abundance of coal within the Cherokee shales seems to be so great that it need not be a surprise if heavy beds be found under any part of the eastern 50 or 75 miles of the state."

Above the Cherokee shales occur the Labette shales, carrying no workable deposits of coal, and above these come the Pleasanton shales, carrying deposits of coal probably second to none save those of the Cherokee horizon. Above the Pleasanton shales come the Thayer shales, which likewise carry some coal; next the Lane shales with no coal; then the Lawrence shales, carrying coal in sufficient quantities to be mined at a profit in places. However, these occur much farther to the west and are not included in the Southern Kansas district.

The positions of the coal deposits within the Cherokee shales vary both vertically and horizontally. Near the extreme base of the shales, thin coal, unfit for commercial uses, is found. Farmers mine it for local consumption from numerous insignificant pits. About 150 feet above the base of the Cherokee shales a better coal bed is found. It lies under a heavy layer of sandstone and is directly covered with a thin layer of shale. But the heaviest coal deposits of the state are found in the Weir-Pittsburg lower and upper formations, the width of the lower run averaging 40 inches, while the upper will average 30. The first scientific recognition of the existence of these coals came in 1868, when a Chicago professor examined the outcroppings and rendered a favorable report, but for years prior to this time coal had been mined by the settlers and transported to the early lead camps in Newton County, Mo. In 1870 a railroad penetrated the coal belt and immediately the

demand increased. From then on, the railroads have consumed a large part of the coal produced, and today half a dozen lines depend on this district for their fuel supply to the north, the west, and the south.

Two lesser beds of coal, outcropping farther to the west, are found above the two Weir-Pittsburg horizons, but their commercial importance is insignificant compared to the more important formations. Another coal deposit, still higher, occurring at the very top of the Cherokee shales, is of some commercial importance and much coal from this level has been produced for commercial purposes, especially from mines in Bourbon County, north of the main Southern Kansas district.

Fig. 3. Jig Room in No. 2 Washery

Virtually all of the mines now running to any great extent in the vicinity of Pittsburg or Weir City secure their product from either the lower or upper coal beds of the Cherokee shales, although an intermediate layer exists, which sometimes is mined, although it is too thin as a rule to be worked at a profit.

The room-and-pillar system of mining is most common in the mines of Cherokee and Crawford counties, although the longwall system occasionally exists. The latter system is found convenient where the coal formation is unusually thin, but where the coal occurs in thicker strata, ranging in width from 3 feet upward, the room-and-pillar system has advantages. The thickness of the coal deposit makes it unnecessary to remove a great quantity of waste rock, as is the case where thinner strata are worked; therefore, the absence of the waste makes it imperative to find other material for support of roofs. It is customary to leave pillars of coal standing, to be removed later, at a time when the immediate workings are to be abandoned.

The double-entry method of operating the room-and-pillar system is employed in many of the Kansas mines. After the shaft has been sunk to a depth below the coal stratum to be worked, the greater depth being employed as a sump into which the mine waters drain, main entries, 12 feet wide and 6 feet high, are driven in opposite directions. On either side of the main entry, paralleling it, but separated by supporting walls of coal, are side entries. All three entries are driven to the boundary line of the property or coal.

From the side entries at intervals of about 100 feet, two parallel entries, 32 feet apart, are driven at right angles, and these extend as far as the workable deposits of coal occur, or as far as the property lines of the company permit. By this system of regular tunneling and cross-tunneling, the entire coal formation is blocked out and rooms are opened from which the larger tonnage of the mine's production comes. The entries are connected at intervals of 100 feet by breakthroughs, while between the two sets of double entries, cross-entries parallel to the side entries are driven and connect the two inner entries of the double sets. Such passageways, which aid in ventilation and which are used for hauling, are driven every 40 or 50 feet along the double entries.

The rooms, which are 35 feet from center to center, and which are worked to any length desired, are opened along the sides of the double entries opposite the pillars. Between the rooms, pillars about 11 feet wide are left, thus making the actual mined-out width of the rooms 24 feet.

Double tracking is used for hauling in the main and parallel side entries, the connections between the two being made at an angle of about 45 degrees. By alternating the openings from the main entry into the side entries at either side, the network of tracks is somewhat simplified, and the number of tracks coming out into the main entry at the same point is reduced. All entrances from the side entries to the main entry angle toward the shaft.

Where the single entry is used, only one main entry, with no side or secondary entry, is employed. Running from this entry, at right angles, are single entries, 100 feet apart, from which the rooms are driven.

Few supports of an artificial nature are required in the Southern Kansas coal mines. The roof, arched properly, is self-supporting as a rule, and timbering is required only where the coal is found beneath unusually soft shale.

In advancing a room, a series of cuttings are employed, obviating the necessity of placing a charge of powder into a solid face of coal. The first cutting is wide enough to permit the free action of a pick. The cut is extended about 3 feet into the coal face and reaches from the top to the bottom of the stratum. With the removal of coal from this small trench, space is left into which the coal can be forced when the charge is fired. Otherwise the blast would show a tendency to "squeal" or would result in a "windy" shot; that is, the coal face, instead of being broken into large, free chunks across the greater portion, would be reduced to dust in the immediate vicinity of the charge, and would not be affected some distance away from the charge. Where "horsebacks" are encountered, the miner receives additional pay for his labors; hence, the cost of mining where "horsebacks" are common is not only higher but the quality of the fuel is inferior.

In the majority of the mines, mules are still employed as motive power, and in virtually all of the mines mules are used in the side entries beyond the parting from the main entries.

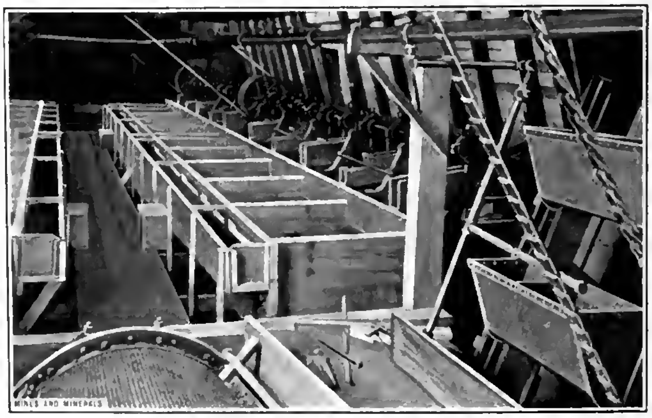


Fig. 4. Jig Room in No. 1 Washery

In some of the larger mines electric motors are employed in the main entries. It is the duty of the miner to load his cars and push them to the entry, where they are hooked on to the train of cars and hauled to the shaft. The underground hauls in some of the large mines are 3 to 4 miles in length, although the average is much less.

The introduction of coal washeries has greatly improved the coal of Southern Kansas. In the summer of 1908, the washery shown to the right in Fig. 2, was completed for the

Central Coal and Coke Co., at Dunkirk, in Crawford County, by the Link-Belt Co. The plant was constructed of creosoted lumber and timbers wherever exposed to water or weather. The contract covered the complete installation, including excavations, foundations, power plant, and a water system comprising supply for operation of the washery and for fire protection. The plant, while built for an average capacity of 125 tons per hour, was at times increased to 200 tons per hour, and even worked overtime to supply the demand for clean coal. By the use of this washery the company's business was increased to such an extent that in 1909 a second washery, shown to the left in Fig. 2, was ordered from the Link-Belt Co., and completed in September, 1910. The introduction of the jig in cleaning the coal has wrought great changes in the past 2 years. The interior of the jig room of the No. 2 washery is shown in Fig. 3, with the draining screen in the background. In Fig. 4 is shown the fine-coal jigs, draining screen, and elevator in No. 1 washery.

In the washer plants jigs are used that are not wholly unlike those employed in the separation of ores in the Joplin district. The jig consists of a rectangular box with hopper bottom, divided about half way down from the top by a partition. In one of the compartments a plunger operates. In the other compartment is fixed a screen plate. The jig tank is filled with water and the pulsating movement of the plunger forces the water up and down through the screen of the other compartment. The coal, in passing over the screen, is held in suspension by the pulsating current, and the lighter material, which comprises the marketable product, floats off of the screens, while the heavier material, which comprises the waste, settles to the screen and is drawn off by means of valves, save enough to form a sufficient bedding of waste material on the screens. Passing through the valves, the refuse falls into a compartment in the bottom of which is a gate through which refuse is drawn out of the hutch.

In the general operation of a fine coal jig, an artificial layer of feldspar is carried on the screen. The coal works its way across the top of the feldspar layer and flows over the end of the jig, while the heavier refuse gradually works down through the feldspar bed and screen into the hopper-shaped jig box, whence it passes out through a refuse sluice by gravity.

In the majority of the Southern Kansas plants the shaker screen system is employed; the coal, on being dumped from the cage, freshly hoisted from the mine, passing through screens of various sized mesh directly into coal cars, awaiting at the bottom of the chutes. That which passes over the 3-inch mesh is known as lump coal; that which passes through the 3-inch mesh, but which is too large to pass through 1-inch mesh, is known as nut, while that which passes through the 1-inch mesh is slack.

Lump and nut coal are now selling for $2 a ton, f. o. b. mine, while mine run brings $1.75, and slack $1.25. No briquetting has yet been undertaken in the Kansas field.

Owing to the unusually warm winter that has prevailed over the Middle West, the demand for coal for domestic purposes has been unusually light, and for this reason the operation of a great majority of the mines has been irregular, the result being that hundreds of men have been without employment some of the time. For commercial purposes, natural gas has taken the place of coal to a large extent in the ore mines of the Missouri-Kansas-Oklahoma district, and the use of gas not only in this ore-mining district, but throughout many of the leading cities of Eastern Kansas, has done more than any other one thing to reduce the demand for coal. However, the epoch of natural gas for commercial usages is waning in the Middle West, especially is this true of the gas produced from the Mid-Continental fields of Southern Kansas; for, although the claim is made by the gas companies that the supply is as great as ever, the shortage is painfully in evidence when the weather becomes severely cold, and under such conditions it is not uncommon for the gas companies to order the discontinuance of gas for all save domestic purposes.

More than a year ago the metal-mine operators of the Joplin district began to realize what it was to receive an immediate order to discontinue the use of gas under boilers. Sometimes weeks would elapse before gas for boilers could be secured. The price of gas per 1,000 cubic feet has been raised to 25 cents, the same as for the domestic consumers. However, with warm weather, the rate has been lowered. It is the uncertainty of the natural gas supply for other than domestic usages that is causing many mine operators, manufacturers, and others to return to the use of coal or to adopt the newer power, electricity, which is beginning to play a conspicuous part in the commercial development of Southwestern Missouri, Southeastern Kansas, and Northeastern Oklahoma.

CATALOGS RECEIVED

In writing for catalogs, please mention Mines and Minerals

The American Asphaltum and Rubber Co. 600-614 Harvester Building, Chicago, Ill., Asphalt Mastic Floors, 20 pages.

American Blower Co., Detroit, Mich., "Sirocco" Mine Fans, 10 pages; folder describing Ventura Disc Ventilating Fans.

Allis-Chalmers Co., Milwaukee, Wis., Bulletin No. 1079, Steam Turbines and Generators, 32 pages; Bulletin No. 1445, Electric Hoists, 28 pages; Bulletin No. 1446, Huntington Mills, 24 pages; Bulletin No. 1521, Allis-Chalmers Air Compressors, 16 pages; Bulletin No. 1624, Centrifugal Pumps—Standard Single Stage, 20 pages; Bulletin No. 1720, Log Machinery, 16 pages.

The Bristol Co., Waterbury, Conn., Bulletin No. 164-A, Bristol's Long-Distance Recording Tachometer, 2 pages; Bulletin No. 147-A, The Bristol-Durand Radii Averaging Instrument for Circular Chart Records, 8 pages.

Buffalo Forge Co., Buffalo, N. Y., "Buffalo" Spray Nozzles and Strainers, 15 pages.

Dempcy-Degener Co., Pittsburg, Pa., Bulletin P, 12-A, Ash Handling Equipment, 8 pages; Bulletin P, 12-C, Locomotive Coaling Stations of the Skip Hoist Class, 8 pages; Bulletin 1-101-C, "Perkins" Pivoted Bucket Carrier, 8 pages.

Ingersoll-Rand Co., 11 Broadway, New York, N. Y., "Little Giant" Rock Drills, 16 pages; "Imperial" Valveless Telescope Feed Hammer Drill (Type "MC-22"), 16 pages.

Keuffel & Esser Co., Hoboken, N. J., Stadia Hand Transit, 4 pages.

The Thew Automatic Shovel Co., Lorain, Ohio, Catalog No. 8, Steam Shovels, 32 pages.

Sullivan Machinery Co., Chicago, Ill., Bulletin 58-J, The Sullivan Straight-Line Air Compressor, 20 pages.

George J. Stocker, St. Louis, Mo., Catalog No. 3, Stocker Cooling Towers, 36 pages.

General Electric Co., Schenectady, N. Y., Bulletin No. 4820, Curve-Drawing Ammeters and Voltmeters—Type CR and CR-2, 6 pages.

Carnegie Steel Co., Pittsburg, Pa., Steel Mine Timbers, Types of Construction and Examples of Installation, 40 pages; Steel Mine Timbers, Data and Tables for the Use of Mining Engineers, 58 pages.

J. R. Robinson, Pittsburg, Pa., The Turbine Fan; Useful Information for Ready Reference for Mine Foremen and Fire Bosses, 20 pages.

Theo. Alteneder & Sons, 945 Ridge Ave., Philadelphia, Pa., Catalog of Drawing Instruments, 109 pages.

Mesta Machine Co., Pittsburg, Pa., Mesta Gas Engine, 16 pages.

Foust Concentrator Co., 322 So. Michigan Ave., Chicago, Ill, Coal Washing, by G. H. Williams, describing the construction and operation of the Foust jig, 12 pages.

A WARNING TO COAL MEN

By Erasmus Haworth, State Geologist, of Kansas

The recent disaster following a gas explosion in mine No. 16, at Mineral, Kans., where five lives were lost on account of an erroneous belief on the part of both operator and miner, compels me to issue a warning to all others similarly situated. I do this in the hope that ultimately lives and property may be saved. We have no state law directly throwing any responsibility on me in this and similar cases, but there is a higher law of human love and human sympathy which in a degree will condemn me if I remain quiet. The explosion occurred about 5 o'clock on Saturday afternoon, March 18, 1911. The day's work had been completed and three shot firers went below to fire the shots. One of them reached the surface with almost no injury, but the other two were killed. As soon as it was known that something had gone wrong the superintendent, Mr. Joplin, with two assistants, rushed below, whereupon another explosion occurred which destroyed their lives.

The Lesson To Be Learned from The Explosion at Mineral, Kansas, in a "Non Gaseous" Mine

Previous to this no explosion had occurred in this mine, nor in this immediate vicinity. It was confidently believed by Superintendent Joplin, and by all the miners, that no gas whatever existed in the mine. The mine floor was unusually wet and muddy, so that they believed there could be no danger from dust explosion. With these beliefs firmly fixed in their minds, apparently no special precaution was used or had been used to guard against explosions. Every one underground carried ordinary lamps, because there was no danger from explosion. Even when Mr. Joplin and his party went to the rescue they did not carry safety lamps, the belief in safety from explosion being so thoroughly fixed in their minds. When I visited the mine a few days later I found an abundance of safety lamps belonging to the company, showing that Mr. Joplin might have provided himself with them had he deemed it necessary, or even advisable.

Mine No. 16 is fairly well developed, and at this time was worked to the north of the shaft. The old and abandoned mine No. 7 lies to the north and northwest, so that shaft No. 7 is from a mile to a mile and one-half northwest of shaft No. 16. From the foot of shaft No. 16 the entry leads 875 feet to the north, then 700 feet east, then 900 feet north. Throughout this last 900 feet two main entries had been driven about 30 feet apart, with only a few doorways cut through the coal between them. Seventeen rooms had been developed on the outside of each of these entries. Two of the shot firers presumably went together to the extreme north end of these two entries, when one of them began firing the shots in the rooms lying east of the east entry and the other one began firing the shots in the rooms to the west of the west entry. If they had worked with equal speed the two should have come out at the south end of these two entries about the same time. It so happened, however, that the shot firer on the east outran his companion, and had fired the shots in all 17 of his rooms before the explosion occurred, while his companion had completed firing only the shots in the north 11 of his 17 rooms.

Now, it seems that the first shot for the east entry broke through into the old workings of mine No. 7, and this is what caused the disaster. It is reasonably sure that explosive gases had accumulated in mine No. 7 during the time it had been idle and that these gases rushed through the opening made by the first shot and mixed with the fresh air in mine No. 16, and that the shot firer, on account of his speed, was able to have traveled this entire 900 feet before the southward migration of the explosive gas overtook him. But finally it did overtake him and was ignited either by a flame from one of the last shots he fired or by the open lamp which he carried.

Mine No. 7, the same as No. 16, was supposed to have been entirely free from gas, and further, was a wet mine, so that there could be no danger from dust explosion. The coal was reached in No. 16 at 60 feet from the surface, while at shaft No. 7 it was 110 feet from the surface. The point of meeting of the two mines near where the explosion occurred was approximately midway between the two shafts. The floor of the old mine No. 7, therefore, rose approximately 25 feet from the foot of the shaft to the wall which was broken through by the shot above described. No. 7 had been abandoned for years, during which time it had filled with water considerably more than enough to form a water joint at the foot of the shaft. In this way the air and gas in the mine southeast of the shaft was carried southward toward mine No. 16. How much pressure would be generated by this process as the water gradually accumulated in mine No. 7 would depend entirely upon the height of the water in shaft No. 7, which is a point of fact I have not yet learned in detail. However, it is well known that the water was a number of feet deep and subsequent events showed that the air and gas were under pressure, probably at least 2 or 3 pounds per square inch above that of the air in mine No. 16.

As time passed after the shot the pressure in the old mine would gradually become reduced by the escape of gas and air into No. 16, and ultimately would become the same in the two mines.

We know first, that with comparative suddenness conditions were changed throughout the northern part of mine No. 16 so that a number of explosions occurred. Mr. Gilday, State Mine Inspector, was called from Pittsburg, and reached the mine about 2 o'clock Saturday night. Immediately he went below and worked his way northward, ultimately reaching the face at the northern extremity, and found the opening which had been broken through into the abandoned mine No. 7. His statement is that at this time, fully 7 hours after the explosion, he found a hole there approximately 8 to 12 inches wide and from 3 to 4 feet long through which air was rushing so rapidly that it was with the greatest difficulty he could stop up the hole by any means at his command. He used canvas and hay and anything he could get hold of in the mine, and finally succeeded in holding these obstructions against the opening and piling masses of coal against them until practically no further exchange of air occurred. He suspected at once that this was the cause of the trouble and made arrangements to clear these northern parallel entries of gas as rapidly as possible by shifting the direction of the air and by increasing the action of the fans.

As late as Tuesday afternoon, March 21, substantially 72 hours after the explosion, some natural gas was still lingering in these entries. A careful examination of the ground showed that the east one of the two entries was swept clean by the onrushing action of the air at the time of the explosion, while the west one showed decidedly less effect of the explosion. The mine inspector reported that during Saturday night and Sunday, from time to time, when in the west one of the two parallel entries, he would force the canvas away a little at a doorway connecting the two entries and would insert his safety lamp a foot or more into the east entry, through which he now had a strong current of air driven by the fans, and that for more than 24 hours after he reached the scene the lamp showed that there was sufficient gas in the east entry to be dangerous. He also found from these and other similar tests that the amount of gas in the east entry gradually diminished until the conditions were reached, as already mentioned, on Tuesday afternoon, the 21st.

It was early tests of this character which determined the mine inspector's actions, to force free air northward into the west entry as rapidly as possible, and carry it entirely to the north wall of the mine, then carry it through to the east entry and force it back south, hoping in this way to carry out of the mine the gases which had entered the east entry in such large quantity. The mine inspector acted with the best of judgment in the way he located the seat and cause of the trouble, and in the methods he employed for righting the difficulty.

During the period between the first explosion and the time Mr. Joplin and his associates reached the south end of the east entry, a fresh amount of gas had migrated southward from its supply store in No. 7 and was mixed with fresh air supplied by the fans to the extent of producing an explosive mixture, so that the lamps which Joplin and his associates carried caused this second explosion.

The fundamental error made by every one connected with mine No. 16 and mine No. 7 is the assumption that no explosive gas would accumulate in either of these two mines. This error is the one which has called for this present report. I wish to emphasize with all the power at my command that nobody has any right ever to assume that dangerous gases will not accumulate in a coal mine. An historical study reveals the fact that "firedamp" or natural gas, is liable to occur in any coal mine.

Any abandoned mine of any nature whatever, which has prop timbers or any organic rubbish, sooner or later becomes dangerous. The danger is vastly greater if the mine is in stratified rocks and in any way connected with coal or carbonaceous shale.

Apply this to old shaft No. 7, a mine which while being worked either gave no gas or so small an amount that the ordinary ventilation kept the mine free from dangerous quantities. This same mine is abandoned and in the course of time is opened up under such peculiar conditions that the air and gas are found to be under extraordinary pressure. Here are two elements of danger—one from gas and the other from the extraordinary pressure under which it exists.

It is recommended that no one ever assume that any coal mine is free from firedamp, or natural gas. It is a most dangerous assumption. A mine may have so small an amount of firedamp in it that ordinary methods of ventilation will prevent its becoming dangerous while ventilation is kept up to a proper degree. Any shot at any time in any mine is liable to break into a gas reservoir and permit a dangerous amount of gas to rush into a mine. This has occurred over and over again in Kansas and elsewhere. But it is vastly more dangerous to assume that an abandoned mine, or an abandoned part of a mine being operated, will remain free from gas. Don't make such an assumption.

It is recommended that all mine operators at once make provisions for relieving the air pressure in all abandoned mines which are at all likely to be broken into by development in adjacent mines. This can be done easily and cheaply by drilling one or more holes from the surface into the part of the abandoned mine where the roof is the highest. Suppose, for example, that a 6-inch hole had been drilled from the surface into the abandoned part of mine No. 7 at Mineral. Then accumulation of water in the mine could not have caused this extra pressure in the air left in the abandoned mine, and, as a result, even though the gas were there it would not have rushed through into mine No. 16 the way it did, and therefore the explosions could not have occurred. Care should be taken to drill these vent holes so that they will tap the abandoned mine at places where the roof is the highest, and a large drill bit should be used, so as to make a large hole, not less than 6 inches, and larger if possible. The drill hole should be cased so it will remain open permanently.

Now, as natural gas is lighter than air, the gas which is likely to accumulate in all abandoned mines will escape first through the drill hole, and later such air will escape as is necessary to prevent an excess pressure should water enter the mine. This is a simple remedy, and I fear on account of its simplicity will not appeal to operators and miners as it should. But I consider it important, as human life and human property are valuable.

All operators and all miners should require a drill hole to be kept in advance of development wherever development approaches an abandoned mine. This has been enacted into law in many states on account of the well-known fact that gases do accumulate in abandoned mines. Had it been obeyed in this case enough gas would have rushed through the little drill hole to give an alarm, and more than likely the explosions would not have occurred.

RAIL BONDING IN MINES

*Written for Mines and Minerals, by Vincent Rhea**

The writer has recently had occasion to test the rail bonding in a number of mines in the Pennsylvania bituminous coal district. The result of these tests, and the methods adopted to remedy the troubles found, should be of interest to every mine manager using electric haulage, particularly those who are not utilizing the full value of their return circuit, and who consequently have an excessive drop in voltage and various motor troubles due to the difference in potential between trolley and rail.

While a few operators are using modern methods of bonding, the majority are depending on channel pins and wire to carry the return circuit. The channel pin when first installed is more or less efficient, but as three-fourths of its contact is between steel pin and steel rail, it is impossible to obtain a union that will exclude air and moisture. Therefore it is only a short time until corrosion has started and a high resistance is introduced at the points of contact. The method of testing prevalent at most mines consists of examining the bond to see that the wire and pins are intact.

The return circuit of a mine that was bonded partly with channel pins and partly with compressed-terminal flexible-cable bonds was tested with a direct-reading bond tester which showed the resistance of each joint as equal to the resistance of a certain length of solid rail. Thirty-one per cent. of the channel-pin bonds showed a resistance equal to or greater than that of 30 feet of rail, or practically an open joint. The average resistance of the balance was equal to that of 13 feet of rail. These channel pins had been installed about 2½ years, and on an exceptionally good roadbed.

The compressed-terminal bonds had been installed 4 years on a roadbed that was in bad condition. The drainage was bad and the soft bed permitted a considerable rising and sinking of each joint when a car passed over, thus imposing a severe strain on the bond terminals; 16 per cent. of these bonds were found defective, the balance showed an average resistance of 6.6 feet. Had these bonds been installed in track similar to that in which the channel pins were used there is no doubt but that the depreciation would have been cut in half.

What particularly impressed the writer in making this test, was the fact that the majority of the compressed-terminal bonds were in good condition after 4 years of service, and under unfavorable and rather uncommon conditions. Had this company tested these bonds at certain intervals, and replaced defective ones as found, they would have had, at a small expense for labor and material, a highly efficient return circuit at all times.

This operation had been suffering from an excessive drop in voltage, and the results of this test proved conclusively that it was caused by a defective transmission of the return current. The channel-pins in this mine are being replaced by compressed-terminal flexible-cable bonds which show an average resistence of 4.5 feet, and the management has decided to discontinue the use of channel pins.

The relative resistances of 21 joints selected at random along the haulage system of this mine, bonded first with channel-pin bonds then with compressed-terminal bonds, are clearly shown in the accompanying curve.

One mine manager reported that, with sufficient copper overhead, he found a drop of 100 volts, at less than 3,000 feet from the generator on a 250-volt direct-current circuit. The

* Mining Department, Electric Service Supplies Co.

channel-pin bonds in this mine were tested, and 90 per cent. of of them found to have a resistance greater than that of 30 feet of rail. The channel pins were replaced by compressed-terminal bonds, the line voltage went up to normal and the efficiency of the locomotive increased to a marked extent.

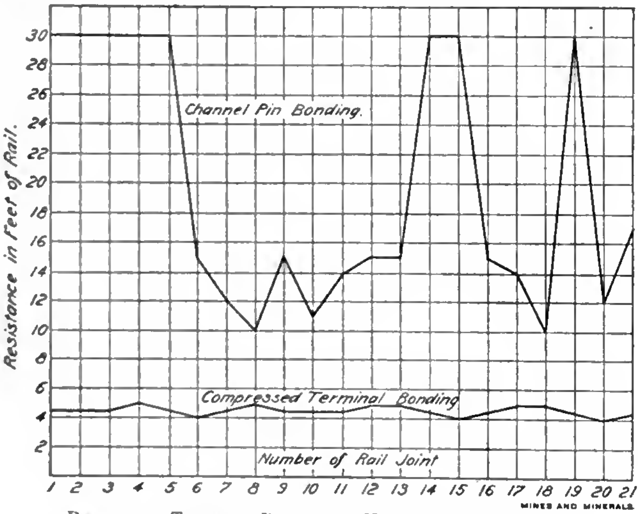
Tests of other operations where low voltage and motor troubles were prevalent showed in every case very defective bonding due to the use of channel pins.

While engineers and electric railway operators fully appreciate that the utilization of the full carrying capacity of the rails for the return circuit is of as great importance as the overhead construction, it is a matter that most mine managers have given little attention.

The bonding of the rails of an electric haulage system is of the utmost importance if the full efficiency of the generating plant and the motors is to be realized.

The compressed-terminal flexible-cable bond properly installed makes an absolutely moisture-proof contact between the rail and bond, and barring mechanical injury should have a long life without any material decrease in efficiency. The flexible cable takes care of the jar and vibration caused by a trip of cars passing over the joint, thereby imposing no undue strain on terminals.

If mine managers, particularly those who are using 250 volts for their haulage, and getting farther away from their source of power every day, will install bonds of this type, and give them the attention their importance deserves, they will find that they have at all times a highly efficient return circuit, and are getting the full value of their equipment, which usually means "increased tonnage."



RESULT OF TESTS OF DIFFERENT KINDS OF RAIL BONDING

LOG SHEET FOR ATTACHED CURVES

Channel-Pin Bonds		Compressed Terminal Bonds	
Joints	Resistance	Joints	Resistance
1	30	1	4.5
2	30	2	4.5
3	30	3	4.5
4	30	4	5.0
5	30	5	4.5
6	15	6	4.0
7	12	7	4.5
8	10	8	5.0
9	15	9	4.5
10	11	10	4.5
11	14	11	4.5
12	15	12	5.0
13	15	13	5.0
14	30	14	4.5
15	30	15	4.0
16	15	16	4.5
17	14	17	5.0
18	10	18	5.0
19	30	19	4.5
20	12	20	4.0
21	17	21	4.5

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals by J. T. Beard

Hoisting Engineers' Examination, Held at Des Moines, Iowa, December 6, 7, 1910

QUES. 1.—State your experience with engines and boilers around Iowa mines.

QUES. 2.—(*a*) What is a unit of heat? (*b*) What heat unit is commonly used in the United States?

ANS.—(*a*) A unit of heat is such a quantity of heat as is required to raise the temperature of a unit weight of pure water, at its maximum density, 1 degree of a specified scale. (*b*) The heat unit commonly employed in the United States is the British thermal unit (B. T. U.), and is the quantity of heat required to raise the temperature of 1 pound of pure water, at 39.1° F., 1 degree of the Fahrenheit scale.

QUES. 3.—How far must a 75-pound weight be placed from the fulcrum of a safety valve that has an area of 4 square inches, the valve stem being 3 inches from the fulcrum, so that the steam will blow off at a pressure of 100 pounds per square inch?

ANS.—Disregarding the weight of the valve, valve stem, and lever, which is not given, the required distance is

$$x = \frac{3(100 \times 4)}{75} = 16 \text{ in.}$$

QUES. 4.—Give a complete code of hoisting signals.

ANS.—A code of hoisting signals adopted for use at mines is as follows:

1 bell, *hoist* when cage is at rest; or
stop when cage is in motion.

2 bells, lower cage.

3 bells, men want to be hoisted. (It is the duty of the topman when this signal is received to give an empty cage and return a signal of 1 bell to the bottom and at the same time ring a jingle bell in the engine room. The engineer should then notify the bottom man by a blast of the whistle or by a signal of 2 bells, that he is ready to hoist men. The men are then allowed to get on the cage, and when all is clear the bottom man gives 1 bell to hoist. The engineer only moves the cage in response to the signal of the bottom man (the topman signals the engineer with a jingle bell). The bottom man must never signal the engineer, in any case, till he has received 1 bell from the topman signifying that all is clear at the top. If the topman cannot hear the engine-room bell, or fails to respond promptly to signals from the bottom, the engineer may repeat the signal by sounding the whistle.

4 bells, stop at lower landing; followed by the usual signal of 1 bell to hoist.

5 bells, hoist or lower slowly and with extra caution, followed by usual signal to hoist or lower, as desired.

6 bells, change cages.

QUES. 5.—Define the following terms: Combustion; dead center; lead; tensile strength; factor of safety.

ANS.—Combustion is any chemical reaction causing a change in the physical condition or state of a substance and accompanied with the evolution of heat and often producing light or flame, or both. The most common form of combustion is oxidation. Dead center is a term used to describe those positions of the crank-arm of an engine in which the piston can exert no turning moment on the crank-shaft. Lead, in engine practice, is the amount the port is open when the crank is on dead center. Tensile strength is the strength of any given material to resist a pull or extending force, per square unit of cross-section. Factor of safety is the ratio of the ultimate or breaking strength of any material to the allowed working strength of that material; or the ratio of the breaking load to the allowable safe load of any structure or member.

QUES. 6.—Find the indicated horsepower of a single-cylinder engine having a piston 10 inches in diameter and a stroke of 12 inches when the crank makes 120 revolutions per minute

and the mean-effective steam pressure is 60 pounds per square inch.

ANS.—

$$\text{I. H. P.} = \frac{60(.7854\times10^2)12\times2(120)}{33,000\times12} = 34.27, \text{ say } 35 \text{ H. P.}$$

QUES. 7.—Describe the proper method of lining the crank-shaft of an engine.

ANS.—The method to be employed will depend on circumstances. What is to be determined is that the axis of the crank-shaft, in a horizontal engine, is level and set at right angles to and in the same plane with the center line of the cylinder. It will not be possible to apply the same method to all types or styles of engines. Assuming that the crank-shaft is in place, its horizontality may be tested by an ordinary spirit level, or by suspending a fine plumb-line from the end of the crankpin in its uppermost position and marking the exact position of the point of the bob or of the line below, and checking by repeating the operation when the crankpin has been brought to its lowermost position. If the two marks exactly correspond, the axis of the shaft is horizontal. To ascertain if the crank-shaft is square with the center line of the cylinder it is often better, or even necessary, to take off the cylinder heads and remove the piston and the crosshead and stretch a fine wire taut through the exact center of the cylinder, adjusting the line till its true position is obtained. Measurements from this line to the face of the crank-arm at the two dead centers will determine if the shaft is square with this line or not; if it is square with the line the two measurements will be exactly the same. By means of a steel square placed against the face of the crank-arm or disk in a vertical position it can be shown whether the axis of the shaft lies in the same plane with the center line of the cylinder.

QUES. 8.—What steam pressure should be allowed in a boiler 60 inches in diameter, made of $\frac{3}{8}$-inch steel plate having a tensile strength of 65,000 pounds per square inch when double riveted; using a factor of safety of 5?

ANS.—A safe rule in common use in boiler practice, is to allow 70 per cent. as the efficiency of a double-riveted longitudinal joint. This gives for the safe allowable pressure in this case,

$$p = \frac{.70\times65,000\times\frac{3}{8}}{30\times5} = 113.75 \text{ lb. per sq. in.}$$

QUES. 9.—How much farther does the crankpin of an engine travel than the piston when the stroke is 36 inches?

ANS.—

$$3.1416\times36 - 2\times36 = 41.0976 \text{ in. each rev.}$$

QUES. 10.—Under what conditions is a second-motion engine to be preferred to one that is coupled direct?

ANS.—In hoisting from shallow shafts where the speed of hoisting does not exceed, say 800 feet per minute, and in sinking operations, and all haulage engines for operating rope haulage, geared or second-motion engines are generally preferred.

QUES. 11.—How would you set and brick in a boiler to allow all its parts to move under a change of temperature?

ANS.—The best method is to hang the boiler from two independent gallows-frames, each consisting of two upright channel bars or legs supporting a cross-bar or **I** beam. The channel bars are built into the brick walls enclosing the boiler, and asbestos packing is used to close the space between the boiler and sidewalls, while a slip-plate attached to the rear end of the boiler and resting on the end wall closes the flue at this point and allows for the free expansion of the boiler.

QUES. 12.—What is saturated steam?

ANS.—Steam that is at the point of condensation, or that will deposit moisture on the slightest reduction of temperature

QUES. 13.—What is one of the most marked characteristics of steam?

ANS.—Its expansive force.

QUES. 14.—In taking up the wear of the connecting-rod brasses what must be done to keep the proper length of the rod?

ANS.—A shim of this metal must be placed behind the brass on the opposite side of the pin from the wedge. The thickness of the shim must be equal to one-half the wear, the wedge being driven in to take up the other half.

QUES. 15.—What are the sources of loss of heat in a steam engine?

ANS.—Condensation of steam, and radiation and conduction of heat.

QUES. 16.—What is a steam jacket?

ANS.—A steam space or chamber surrounding another chamber or receptacle, having for its purpose the retention, as far as possible, of the heat of the inner vessel.

QUES. 17.—How may loss of heat in the exhaust of an engine be lessened?

ANS.—(1) By conducting the exhaust steam into a feed-water heater and thus utilizing its heat. (2) By exhausting at a lower pressure, either by use of a condenser or by increasing the expansion of steam in the cylinder by an earlier cut-off.

QUES. 18.—What is back pressure?

ANS.—By back pressure, in steam-engine practice, is meant the pressure under which exhaust takes place from the cylinder, and caused by the resistance of the steam ports and valve to the escaping steam; or the steam pressure that resists the forward movement of the piston.

QUES. 19.—The piston speed of an engine 420 (feet per minute), the number of revolutions 110; what is the length of stroke?

ANS.— $$\frac{420\times12}{2\times110} = 22.9+ \text{ in.}$$

QUES. 20.—If the length of stroke of an engine is 2 feet and the piston speed 800 feet per minute, what is the number of revolutions per minute?

ANS.— $$\frac{800}{2\times2} = 200 \text{ rev. per min}$$

BANNER MINE EXPLOSION

Robert A. Neill, Associate Mine Inspector, reported to the Governor of the state of Alabama on April 20, his investigations and conclusions in regard to the explosion which occurred at the Banner mine on the morning of April 8, at 6:35 A. M., when 128 men, mostly convicts, lost their lives. From this official report we deduce the following:

The mine is operated by the Pratt Consolidated Coal Co., and is situated in the extreme western part of Jefferson County, about 20 miles from Birmingham, on the Cane Creek Division of the Louisville & Nashville Railroad. The Banner mine operates on the Big Seam and is 140 feet deep. It gives employment to 285 men, 250 of whom are inside miners, and 35 work on the outside.

The capacity of the mine is approximately 1,100 tons of coal daily. The mine has three openings, known, respectively, as the slope, No. 1 shaft, and No. 2 shaft. At No. 1 shaft there is a 17-foot diameter Crawford & McCrimmon fan used only in case of emergency. At No. 2 shaft there is a 20-foot diameter fan of the same make, which furnishes from 180,000 to 200,000 cubic feet of air per minute. This furnishes about 800 cubic feet of air per minute to each man, and to overcome the extraction of moisture from the mine by the air, 96 water sprays were distributed throughout the mine at intervals of 100 feet apart, each delivering 30 gallons of water per hour in the form of vapor. This vapor is carried throughout the mine by the air-current and deposited on the walls, roof, gobs, and floor of the mine.

The headings in this mine are driven 7 feet high by 18 feet wide, carrying with them a gob of 9 feet by 5 feet, which leaves a sectional area of from 75 to 85 feet for haulage and ventilation. The air courses have the same sectional area as the headings. The split system of ventilation is used throughout. All overcasts are built of concrete, the floors being 2 feet in thick-

ness and the arch from 6 to 8 inches in thickness. All brattices are built of stone and cement 18 inches in thickness. No doors are used in the ventilating system, thus making it impossible to interfere with the ventilation at any part of the mine by leaving doors open, which would permit the air to short-circuit. So long as the ventilating fan was kept in operation the ventilation could not well be interfered with and the automatic fan recorder shows the fans to have been in operation regularly and continuously up until the minute of the explosion.

In this mine all coal was undercut by electric machines before blasting was done. As is the usual case in machine mining, considerable dust is made, and the air in the immediate neighborhood of the machines was very dusty when they were in operation. To meet this condition, in addition to the moistening appliances, all dust not in suspension in the air was loaded in cars and sent out of the mine before any blasting was done in places where machines were at work. It was the custom for a free man to have under his charge three or four convicts. This man had control of two working places, in order that loading in one place could be done while the machines were undercutting in the other. The blasting was done by shot firers, each heading having separate shot firers who prepared the charges and fired them while the miners were in the mine.

The explosive used was Bituminite No. 1, a permissible explosive that was carried into the mine daily in quantities to do a day's work. This was distributed so that each entry had its proportionate supply. But the quantity necessary for daily use in each heading was kept together so that from 25 to 100 sticks were in each place of storage. Bituminite is a permissible powder, which successfully passed the test of the United States Testing Station. Its base is nitroglycerine, and its composition is such that a spark from a mine lamp or other source might set it off. The Banner mine generates gas, but not in large quantities, as is shown by the daily reports of the fire bosses; and the large quantity of air circulated would indicate that gas was not the agent which extended the explosion, even if the possible ignition of a pocket of gas might have originated it.

The force of the explosion up to the air-shaft failed to show evidence of heat, that is, scorching on the shaft timbers, as in the case of Virginia, Mulga, and Palos mine explosions. At each of the latter places flames shot out of the mines and the explosive gas was found when recovering the dead bodies.

There were 50 men who escaped the firedamp in the Banner mine. The machine men and their helpers and the bosses had not yet entered the mine, with the exception of O. W. Spradling, under foreman, and Lee Jones, a shot firer, both of whom were killed.

After a thorough examination of the mine, Mr. Neill arrived at the following conclusion as to its origin and cause:

"The disaster originated by a premature explosion of Bituminite at the magazine located in the cross-cut between the entry and air-course about 21 feet inside of room No. 17 on the seventh left heading. I am led to this conclusion because of the fact that the force of the explosion was divided here, traveling in the direction of the heading face and also in the outer direction toward the slope."

That some Bituminite exploded at this plant is evidenced by the fact that the records of the mine officials show that 160 sticks of Bituminite should have been found in this locality, made up of 40 sticks left over from the previous shift, 20 shots made up ready for use, and 100 sticks carried down that morning by a shot firer. The body of one of the shot firers for the seventh left entry was found in the near neighborhood of the magazine badly mutilated, and there was evidence of greater force and heat in this entry than in any other.

The evidence, as shown by coke deposited on the timbers, cars, walls, etc., leads to the conclusion that the premature explosion of the Bituminite occurring in the magazine in the seventh left entry aided by dust was the direct cause of the disaster.

Inspector Dickerson, who also examined into the accident, in his report does not agree with Inspector Neill as to the initial cause. While Inspector Neill attributes the initial cause to the explosion of the Bituminite, Inspector Dickerson says: "I examined the place carefully, but failed to find any indication of any powder having been exploded, either there or elsewhere in the mine." He bases this conclusion on the fact that if such a quantity as 160 sticks of Bituminite had exploded there would have been an indentation in the floor of the room in which the explosion occurred. While the reports of these two capable men differ as to the probable cause, the conditions existing previous to the accident favor the conclusions of Inspector Neill.

In the first place the mine was well laid out and the ventilating current was sufficient to dilute and carry off the comparatively small amount of gas evolved, especially from the live workings. The accumulation of 160 sticks of Bituminite at one place, 100 sticks of which was being taken in by one man, was in itself a menace to safety. A spark from the shot firers' lamp or from some other cause could easily ignite it, and while it was a permissible powder in the meaning of that term as used by the United States Testing Station, it would produce such a stirring up of dust and so much flame and heat that the dust could easily carry the explosion to distant parts of the mine, especially as the practice of driving the headings 18 feet wide and gobbing 9 feet of this width made a great receptacle for dust, which could not be effectively dampened, even by the extensive vaporizing system used. Had it not been for dry dust, such as would naturally be thick in the gob, and probably in other comparatively inacessible places, the initial explosion would have probably affected only a limited area of the mine. The practice of taking in the explosive, as far as limiting the amount to the quantity required for each shift, was all right as far as it went, but the practice of storing all required for each heading in one place was radically wrong, and it is a great wonder that the supply stored in each of the other headings did not explode and make the damage to the mine, at least, far greater than was the case. Familiarity through constant use of explosives frequently makes men careless in handling them, and it is possible that the shot firer who had with him 100 sticks of Bituminite was not as careful as he should have been, and he not only lost his own life, but caused the death of 127 others. That the explosion did not extend through all the workings instead of through only a portion of the left heading was undoubtedly due to the vaporizing system keeping the dust in the main haulage road damp enough to prevent the flame carrying into the right headings to any serious extent.

TRADE NOTICES

Telephones in Mines.—The Western Electric Co., of New York, among its recent installations of mine telephones, has equipped the mine of the Springfield Coal Co , at Springfield, Ill., with a system of eight telephones of the ironclad type, which were specially built for mine service. One telephone is located in the engine room on the surface, one at the foot of the shaft, and the other six at convenient points in the mine about 1,000 feet apart. The chief use of the installation is in the systematic operation of the mine haulage and the direction of the underground work generally. In addition to the convenience and facilities for expediting work, this system is a great safeguard as a means for notifying the officials and men of any stoppage in work, accident, or possible danger that may necessitate the speedy withdrawal of the men from the mine.

Electric Mine Hoist.—The St. Clair Coal Co., at St. Clair, Pa., has just installed an electric hoist built by the S. Flory Mfg. Co., of Bangor, Pa. This hoist is equipped with a drum 6 feet in diameter by 5 feet 6 inches between the flanges. A special

feature is the application of the Werner patent band friction, which is operated by an electrical air compressor equipment. When the friction is thrown in contact, it will remain in position until released. The drum and gear form a unit and can be operated as a reverse-motion hoist without any end strain on the bearing. After the friction is released, the drum is loose on the shaft and will revolve freely at high speed when lowering or paying out rope. There are a number of new and advantageous features embodied in the Flory electrical hoists, which are constructed with single drums or two drums placed either side by side or tandem.

Coke-Oven Equipment.—The Mining and Coking Equipment Co., with main offices in Pittsburg, Pa., has recently been incorporated under the laws of Pennsylvania. The company will, beside manufacturing and selling the Beutlich coke oven door and other equipment, design special requirements of the mining and coking industry. The officers are A. F. Ehrenhaft, president, and Richard F. Beutlich, secretary and treasurer.

President of Westinghouse Company.—John F. Wallace, formerly chief engineer of the Panama Canal, who retired after inaugurating the American work on the canal, has assumed active charge as president of the Westinghouse, Church, Kerr & Co., who were the engineers for the new Pennsylvania station in New York. Mr. Wallace also acted as railway expert for the reorganization committee of the Seaboard Air Line, which was reorganized on the basis of his report and the prospertiy of this railroad since its reorganization has more than justified his conclusions and predictions as to its earning capacity. Mr. Wallace succeeds Mr. H. H. Westinghouse.

New Office.—The Mineralized Rubber Co. announce the removal of their offices to 165 Chambers Street, New York City.

Memorandum Books.—Chas. C. Moore & Co., engineers, San Francisco, Cal., are sending engineers neat perforated memorandum books worth having. Send for one.

The Weber Subterranean Pump is a distinct advance in the line of simple and effective pump mechanisms. It is not only effective, but automatic in action. It is operated by compressed air and can be adjusted for starting and stopping at predetermined water levels in the sump when used for mine drainage; or when in use for deep well pumping it can be adjusted to stop when the water in the storage tank reaches a predetermined height. A strong feature of the pump is in the economy of power used in its operation. The height to which it will pump water is dependent on the pressure of the compressed air; with a pressure of 125 pounds it will theoretically pump about 300 feet, but the manufacturers, to be on the safe side, advise 200 feet, although, by compounding, it can be made to pump to any desired height. An attractive circular, illustrated with sectional views, showing the operation of the pump, is issued by the manufacturers, the Weber Subterranean Pump Co., 90 West Street, New York City, and is worth the attention of mine managers generally.

Electric Mine Lamp.—The use of incandescent electric lamps by mine officials and miners has long been considered to be a great advantage in mining operations, both as regards convenience and safety, provided such a lamp, with its supply of electricity, could be practically self-contained and capable of being used in any part of the mine, and be as portable and convenient to move around as an ordinary mine lamp. This desirable feature has been attained in the Pilley electric mine lamp which can be worn on the cap or hooked to any part of the clothing. In addition to furnishing a brilliant light, without smoke and without flame coming in contact with the air, it is economical in operation. The battery furnishing electricity for the lamp is simple and is armor clad, so that it cannot be broken or damaged. It can be carried in the pocket or fastened to the belt without in the least interfering with the free movements of the workman wearing the lamp. A number of mine managers have already commended this lamp in the highest terms after giving it a thorough trial. Full information as to the use, cost of operation, and maintenance of Pilley lamps, will be sent to readers of MINES AND MINERALS who will write to the Westfalia Engineering Co., 42 Broadway, New York City, or the Pilley Mfg. Co., 608 South Third Street, St. Louis, Mo.

Electric Plant of Victor-American Fuel Co.—The Victor-American Fuel Co., which owns one of the most up-to-date coal-mining plants in the Southwest, and operates steam plants at three of its coal properties near Gallup, N. Mex., is installing a central turbine plant to transmit power from this to its other properties. The plant consists of a fireproof brick power house, with steel-supported roof in which are installed one 300-horsepower water-tube boiler, and three 100-horsepower marine boilers with Illinois stokers, space being left for additional boilers to be installed later. Steel coal bunkers receive coal from a tipple about 200 feet away. The ashes from each boiler fire-pot are emptied into a special car underneath the boilers and these cars are run out and the ashes emptied into a pit from which they are conveyed by an aerial tramway to an ash dumping ground some 2,000 feet away.

In the generator room, the present installation consists of two ATB 500-kilowatt, 2,300-volt, three-phase, General Electric, horizontal, condensing steam turbines, operating at a speed of 3,600 revolutions per minute and a frequency of 60 cycles, space being left for two additional units. The exciters will consist of two continuous-current, 15-kilowatt, 125-volt, General Electric condensing turbines operating at a speed of 4,500 revolutions per minute.

The switchboard was purchased from the General Electric Co., and will have eight panels for the necessary connections and all necessary instruments including wattmeters, synchronism indicator and voltage regulator.

There are at present four feeder circuits radiating from the power house. One circuit goes to the Heaton mine, about 1¼ miles distant, where about 300 kilowatts in motors are installed; another runs in the opposite direction to the company's Navajo mine, where it has approximately 600 kilowatts in motors; the third circuit goes to the Bartlett mine, about 1½ miles away, where 200 kilowatts in motors are installed; the fourth circuit goes to the town of Gallup, 5 or 6 miles away, where the electricity will be used for lighting and power. A voltage of 6,600 will be used for transmission as some of the developments are 4 to 6 miles from the plant. All the circuits are protected by General Electric aluminum cell lightning arresters. The necessary transformers and motors for use at the various plants for hoisting, pumping, and driving fans, have also been ordered.

WASHED HIS HOME FOR GOLD

The *Cape Town Argus* says a man named Whalen bought a few acres of ground near Ballarat with his wife's savings. From the mud and gravel of a sluggish spring he made sun-dried bricks and built a cabin, in which he started a bar for miners. Quite contrary to their usual habits, a colony of Chinamen commenced to visit his bar every night. Then Mrs. Whalen discovered that some one had, bit by bit, carried off the mud pigstye and its surrounding wall so gradually that it had almost gone before she noticed it. Soon the chimney and the cabin walls also began to vanish. After a careful watch Mrs. Whalen discovered that while one band of Chinamen kept her husband busy in the bar another band was stealing the chimneys and walls.

Whalen, acting on his wife's suggestion, also "stole a pan of dirt" from his own chimney and washed it. Then he ordered tents to live in and washed away the entire house. It was literally built of gold bricks. After that the pool and the spring were also attacked, and the result was a big fortune for the Irishman and his cute little wife.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

WESTERN EDITORIAL OFFICE: 728 Equitable Building, Denver, Colo.
CHICAGO OFFICE: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to MINES AND MINERALS, Scranton, Pa.

CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER .... MANAGER
EUGENE B. WILSON, SCRANTON, PA .... EDITOR
GEORGE F. DUCK, E. M., DENVER, COLO .... WESTERN EDITOR
P. G. MOORE .... CIRCULATION MANAGER
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., LTD., DRYDEN HOUSE, 43 GERRARD ST., SOHO, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 PITT STREET, SIDNEY.
SOUTH AFRICAN AGENT: E. HERBERT BYSSHE, CAPE TOWN, MANSION HOUSE; JOHANNESBURG, TRUST BUILDING; DURBAN, NATAL BANK CHAMBERS; BLOEMFONTEIN, FLORENCE BUILDINGS.

SUBSCRIPTION RATES

Single Copies .... 25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines $2.00
To Canada 2.50
To all other countries 3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office.

When notifying us of a change of address always give your FORMER as well as your present address.

TABLE OF CONTENTS

BUFFALO-SUSQUEHANNA SAGAMORE MINE, by R. Dawson Norris Hall 645
COAL NOTES 649
CARE OF MINE MULES 650
OBITUARY—
Samuel Calvin 652
THE COAL MINES OF DAWSON, N. MEX., by Jo. E. Sheridan 653
THE COKEDALE, COLO., EXPLOSION, by Geo. F. Duck 658
PUMPING AND SIPHONING HOT WATER, by J. T. Beard 663
INTERLOCKING STEEL MINE SUPPORTS 664
REPORT OF COAL MINE INSPECTOR OF COLORADO, by Geo. F. Duck 665
BRITISH COAL MINING NOTES 666
PERSONALS 667
SOUTHERN KANSAS COAL DISTRICT, by Lucius L. Wittich 668
CATALOGS RECEIVED 671
A WARNING TO COAL MEN, by Erasmus Haworth 672
RAIL BONDING IN MINES, by Vincent Rhea 673
EXAMINATION QUESTIONS—
Hoisting Engineers, Des Moines, Iowa, December, 1910 674
BANNER MINE EXPLOSION 675
TRADE NOTICES 676
EDITORIAL—
Lessons From the Price-Pancoast Fire—Convict Labor in Alabama Mines 678
BOOK REVIEW 680
CORRESPONDENCE—
Filter Presses—Determination of Meridian—Rolling Friction 682
THE CHEMIST'S PLACE IN MINING 683
SCIENTIFIC STUDY OF COPPER DEPOSITS, by A. J. Sale 684
IDAHO'S METAL PRODUCTION IN 1910 686
THE VALUE OF SURFACE TRENCHING 686
SURFACE ALTERATIONS OF GOLD ORES, by Albert D. Brokaw 687
THE ALCHEMIST IN MODERN INDUSTRY 688
CHEMISTRY OF THE CYANIDE PROCESS, by W. H. Seamon 689
STANDARDIZATION OF BULLION ASSAYS 690
THE CROWN MINES, LIMITED 691
THE A B C OF ELECTRICITY IN MINES 692
PHOSPHORUS POISONING 693
TREATMENT OF COBALT ORES 693
PROSPECTING POVERTY GULCH CLAIMS, by Charles W. Henderson 694
DRIFTING WITH A STOPING DRILL, by H. E. Moon 697
BLAIR'S ZINC-LEAD PIGMENT PLANT, by J. J. Blair 698
METAL MINING FATALITIES IN IDAHO 700
THE PORCUPINE GOLD FIELDS, by Ralph E. Meyer 701
ELECTRIC FURNACE FOR ZINC SMELTING, by Francis A. FitzGerald 703
COBALT ORE BUYING BY SMELTERS 705
ORE MINE NOTES 706
SOCIETY MEETINGS 708
NEW INVENTIONS 708
SQUIBS—
To Determine Electrical Horsepower, 652—Obstacles to Development of Alaskan Coals, 657—Library of the Engineering Societies, 652—Sheelite in Nova Scotia, 665—Washed His Home for Gold, 677—Rules for Handling Kanite, 681—Disastrous Air Explosions, 683—Copper Refining, 690—Tayabas Petroleum, 697—Coaling Station in Southern Pacific, 700—Metallic Strontium, 703—Examination of Steels by Corrosion, 703—Sodium Carbonate as an Oxidizer, 704—The Menne-Oxygen Melting Process 708

LESSONS FROM THE PRICE-PANCOAST FIRE

THOSE who follow coal mining have undoubtedly noticed after each mine disaster that some one in commenting on the means of escape will say had the men taken such and such route they might have escaped. In a few instances men who took roundabout routes have gotten out, and then the public wonders why all did not follow their example. Of the possible reasons that might be advanced to explain this apparent neglect on the part of the dead in the recent Price-Pancoast fire three are cited because of their prominence and are:

That the men evidently were unacquainted with the mine and the second means of exit.

That too much dependence was placed by miners on the mine management; and

That the expression "All over" was not sufficient to warn men of impending danger.

During the excitement over the deaths of the 73 men in the Pancoast mine the impression prevailed that there was but one exit from the China bed, the public in the meantime overlooking the fact that those who escaped did so through a second exit. It would seem therefore that the men who perished did not know the way out of the mine through the other openings; and this is so evident to mine managers that two large coal companies are placing signboards in their mines on which are painted arrows pointing to the way of exit. One of the first things a wise man does when the goes to work in a strange place is to learn his way about; and if this be wise on the surface, it would seem to be infinitely more so in a mine. While the use of signboards is commendable, assume that the men's lights are out, which of course precludes any observation of the arrow, and this immediately suggests some token that can be made use of in the dark; however, the best guide post is a good general knowledge of the mine. John Gall with his knowledge of the Pancoast mine was able to lead 15 men through the dark to safety, and it is suggested that at least once a week the miners in a body be led through the various ways of escape; in fact, given a sort of fire drill.

One clause in the verdict of the Pancoast coroner's jury is the statement that "too much attention was given to fighting the fire and not enough to getting the men out." While this was uncalled for, in the face of the abundant evidence produced to show that the men were warned in time, nevertheless it suggests that miners place too much reliance on the mine management caring for them. In this particular case some men who escaped were warned twice to leave, one man who left immediately on being warned went home and did not learn of the seriousness of the fire until late that afternoon when he read the newspaper account. There is no doubt but that a considerable period elapsed between the time wood smoke from the fire was wafted through the mine and the time the cars caught fire, which makes it evident that the men placed too much

dependence on the mine management and failed to make use of nature's warning. That the men rely on the judgment of the mine officials should show the public that the latter are using every known precaution to protect those under them; nevertheless the miners should use their heads and not rely on any one for protection.

Once at least previous to the Pancoast fire the phrase "*all over*" (meaning no more work for the day) has led to loss of life because men did not take it as a warning. As an instance at the Pancoast mine, one miner who escaped heard the first cry of "all over," but having two cars proposed to load them before leaving the mine. The second alarm which was couched in proper language caused him to hustle from the danger zone.

Whenever there is any reason to call men out of the mine because of impending danger, "all over" should be omitted and some danger signal substituted.

CONVICT LABOR IN ALABAMA COAL MINES

THE editorial entitled "Legalized Slavery," which appeared in our issue January, was one that was not based on fact. It was not, however, inspired by intention of misstatement on the part of the editor. He was misled by a misrepresentation in connection with a test case involving the contract labor law of Alabama recently decided by the United States Supreme Court, and the methods and rate of payment for the work of the convicts.

The statement that no white men are working as convict miners was untrue, because about 10 per cent. of such convicts are white. This seems a small proportion, but it must be remembered that in that class of population in Alabama from which criminals come the negro race largely outnumbers the white race. The statement that the convict labor law was enacted to ensnare negro laborers through a scheme of forcing ignorant negroes to enter into a labor contract, and then for some alleged violation of the contract drag him into court and have him convicted of a misdemeanor to be punished by hard labor and a long sentence, is at variance with facts. Out of 1,350 convicts working in the mines of Alabama on January 1, the records show that but 10 were convicted of violation of labor contracts, and two of these 10 were charged with other misdemeanors as well.

The statement that the laws of Alabama require the operator to pay the miner $1 per ton for coal mined is so ridiculous that we feel humiliated that it should appear in our columns. No state in the Union fixes the price of mining by legal statute. The price is fixed by agreement, and, as in other states, it is regulated by the law of supply and demand.

The statement that ignorant men from farms are sent into gaseous mines, and whipped if they do not dig their allotment of coal, and that they are shot if they attempt to escape, is also at variance with facts.

At the various mines where convicts are worked, from 50 per cent. to 75 per cent. are actually doing hand mining. The balance are running mining machines, mine locomotives, firing boilers, working around the tipples, or in the blacksmith shop, or such other labor inside and outside the mines as they are capable of performing. A considerable number work in the gardens which produce the vegetables on which the convicts are fed. They all work exactly the same number of hours per day as do free workingmen. All extra work, such as work on Sundays, overtime, or work on holidays is optional with the convicts, and they are paid for it in cash at the same rate as free labor, less the cost of maintenance. The money thus earned is paid them on each regular pay day and may be sent by the men to their families and disposed of in any proper manner. It is stated that of the 1,350 convicts working in and about the mines on January 1 of this year, about 67 per cent. earned extra money, and this extra money for the month of January amounted to about $4,100, or an average of $4.60 per man. The convicts who are mining coal or operating mining machines are given a certain task to do, based on their physical condition. This task is not fixed by the operator, but by the State Prison Physician. The men are divided into four physical classes, and their tasks are fixed according to their classification. Every few months the State Physician reduces the men's classification for the purpose of breaking the monotony of the regular task.

There is a physician in constant attendance at the prison, and, if from sickness or any other cause a convict is not in proper shape for work in the morning, he can, by applying to the physician, be excused from work or his task reduced.

There are well equipped hospitals at all the prisons, the sleeping wards in the barracks are large, commodious, well ventilated, and very sanitary. Each convict is required to take a bath and put on clean clothing every evening as he comes from his work, and it is claimed that 90 per cent. of the convicts are in better physical condition when freed than they were when they were put in prison.

Very little corporal punishment is necessary, and when it is inflicted it is on very vicious and unruly fellows. The statement that they are shot if they attempt to escape is true in the same sense as would be a similar statement made regarding convicts in the Pennsylvania or New York penitentiaries.

The statement that some have tried to commit suicide, and that others have run the risk of being killed by the guards rather than endure their prison life needs but one comment. In every prison the world over, prisoners try to commit suicide and others take great risks in endeavoring to escape, no matter how lenient the prison regime may be.

The above information refutes completely the mis-

statements in our January issue, and we cheerfully publish them as an act of justice to the state of Alabama and the operators of the coal mines in that state.

Having done this we have a perfect right to condemn the convict labor practice of Alabama on fair and rational grounds. In doing this we do not feel that the operators who employ such labor are the persons to blame. They need labor and if they did not use this cheap labor their competitors would. The fault is with the people of Alabama as represented in their Executive and Legislature.

There is no question but that convicts should be made to work, and at least earn their keep and the expense of maintaining the penal institutions, if the products of their labor do not enter into competition with those of free labor. We do not believe that they should be employed as coal workers as they are in Alabama. Their employment as such necessarily keeps out of the mines just that number of free men, both black and white. Besides, the practice is one in which if the prison contractor and mine owner happen to be unscrupulous, there is room for very grave abuses affecting not only the convicts, but the owners and employes of mines employing free labor only. The writer has always held the opinion that every convict should be made to work for two reasons. One for the salutary effect of occupation on the mind, and the other because the majority of convicts are such because of their abhorrence of honest labor, and to make them earn at least a part of their keep does them good in a corrective way.

There is a kind of work they can do, and it is work that interferes least with free labor. They can crack stone in the jail or prison yards, and this cracked stone can be effectually used to make, repair, and keep in order the public roads. It is true stone crushers can do this cheaper than it can be done by hand, but the hands that are employed when convicts do it, cost nothing but their keep, and this is usually as cheap as it can be made consistent with health and sanitation. If each state would put its prisoners to such use, it would materially reduce the just complaints against our abominable roads; and, besides, the privileges and rights of free honest labor would be interfered with less than by any other work.

BOOK REVIEW

Guide to Technical Writing, by T. A. Rickard, editor of the *Mining Magazine*. Published by Mining and Scientific Press, San Francisco, Cal. The book contains 168 8mo pages embodying the criticisms, pleas, and suggestions of an earnest man for better mining and metallurgical writing. Several years ago Mr. Rickard sent the writer a pamphlet accompanied by a letter in which he suggested that uniformity in expression and style be adopted. The United States Printing Office, the United States Geological Survey, The International Textbook Co., and others, have adopted "style sheets" that contain rules for editors, writers, proof readers and printers, but this is not what Mr. Rickard had in mind, for an author could conform to the rules yet the subject matter be written poorly. A plea for "Greater Simplicity in the Language of Science" was another paper by Mr. Rickard and was read before the American Association for the Advancement of Science. By this time Mr. Rickard had come to the conclusion that to disseminate mining and metallurgical information where it would do the most good, the literature must be stripped of pedantry. He commenced by telling the pedants, in the parlancy of Wall Street, that while they were long on orthography and etymology, they were short on syntax. While Mr. Rickard's criticisms pointed out the idiosyncrasy in others, they were not always received in the right spirit, for which reason he made a few enemies and a host of friends. Being in sympathy with Mr. Rickard, it is difficult to criticise his "Guide to Technical Writing," for when following the argument interestedly, carelessness like that on page 24, where *then* is used for *thus;* or "12 divided by 3 making 4 ft.; or Thus 8 by 12 in., not 8×12 inches" is apt to be overlooked. Again, some might differ with him on page 25 and write ten is accurate, 10 is exactly, 10.0 is accurate to tenths only. Probably no one has done more to eliminate superfluous words in technical writing than Mr. Rickard, nevertheless on page 62 he writes "So far so funny, but . . ." When writing on the use of titles, he has this statement "In America, it is chaos; the titles Professor and Doctor are employed so loosely that they are well-nigh meaningless." The criticism is that "if it be chaos" his sentence is just as fulsome without that phrase. Usually when an author trips in one paragraph, the critic finds two other stumbles, for example "Surgeons, veterinaries and dentists are denied the privilege," could have been written "The privilege is denied to surgeons, etc." without making use of a false passive. Mr. Rickard's paper on "Standardization of English in Technical Literature" fell like a bombshell among the Englishmen, who were using dreadful words in their writings. It made them gasp and exclaim "Whatever manner of man is he." Read his "Guide to Technical Writing"; hit him in the back when he is looking at you and you will ascertain as the *Canadian Engineer* remarked, that "His pen has a sting so sharp as a serpent's tooth." The intent of the author on Technical Writing was to benefit the mining and metallurgical professions, and this he has accomplished to a greater extent than he is aware. Mr. Rickard's writing is not above criticism; he does not profess to be a hierophant; he is always fair and open to conviction; and does not use his time in useless fault finding.

When *one* is above criticism *one* lacks initiative and becomes what this American thinks an Englishman means when he speaks of another Englishman as a "Rotter."

Tables for the Determination of Minerals is the name of a new book published by the McGraw-Hill Book Co. The price is $2. By means of these tables Prof. E. H. Kraus, and his assistant, W. F. Hunt, A. M., of the University of Michigan, propose to determine 250 minerals by means of their physical properties, occurrences, and associates.

It is difficult to determine minerals by means of physical tests with blowpipe tests for confirmation and vice versa. This the authors seem to realize, for they refer each mineral to one of three standard mineralogies. The authors state that the fundamental basis of these tables depends on luster and color, further subdivisions being introduced by the streak and hardness. We have frequently known of eminent mineralogists making snap determinations of minerals by their appearance, and being caught, but when something depends on it, the blowpipe and wet tests are requisitioned, therefore, why have two or three books where one must be owned. It is difficult for one to determine microline from orthoclase, or pyrite from marcasite, by the use of tables alone, and when it comes to minerals like columbite, wolframite, chromite and others having similar appearance, physical tests alone are not sufficient.

The Thirty-Eighth Annual Edition of The Coal Trade has just been issued by F. E. Seward, for 42 years the editor

and proprietor of *The Coal Trade Journal*, the oldest newspaper in the United States entirely devoted to the interests of the coal trade. The 1911 edition of his annual review of the trade comprises 192 pages, well printed and substantially bound. An index embracing some 600 lines of type indicates how numerous are the subjects covered and how great a mass of information is gathered together in the compactly arranged pages of the volume before us. The book is published by F. E. Seward, 20 Vesey St., New York City. Price $1.50.

BOOKS RECEIVED

United States Geological Survey Publications, Washington, D. C. Bulletin No. 431-B, Advance Chapter From Contributions to Economic Geology for 1909, Part II, Mineral Fuels, Coal and Lignite, by Charles Butts, M. R. Campbell, E. G. Woodruff, C. T. Lupton, J. S. Diller, M. A. Pishel, H. E. Gregory, and A. C. Veatch; Bulletin No. 447, Mineral Resources of Johnstown, Pa., and Vicinity, by W. C. Phalen and Lawrence Martin. This will be found of interest to coal miners. Bulletin No. 465, The State Geological Surveys of the United States, by C. W. Hayes; Production of Asbestos in 1910, by J. S. Diller; The Production of Gold and Silver in 1909, by H. D. McCaskey; The Manufacture of Coke in 1909, by Edward W. Parker; The Production of Monazite in 1910, by Douglas B. Sterrett; Zinc and Cadmium in 1909, Smelter Production, by C. E. Siebenthal; The Production of Coal in 1909, by Edward W. Parker; Water-Supply Paper 257, Well-Drilling Methods, by Isaiah Bowman; Water-Supply Paper 258, Underground Water Papers, 1910, by M. L. Fuller, F. G. Clapp, G. C. Matson, Samuel Sanford, and H. C. Wolff; Geologic Atlas of the United States, Sewickley Folio No. 176, Pennsylvania, by M. J. Munn.

Bureau of Mines, Washington, D. C., Miners' Circular No. 3, Coal-Dust Explosions, by George S. Rice; Bulletin No. 7, Essential Factors in the Formation of Producer Gas, by J. K. Clement, L. H. Adams, and C. N. Haskins.

Department of Mines, Mines Branch, Eugene Haanel, Director, Ottawa, Can., Preliminary Report on the Mineral Production of Canada for 1910, by John McLeish, B. A.; Chrysotile-Asbestos, Its Occurrence, Exploitation, Milling, and Uses, by Fritz Cirkel, M. E.

The Mineral Industries of Canada, by H. Mortimer-Lamb, Secretary, Canadian Mining Institute, Montreal, Can.

Mineral Production of Ontario for 1910, Bulletin No. 7, The Bureau of Mines, Hon. F. Cochrane, Toronto, Can.

Contributions to the Economic Geology of the Republic of Argentina, by Dr. Richard Stappenbeck, of the Division of Mines of Geology and Hydrology, Ing. E. Hermitte, Chief of the Division, Buenos Aires, Argentine Republic.

Proceedings and Collections of the Wyoming Historical and Geological Society, Volume XI, Wilkes-Barre, Pa. The price of this book is $3.

Report of the State Bureau of Mines of Colorado for the Years 1909 and 1910, by T. J. Dalzell Comissioner of Mines, Denver, Colo.

South Dakota School of Mines, Bulletin No. 9, The Badland Formations of the Black Hills Region, by Cleophas C. O'Harra, Rapid City, South Dakota.

Register of the School of Mines of New Mexico for 1910 and 1911, Socorro, N. Mex.

United States Geological Survey, Washington, D. C., Chart of the Mineral Products of the United States for the Calendar Years of 1900 to 1909.

Tennessee State Geological Survey, Nashville, Tenn., Bibliography of Tennessee Geology, Soils, Drainage, Forestry, Etc., by Elizabeth Cockrill; Bulletin No. 4, Administrative Report of State Geological Survey for 1910, by George H. Ashley, State Geologist; Bulletin No. 10-A, Preliminary Study of Forest Conditions in Tennessee, by R. Clifford Hall.

North Carolina Geological and Economic Survey, Joseph Hyde Pratt, State Geologist, Chapel Hill, N. C., Bulletin No. 22, Cid Mining District, of Davidson County, N. C., by Joseph E. Pogue.

Geological Survey of Ohio, J. A. Bownocker, State Geologist, Columbus, Ohio, Fourth Series, Bulletin No. 12, The Bremen Oil Field; Fourth Series, Bulletin No. 11, The Manufacture of Roofing Tiles, by Worcester & Orton.

A Geographical Report on the Franz Josef Glacier, by James MacKintosh Bell, Director, New Zealand Geological Survey, Wellington, New Zealand.

Review of the Joplin District Folio; Highway Construction; Sleet Storm in the Ozark Region of Missouri; Lead and Zinc Mining in the Central States; The Origin of the Lead and Zinc Deposits of Southwestern Missouri; Zinc and Lead Deposits of the Upper Mississippi Valley; Genesis of the Lead and Zinc Ores of the Mississippi Valley, by E. R. Buckley, Ph. D., State Geologist, Rolla, Mo.

Annual Report of the Government Mining Engineer for the Year ending June 30, 1910, Transvaal Mines Department, H. Warrington Smyth, Secretary, Pretoria, South Africa.

Report of the Superintendent of the Coast and Geodetic Survey for 1910, Hon. Charles Nagel, Secretary of Commerce and Labor, Washington, D. C.

RULES FOR HANDLING KANITE

The kanite generally remains loose in the cartridge for any length of time. However, if a cartridge be found packed too tightly, it should be rolled or kneaded between the hands for a few moments until the kanite becomes loose. This operation is absolutely safe and allows the cap or electric fuse to be easily inserted.

The cap or fuse is inserted in a kanite cartridge when loose by turning the cartridge with left hand, at the same time squeezing the top end with two fingers of this hand, while the right hand lightly and easily pushes the cap into the cartridge.

Kanite should be kept in as dry a place as possible. Do not allow packages or cartridge to remain open. This precaution is simply that which has to be followed with any other powder.

The caps are electrical detonators recommended as of greater strength than the so-called double-strength caps.

For placing fuse in cap or in adjusting electrical fuses, the general instructions applied to dynamite can be followed. Either one or the other of the fuses should be imbedded entirely in the explosive and at the top end of the top cartridge.

Kanite is absolutely safe to handle; but after the insertion of the cap the cartridge must be handled with care, not on account of the kanite, but on account of the cap. Kanite should not be stored in contact with the caps or exploders.

When kanite cartridges are placed in a drill hole they should be pressed lightly together. In tamping, after the hole is loaded the first tamping of loose clay, earth, or dummies of same, upon the explosive should be done lightly for a few inches, then tamped in degrees tighter and tighter throughout the remainder of the length of the bore hole. The upper part of the tamping should be as tight and hard as possible to gain the best results of blast.

In calculating a charge of kanite in comparison with 40 or 60 per cent. dynamite and black powder, the weight of kanite should equal the weight of the dynamite and one-fourth the weight of black powder.

In comparison with dynamite only, the weight of kanite should equal the weight of dynamite.

In comparison with black powder the weight of kanite should be one-fourth of the weight of black powder.

This comparison is for the first shot only. By gradually reducing the weight of kanite in each succeeding shot the miner will readily ascertain for himself the proper weight to do the most satisfactory work.

CORRESPONDENCE

Filter Presses

Editor Mines and Minerals:

SIR:—On page 597, of the May issue of MINES AND MINERALS, was published an article by Percy E. Barbour, entitled "Developments in Cyanide Practice." From this article I quote the following:

"Today, five years later, still there is only one wholly filter-press plant of any size in the country and it is unlikely that it would be duplicated if any increased capacity were desired, although the working costs are satisfactory."

I presume the plant above mentioned is the Homestake slime plant, where 1,600 tons are treated daily, as this is the largest slime plant using filter presses.

Mr. Barbour is certainly correct in saying that costs were satisfactory, but might have gone further and said that the working costs at this plant are the lowest in the world. As to its duplication being unlikely, in case of increased capacity, it would rather seem as if Mr. Merrill, the president of this company, and the designer of the plant in question, who still retains the position of consulting metallurgist for the Homestake company, should be equally as well informed as Mr. Barbour, and this statement is certainly news to him. In fact, the plant is being continuously enlarged and a greater proportion of Homestake tailings treated therein. As to its being the only wholly filter-press plant in the country, I would suggest that Mr. Barbour visit the Esperanza, at El Oro, Mex., where he will find that two 90-frame and three 76-frame Merrill presses, have been working for an extended period treating 700 tons per day and with operating costs exceptionally low even for this district.

Moreover, he will find here increased capacity being installed including a sixth Merrill press of 90 frames.

If Mr. Barbour lives in the West, instead of going so far as Esperanza, he need only go to Blair, Nev., and watch three Merrill filter presses at the Pittsburg Silver Peak mill, treating 480 tons per day of a very talcose slime, again with operating costs considerably below any other installations operating under like conditions.

If anything further were needed to show that Mr. Barbour's statement is misleading, it will be found in the following list of orders for filter presses placed with this company in the last year:

Name	Number of Presses	Number of Leaves
Compania Beneficiadora de Pachuca (Santa Gertrudis)	4	360
New York and Honduras Rosario Mining Co.	2	180
Dome Mines Syndicate	2	152
Motherlode Sheep Creek Mining Co.	2	70
Palmarejo Mining Co.	3	270
San Luis Mining Co.	1	35
Portland Gold Mining Co.	2	152

It will be noted that this totals 16 units containing 1,219 leaves distributed over seven companies operating in Canada, the United States, Mexico, and Central America. The approximate tonnage of the above per day is about 2,100 tons.

Evidently filter pressing is not quite as dead as Mr. Barbour would have us believe.

C. C. BROADWATER, Vice-President,
Merrill Metallurgical Co.

Determination of Meridian

Editor Mines and Minerals:

SIR:—I have read with interest and profit the articles in MINES AND MINERALS on the determination of the meridian, by the single altitude method, from an observation of the sun. I introduced the method into this section some 7 or 8 years ago and have used no other for many years. I have used almost every formula that has been suggested, but have used the McElroy formula since it came to my notice. I have modified this formula, however, so as to eliminate the use of the table of natural functions, which seems to me a decided advantage.

The general formula for the solution of the spherical triangle is:

$$\cos a = \cos b \cos c + \sin b \sin c \cos A \qquad (1)$$

In the particular triangle we have to solve

$a = 90° -$ declination
$b = 90° -$ altitude
$c = 90° -$ latitude, and
$A = Z$, or azimuth

which, substituted in (1) converts that equation into

$$\sin d = \sin a \sin l + \cos a \cos l \cos Z \qquad (2)$$

whence,

$$\cos Z = \frac{\sin d - \sin a \sin l}{\cos a \cos l} \qquad (3)$$

which is McElroy's formula, although not in its ultimate form, and in which

$a =$ altitude
$b =$ latitude, and
$d =$ declination

If, now, in order to adapt this formula to logarithms, we let

$$\tan m = \frac{\sin a \sin l}{\cos d} \qquad (4)$$

and substitute this in (3), we shall obtain

$$\cos Z = \frac{\sin d \cos m - \cos d \sin m}{\cos a \cos l \cos m},$$

or

$$\cos Z = \frac{\sin (d-m)}{\cos a \cos l \cos m} \qquad (5)$$

Formulas (4) and (5) solve the problem and permit the use of logarithms. It is to be observed that when the declination is south, $d-m$ becomes $d+m$.

The writer uses printed blanks in a loose-leaf binder and has no difficulty in calculating the azimuth in a few minutes in the field. An example is given herewith, showing the features of the printed blanks and the manner of making the calculation. The sines and cosines are taken out at one opening of the tables and written in their respective places for both formulas. The angle m and its cosine are taken out together and then the sine of $d-m$. The negative and positive quantities are added directly, which saves time, and the problem is solved.

HENRY O. HIPPER

Ocampo, Chihuahua, Mex.

AZIMUTH, LATITUDE AND TIME

Refraction = 57″ × cot. obs. alt.

$$Z = \frac{\sin (d-m)}{\cos a \cos l \cos m}, \quad \tan m = \frac{\sin a \sin l}{\cos d}; \quad H = \frac{\sin (a-n)}{\cos d \cos l \cos n}, \quad \tan n = \frac{\sin d \sin l}{\cos d}$$

$$d = \text{dcln. Green'h} \pm \left(\frac{\text{Long. west of Green'h}}{15} \begin{matrix} +\text{hrs. after, or} \\ -\text{hrs. before, noon} \end{matrix} \right) \times \text{dif. for 1 hour}$$

Latitude = Z ± declination.

Station: S. E. Cor. San Jose claim. Mark: S. W. Cor. same
Time: $8^h\ 34^m$ A. M. Latitude: 28° 20′. Longitude: $7^h\ 14^m$ W
Mag. az. mark: 305° 29′ Mag. az. sun: 70° 07′

ALTITUDE		DECLINATION		LATITUDE	
Obs. alt.	38° 17′	Long.	7.14	Obs. mer. alt.	
Semidiam. Sun +	15′ 30″	Hrs. before N. −	3.43	Semidiam. Sun −	
	38° 32′ 30″		3.71		
Refraction −	1 13	Hourly change ×	8″.05	Refraction −	
Cor. Alt. a =	38° 31′ 17″		29″.8	Cor. mer. alt. −	
		Decln. Gr'n'h	23° 14′ 30″.0		90° 00′
		Declination d	23° 14′ 59″.8		
				Decln. 12 M. −	
				Latitude l =	

AZIMUTH

a .... sin	9.79431	$d-m$ ... sin	8.97629	Z	81° 42′
l ...... sin +	9.67633	a ...... cos −	9.89344	Mag. az.	70 07
d ...... cos −	9.96322	l ...... cos −	9.94458	Variation	11° 35′
m ...... tan	9.50732	m ...... cos −	9.97866		
m .... −	17° 49′	Z ...... cos	9.15961		
d .... +	23° 14′ 59″	Z	81° 42′		
$d-m$	5° 25′ 59″				

The angle between sun and mark is 124° 38′, hence the true bearing of mark is 124° 38′ − 81° 42′ = 54° 31′ − 11° 35′ = N 42° 56′ W.

TIME

d ......sin		*a*		*a* − *n* ......sin	
.......sin +		*n* .... −		*d*. ....cos −	
a ......cos −		*a* − *n*		*l* ....cos −	
n ......tan				*n* ....cos −	
n				*H* ....cos	
				H...........	

Rolling Friction

Editor Mines and Minerals:

SIR:—Some people maintain that the resistance due to rolling friction of mine cars is greater in summer than in winter. In the catalog of a large locomotive concern is this sentence: "The resistance is greater in winter." Will you kindly state which is correct and give reasons? W. C. A.

THE CHEMIST'S PLACE IN MINING

The watchword of modern industry is no longer "increased production" alone, it has changed to "increased efficiency of production." The day of cheap raw material is passing, if indeed it has not already gone, and the mining industry faces a new set of conditions. Nowhere more than in the production and transmission of power are there greater opportunities for improvement—for the saving of time, and labor, and money. That the chemist ought to take the foremost part in solving the problems of power efficiency is the argument convincingly set forth in a paper read before the American Chemical Society by Arthur D. Little, of Boston. Mr. Little makes a vigorous presentation of the chemist's special qualifications for dealing with questions of energy, and urges a stand against the customary monopolizing of power problems by the mechanical engineer.

Chemists forget that their science deals with energy no less than with matter, and that in fact chemistry as a science began with the recognition of the principles, materials, and products of combustion. They have allowed the mechanical engineer to usurp many things which properly come much more directly within their own province.

The combustion of coal is a typical chemical process. The selection of the most efficient coal and the determination of the conditions necessary for its most efficient combustion are essentially chemical problems. Chemical problems also are those arising in the manufacture of producer and illuminating gas, their utilization in gas engines, the development of power from the waste gases of the blast furnace, the adaptation of conditions to the proper handling and burning of lignite and waste coal, the thermometric exploration of coal piles to forestall spontaneous combustion, smoke abatement, the control and improvement of fireroom conditions by draft regulation, flue-gas analysis, temperature measurements, and even the placing of firemen on the bonus basis. Taking power plant practice and the conditions of coal purchase as they stand, the properly equipped chemist should be able to increase the efficiency of power production from 5 to 30 per cent.

The analysis of boiler compounds as an end in itself presents little to excite enthusiasm, but when such analyses are made the means of saving $3,600 a year in the power plants of a single company they take on a new and larger aspect, not only in the mind of the chemist, but in the mind of the chemist's client.

The chemist who attacks the problems of power production will not hesitate to go outside the laboratory and take his property wherever he finds it. He will conduct boiler and engine tests, study the efficiency of grates and stokers, familiarize himself with the marvelous promise of the low-pressure turbine as an agent in efficient power production. While straining every resource of his science to produce steam economically by the combustion of coal, is it common sense for the chemist to stop there in ignorance of the fact that the efficiency of that steam can be increased at once from 25 to 100 per cent. by coupling a turbine to the exhaust?

The distribution of power supplies problems no less directly within the province of the industrial chemist. He may begin with the analysis of lubricating oils. He proves his own inefficiency if he stops there. He must inform himself regarding the market prices of oils used elsewhere for similar service, the adaptability of the oils in question to application to the bearing by soaked waste, sight feeds, or gravity cups. He must be prepared to interpret his analysis in terms of practice, and to follow the oil through the plant in order to prescribe conditions which shall keep down waste. There are few plants in which the industrial chemist working along legitimate lines cannot save from 20 to 60 per cent. of the entire lubrication account, while the oil analyst has to his credit merely a few figures which his client probably fails to understand.

The efficiency and life of bearing metals varies over an extraordinarily wide range. Some are merely the refuse from type foundries, others are so carefully adapted in their composition to the requirements of particular service as to show an efficiency 15 times or more as great as that of inferior material. Here again the mere analysis means little, the practical question is "Which is the more efficient metal under the conditions imposed by practice?"

Much additional might be said regarding the opportunity before the chemist when any material concerned in power transmission is the subject of his study, whether it be leather, rubber, or canvas belting, belt dressing, insulating material, trolley wire, trolley cars, or trolley wheels. In every case it is within his power to create new standards of efficiency.

The chemist is one of the most efficient arms of ore mining and metallurgy, although his field of activities has been narrowed in these industries by his engineering deficiencies and office routine; in fact his specializing process of hibernation prevents his boarding and climbing the fence to his natural field of endeavor.

DISASTROUS AIR EXPLOSIONS

Violent explosions have on several occasions occurred in air compressors on the Rand, fume and poisonous gases descending the air mains to the workmen below with calamitous results; on one such occasion over 100 men were "gassed," some of them fatally. These explosions, which also occur in other mining fields, are brought about in the air cylinders of compressors as the result of carelessness, the use of too much oil, or poor qualities of oil, for lubrication in the cylinders, or defective or leaking delivery valves. Any of these may cause the cylinder lubricating oil to char and the resulting gases to ignite and explode. Very little lubricating oil is needed in the air cylinders—less than in a steam cylinder; good oil in small quantities should be used; an excess of oil only results in the ports and valves being obstructed by a deposit of carbon, and this is apt to bring about a high temperature. Air cylinders may be cleaned by soft soap and water; kerosene and similar inflammable oils should never be used for this purpose. A thin oil, with very little carbon in it, and with a high flash point is the best lubricant for the air cylinder. The explosions in compressors frequently arise from the breaking or sticking of the air-discharge valve, which results in some of the hot compressed air returning to the cylinder, and the additional heat given to this air by the recompression is such as to exceed the flash and ignition point of the oil present, which, therefore, vaporizes and explodes. The ports and valves of compressors should be periodically inspected if accidents are to be avoided.

SCIENTIFIC STUDY OF COPPER DEPOSITS

Written for Mines and Minerals, by A. J. Sale

The Application of Such Information as May Be Obtained from Sampling and Assaying

If an article on this subject were to be complete and involve all considerations and geological features it would fill a whole volume. Such points as the difference between silicified, leached and oxidized monzonite and their twin sisters in the limes, that is, silicified, altered and ferruginous jasperoid; the effect of water level upon oxidization; the difference between kaolinized lime and highly altered rhyolite; the age of rhyolite surface flows; the action of faulting upon deposits; causes of copper deposition: the probable formation of secondary iron pyrites; the action of jasperoid; and hundreds of other geological, lithological, and mineralogical features are of extreme importance in the study of large secondary copper deposits. But in this article I will confine myself to the application of only such information as can be furnished by any assayer. The principal auxiliary in using this information is ordinary coordinate paper.

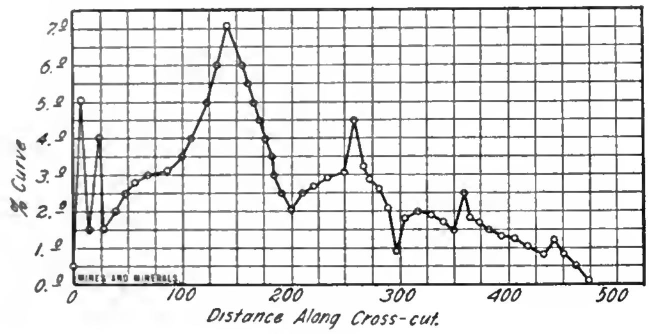


Fig. 1

The first chart that I have found of value, where an ore body is partly blocked out by drifts and cross-cuts, is made by plotting the spacing of the copper assays along the abscissa of the paper and the assays up the ordinates. The paper used for this purpose is a coordinate tracing paper of the same scale of abscissa spacing as the assay maps of the mine. For then the paper can be laid directly on the map and the spacing of the assays obtained without scaling. One inch to 1 per cent. copper is a convenient scale for the ordinates. Fig. 1 shows a typical chart of a cross-cut. A chart of this type is of special advantage where the assays were taken as breast samples and the spacing irregular. I am of the opinion that if a smooth curve were sketched in as a compromise curve to the irregular chart, the errors of sampling will be considerably eliminated; for the spacing can then be taken uniformly along the abscissa and the compromise assay at any point taken from the chart, from which the exact mean assay of the cross-cut or drift can be figured. These charts can be used to determine whether any heading is a drift or a cross-cut. A true drift, which is driven parallel to the axis of the ore body, will have comparatively uniform assays and will show in the chart as a straight line parallel to the abscissa, while headings which are driven at an angle to the axis of the ore body will approach more nearly to the typical form as they approach a condition of true cross-cut, that is, perpendicular to the axis of the ore body. Where the chart is very jagged, the gangue material is usually clay or talc and the copper mostly contained as chalcopyrite. Where any one point is very much higher or lower than the normal, there are probably errors in sampling which should be checked if possible.

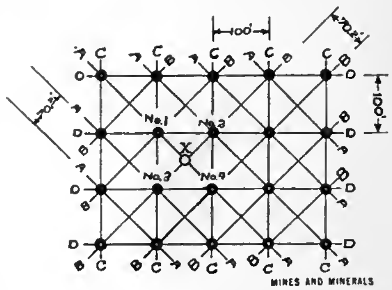


Fig. 2

Where churn drills are used to block out a deposit, the best information is obtained where the holes are drilled at the corners of blocks, usually one or two hundred feet spacing. In studying the results from churn drills, I consider that more accurate results can be obtained from the diagonal cross-sections (*AA* Fig. 2) and diagonal longitudinal sections (*BB*) than from the normal cross-sections (*CC*) and normal longitudinal sections (*DD*). For while the spacing between the holes will be $\sqrt{2} = 1.414$ times as far as the normal sections, the spacing between the sections will only be $\frac{1}{2}\sqrt{2} = .707$ times as far. It is probable that more accurate calculations can be obtained where the spacing between the sections is a minimum. Also the actual conditions can be judged more accurately. In Fig. 3 the ore is assumed to be in a position as shown between hole No. 1 and hole No. 3, and there is no check on this assumption. In Fig. 4 the ore is assumed to be in the position as shown between hole No. 2 and hole No. 3. Assume a hypothetical hole *X* to be drilled midway between holes No. 2 and No. 3. In Fig. 5 the ore is assumed to be in the form which appears most logical. In this figure draw in *X* between holes No. 2 and No. 3. From Fig. 2 it is seen that this should show the same conditions as *X* in Fig. 4. If so all assumptions are probably correct; if not, some other consideration may determine which *X* is nearest to the true condition; after which it can be used as a valuable auxiliary.

Having completely drilled an ore body and made the sections, several important points must be considered before final calculations can be made. The first is that only $\frac{7}{10}$ of any copper assay can be recovered in the mill. Consequently the sections should be readjusted with a new set of assays, each of which is $\frac{7}{10}$ of the assay obtained from the assayer. The next important question is—What is the minimum net assay of copper that can be considered ore? This will be a variable limit according to the price of the metal and many other conditions. It is best to outline the sections to fit the condition of variable minimums of 1.0, 1.5, 2.0, 2.5, and 3.0 per cent. If there are more or less mine workings in connection with the drilling, much assistance can be obtained by the study of "isochemic lines." This is a term that I have taken the liberty of inventing to mean lines of equal chemistry or net copper

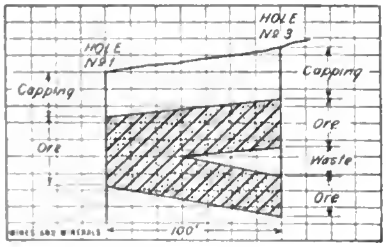


Fig. 3

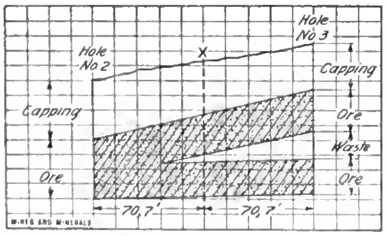


Fig. 4

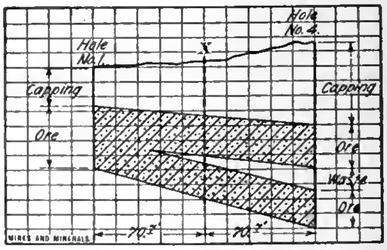


Fig. 5

assay. Fig. 6 shows the application of these lines to a mine map. Fig. 7 shows how the lines in Fig. 6 are used to assist in outlining a drill section for minimums of 2 and 3 per cent. Having outlined each set of sections for variable conditions of minimum the areas can be figured from geometric figures or by a planimeter.

The spacing of the holes being uniform, and the concentration loss adjusted, the net mean grade for any section is probably best obtained by taking the sum of the multiples of the number of feet of ore in each hole by the net mean grade for this number of feet and dividing by the total number of feet of ore in all of the holes.

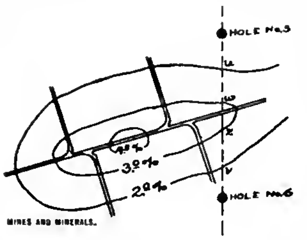


FIG. 6

Having properly adjusted each section, the next thing to do is to study them collectively. Each percentage minimum should be figured separately. In studying the sections collectively, simple isogonic projections are very useful. These are made by assuming that the area of each section lies in a square. Fig. 8 shows how one of these projections is made. If the spacing of the drill holes is 100 feet, and the projection is made from diagonal sections, the distance between the sections will be $\frac{1}{2}\sqrt{2}\times 100 = 70.7$ feet. Suppose that the isogonic projection is to be made to the scale of 1 inch 100 feet. Start with the line AB (Fig. 8) and draw a vertical line every .7 of an inch. Now, since the projection is to be made to the scale of 1 inch to 100 feet, a section 1 inch square will represent 10,000 square feet of area. That is, $h=1$ inch for a sectional area of 10,000 square feet. And, since the area of each section is assumed to lie in a square, each area will be proportional to the square of its h, or h for any area can be found from the formula $h=\sqrt{\frac{\text{area}}{10,000}}$. Having plotted h for each section, the isogonic projection can be drawn in various ways, according to the assumed point of vision. A convenient method is to draw through the top and bottom of each h a line making an angle of 30 degrees with the horizontal. Along each of these lines a distance of one-quarter of its corresponding h is laid off in each direction and the projection completed as shown in Fig. 8. It is probably best to use the prismoidal formula to figure the volume of each segment.

Having figured the volume for any minimum percentage, the tonnage can be obtained by dividing the volume by the number of cubic feet of rock in place to the ton (probably 12 to 15). The final mean grade is probably best obtained by taking the sum of the multiples of each area by its mean grade and dividing by the sum of the areas. The loss of concentration having been compensated for, this will be a net mean grade. We now have a series of tonnages and mean grades figured for 1, 1.5, 2, 2.5, and 3 per cent. minimum of net copper extracted. Suppose it becomes important to know figures for such minimums as 1.3 per cent. or 2.25 per cent. or any other figure not ending in .0 or .5 per cent. This can readily be obtained from a chart of variable minimums. Fig. 9 shows how such a chart is made. The minimum percentages involved are plotted along the abscissa and the results up the ordinates. Any convenient scale can be used. The points on the curve of net production for each figured minimum are obtained by multiplying the corresponding points of tonnage and net mean grade and the product by 20; for each net per cent. is equivalent to 20 pounds of copper recovered per ton of ore. If the chart is correctly made, any intermediate point on the curve of net production will be equal to the corresponding point on the curve of tonnage, times the corresponding point on the curve of net mean grade, times 20. It is seen that a chart of this type furnishes the information for determining the tonnage, net grade, and total net recovery for any percentage.

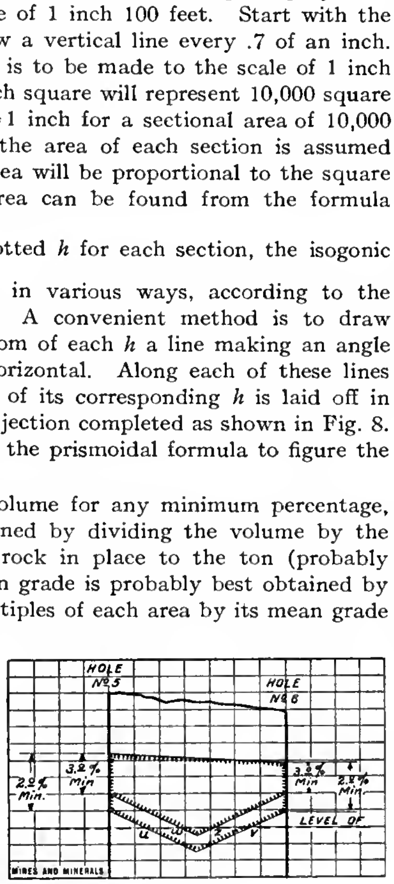


FIG. 7

Having completely figured an ore body for any assumed minimum of recovery, it is usually desirable to show at a glance how the ore body looks in the ground. A very simple and convenient preliminary model can be made by simply tracing any set of cross or diagonal sections on glass plates and setting them vertically in a wooden frame at a scaled spacing proportionate to the distance of the sections apart and at a uniform base of elevation. A more complicated and probably more useful model, where the ore body has been completely drilled, is made by drawing a set of contour lines for the ore body and tracing them, along with the topographical contours, on glass plates which are placed horizontally in a frame at a distance apart proportionate to the distance between contours.

The final information that can be obtained for an ore body is what is to be expected in the smelter. The actual iron analysis will be of little value as there is usually more or less ferrous or ferric oxide in the gangue as well as the concentrate. The net copper, sulphur, and magnetite are the important features. Sulphates should be removed before the sulphur analyses are made. Probably 85 per cent. will be a reasonable estimate of the amount of sulphur, that will go into the concentrate, as the loss of pyrite should not be as great as of chalcocite.

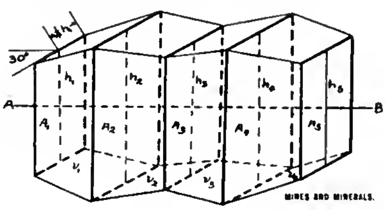


FIG. 8

The magnetite is removed with a magnet and weighed as such. Eighty per cent. will probably be a fair estimate of what goes into the concentrate. Gold and silver should be estimated at from 50 to 70 per cent. of their mean assays, according to the condition in which they occur in the ore.

Assume that the final proportionate mean analysis of an ore body is: Copper, 2.61 per cent.; sulphur, 3.52 per cent.; magnetite, 1.32 per cent.; and gold, 41 cents; silver, .31 ounce.

Now, if the copper is contained as chalcocite along with pyrite, the formula of the concentrate, neglecting the magnetite, will be $Cu_2S+xFeS_2$. Let S' represent the sulphur contained as chalcocite, S'' the sulphur contained as pyrite, and S the total sulphur. Taking the automic weight of copper as 63, of iron as 56, and of sulphur as 32, the ratio $S' : Cu = 32 : 126$ or $S' = \frac{32}{126}\,Cu$ is obtained. Therefore $S - S' = S'' = S - \frac{32}{126}\,Cu$. Also $Fe : S'' = 56 : 64$ or $Fe = \frac{56}{64}S'' = \frac{56}{64}(S - \frac{32}{126}Cu) = \frac{7}{8}S - \frac{2}{9}Cu$. Therefore the iron required is seven-eighths of the sulphur minus two-ninths of the copper. For the above analyses $Fe = \frac{7}{8}\times 3.52 - \frac{2}{9}\times 2.61 = 2.50$ per cent.

The above analysis then becomes: Copper, 2.61 per cent.; sulphur, 3.52 per cent.; iron, 2.50 per cent.; magnetite, 1.32 per cent. Total, 9.95 per cent.

Dividing 9.95 into 100 we get practically 10. Therefore the ore will theoretically concentrate 10 : 1. This I call the theoretical concentration ratio. The theoretical concentrates will then be 10 times each respective analysis, or: Copper, 26.1 per cent.; sulphur, 35.2 per cent.; iron, 25.0 per cent.; magnetite, 13.2 per cent.

But magnetite is $\frac{21}{29}$ iron. Therefore the final theoretical concentrates will be: Copper, 26.10 per cent.; sulphur, 35.20 per cent.; iron, 34.56 per cent.; and gold, \$4.11; silver, 3.10 ounces.

The losses of concentration having been already compensated, the only probable difference between the theoretical

concentrates and the actual will be the amount of gangue (or silica) contained in the concentrate. If for the above analysis the concentrate is assumed to contain 20 per cent. SiO_2, all figures should be multiplied by $\frac{8}{10}$ and the actual concentrate would contain: Copper, 20.88 per cent.; sulphur, 28.16 per cent.; iron, 27.65 per cent.; silica, 20.00 per cent.; and gold, $3.27; silver, 2.48 ounces.

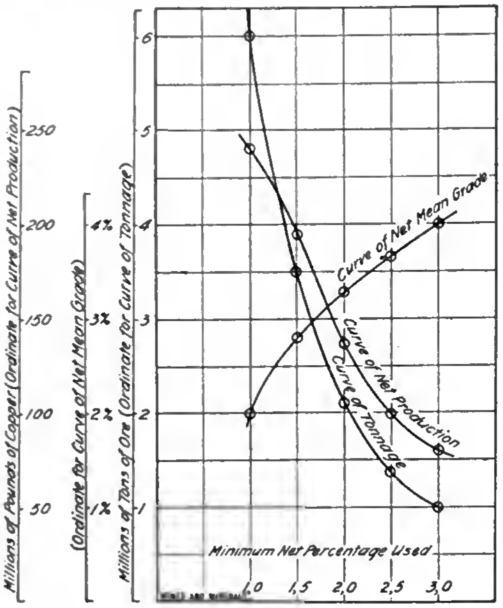


Fig. 9

The concentrate has been increased by one-quarter the weight of the theoretical at the expense of the gangue, therefore the actual concentration ratio will be $8\frac{3}{4} : 1\frac{1}{4}$ or 7 : 1.

IDAHO'S METAL PRODUCTION IN 1910

In his annual report for 1910, Mr. F. Cushing Moore, State Inspector of Mines for Idaho, complains that "very little progress has been made in the mining development of the state (Idaho) during the past year (1910), but the producing mines have maintained their output with a very few exceptions, and a few properties have been added to the producing list by having transportation facilities brought within commercial limits. This lack of progress in development has been caused by the apathy displayed by the investing public, due primarily to the depressed condition of the money market and the reflex action of former flotations and developments which have proven failures."

Notwithstanding the "skinee" for the time being refuses to return to the "skinner" to be further "skun," an increase of nearly 10 per cent. in the value of the metal production over that for the year 1909 indicates a healthy growth in a period of general financial depression, coming as it does mostly from the long established Cœur d'Alene district of Shoshone County.

The totals of metal production for the state are:

Gold, fine ounces	49,289.22	$ 1,018,808.20
Silver, fine ounces	7,890,388.00	4,268,813.00
Lead, pounds	239,144,570.00	10,761,057.70
Copper, pounds	5,837,639.00	753,055.40
Zinc, pounds	5,995,600.00	333,513.60
Grand total for 1910		$17,135,695.90
Grand total for 1909		15,606,862.15
Increase		$ 1,528,833.75

It will be noted that the individual items when totaled amount to $448 less than the sum given by Mr. Moore.

The production of Shoshone County, the Cœur d'Alene district, for 1909 was valued at $13,723,105 and for 1910 at $15,275,024.15, or approximately 90 per cent. of that of the entire state.

THE VALUE OF SURFACE TRENCHING

Written for Mines and Minerals

If the true anecdotes of all the mineral finds made by surface trenching could be collected, a volume would doubtless be the space required for their preservation. Certain Spokane operators, for instance, might relate how a Montana deposit was developed by a little trenching from a $50,000 gamble into a sure thing they were lucky to acquire for 10 times that amount. Goldfield, Cripple Creek, Cobalt, and Porcupine operators could relate many instances of finds made by trenching that have added millions to the world's store of bullion. Yet in spite of the evidence known to all well-informed engineers and practical mining men, only at intervals is it known how neglect of this simple elementary principle of mining has led to great waste of mineral or capital.

For instance, the other day it was learned that a certain 4-foot coal seam was mined by long wall and after scores of acres had been crushed and caved a 6-foot seam of superior coal was discovered only 40 feet vertically above. In this case a little shallow surface trenching through the loose alluvium covering the outcrop would have quickly shown the presence of the upper coal seam before mining started.

A more ridiculous case than this, however, was recently observed by the writer at Globe, Ariz. In the early days of a certain mine at this camp, the outcrop of a 6-foot vein of copper ore was discovered on a hill top. In order to gain depth on the vein, the prospectors started a would-be drift in a gully some hundreds of feet below, thinking to encounter the vein and drift in on the ore. Instead of trenching through the wash of the hillside slope, near the gully bottom to positively locate the outcrop, however, the drift mouth was located by guess and pointed toward the outcrop on the hill top. Suffice it to say that the drift was driven roughly 600 feet in hard rock before intersecting the vein at an acute angle. Had the prospectors first trenched the hillside, however, they would have located the vein outcrop not 50 feet from the mouth of their drift and so gone into the hill on fine solid ore that would have more than paid the expense of the development. As it was their finances were exhausted by the time ore was struck and they had the usual prospectors' experience of disposing of the property for a song. Such instances as these make the engineer weary, you say, and perhaps add—"If only those prospectors had sought the advice of some of us who know"—but listen.

In a certain recently discovered Nevada camp, the paying mines were all located at the foot of a hill. The depth of surface wash was here more than 50 feet deep, but 100 feet above the level of a certain shaft the rock outcrops were bare. Now certain Eastern investors believed that the mineral zone continued beyond the line of mines at the foot of the hill slope and out under the surface wash. They located the alluvium flat and proposed to locate their shaft not far from the line of established mines; they next hired a geologist to examine their property and report upon a feasible location for their shaft. His work was easy. The most superficial examination of the bare rock outcrops determined the fact that there were no fissures extending at right angles to the great group occupied by the developed mines, and out under the alluvium wash held by his clients. A short examination of the workings of the nearest mine confirmed his deductions from study of the outcrops. But the owners were not to be thus bluffed out of their valuable ground. So they picked a shaft location at a point commanding a splendid view of the desert valley and started sinking. They went 600 feet through wash, and then 200 feet through bed rock, cross-cut 500 feet to each side of the shaft, and then gave up in disgust.

SURFACE ALTERATIONS OF GOLD ORES

*By Albert D. Brokaw**

Experiments to Determine the Action of Surface Waters in Solution of Gold

The leaching of gold from the outcrop of auriferous lodes has been the subject of much discussion, and many contradictory statements regarding the chemistry involved have arisen in the literature. Frequently these statements are based on experimental evidence, and the contradictions may be explained in part, at least, by a lack of uniformity in the conditions under which the experiments were carried out, and a tendency to ignore the conditions of temperature and concentration of solution that we are justified in supposing to be operative in the surface alteration of such deposits.

At the suggestion of Prof. W. H. Emmons, the writer undertook a series of experiments with a view of determining which of the various solvents noted are most effective in the solution of gold. By limiting the problem to alteration, many substances are eliminated; only such as are known to occur in mine waters or in the gossan were studied, and the concentrations used are comparable to those shown by mine waters. The experiments were carried on at room temperature (18 degrees to 25 degrees), as Stokes† has shown that elevated temperatures have a very marked influence on the solubility of gold in ferric salt solutions.

A few of the solvents suggested by Don, Rickard, Lenher, and others, were made the subject of a comparative study, the conditions of temperature and concentration being practically uniform for the series. The substances studied were ferric sulphate, ferric chloride, sulphuric acid, hydrochloric acid, and manganese dioxide. These were covered by the following experiments, each in duplicate. Solution of gold is shown by loss of weight.

1. $Fe_2(SO_4)_3+H_2SO_4+Au$
   (*a*) No weighable loss.
   (*b*) No weighable loss.
2. $Fe_2(SO_4)_3+H_2SO_4+MnO_2+Au$
   (*a*) No weighable loss.
   (*b*) .00017 gram loss. (The duplicate was found to contain a trace of *Cl*, which probably accounts for the loss.)
3. $FeCl_3+HCl+Au$
   (*a*) No weighable loss.
   (*b*) No weighable loss.
4. $FeCl_3+HCl+MnO_2+Au$
   (*a*) .01640 gram loss. Area of plate, 383 square millimeters.
   (*b*) .01502 gram loss. Area of plate, 348 square millimeters.

The solutions were tenth normal‡ with respect to ferric salt and to acid. In each duplicate 50 cubic centimeters were used. In experiments 2 and 4, 1 gram of powdered manganese dioxide was added to each duplicate. The gold was obtained from Goldschmidt Bros., and assayed 99.9 per cent. pure. It was rolled to a thickness of about .002 inch, and cut into pieces of about 350 square millimeters area, and one piece, weighing about .15 gram, was used in each duplicate. The gold was washed with alcohol and ether and dried, then each piece was carefully weighed. The experiments were carried on in tightly stoppered test tubes, which were thoroughly shaken from time to time. After 2 weeks the pieces of gold were removed by means of a platinum wire, and washed with water, alcohol, and ether, in turn, before weighing. In experiments two and four, a small amount of manganese dioxide adhering to the plates was removed by means of a solution of ferrous sulphate acidified with sulphuric acid, after which the plates were treated as above. At the end of 2 weeks all but experiment 4 gave negative results when weighings were made to .0001 gram, and the balance was exchanged for one sensitive to .00001 gram, which was used from that time on. The whole time was 34 days. In experiment 4 it will be seen that the losses in (*a*) and (*b*) are approximately proportional to the areas of the plates.

The results of these experiments, as given above, show conclusively that, of the conditions under consideration, the most favorable for the solution of gold involve the presence of manganese dioxide and chloride. Although it is frequently stated that gold is readily soluble in ferric sulphate solutions, no loss of gold was detected after 34 days contact with a tenth normal solution of that salt.

In order to reproduce more nearly the conditions in nature, experiment 5 was prepared as follows: A solution was madeN/10 with respect to ferric sulphate and sulphuric acid, and N/25 with respect to sodium chloride. To 50 cubic centimeters of solution 1 gram of powdered manganese dioxide was added and the experiment was carried on as before. (This is essentially the experiment of Rickard, Trans. A. I. M. E., XXVI., 798. From experiments 6 and 7 it appears that the ferric salt is unnecessary.) The loss is comparable to that found in experiment 4, allowing for the shorter time and the greater dilution of the chloride.

5. $Fe_2(SO_4)_3+H_2SO_4+NaCl+MnO_2+Au$
   Loss of *Au* .00505 gram. Time 14 days.

The same experiment without MnO_2 showed no loss of gold.

After it had been shown that chlorides and manganese dioxide were necessary under these conditions, the next point to be determined was whether the free acid or the ferric chloride is the active agent in bringing about the solution. In experiment 6, 50 cubic centimeters of N/10 *HCl* was used with 1 gram of powdered MnO_2. In experiment 7, sodium hydroxide was added to 50 cubic centimeters of N/10 ferric chloride solution until the precipitate formed barely redissolved on shaking (even then the solution was somewhat acid owing to the hydrolysis of the ferric chloride), after which 1 gram of powdered MnO_2 was added.

6. $HCl+MnO_2+Au$
   Loss of *Au*, .01369 gram. Time, 14 days.
7. $FeCl_3+MnO_2+Au$
   Loss of *Au*, .00062 gram. Time, 14 days.

The experiments were conducted as before. The results show clearly that the free acid, rather than the ferric chloride, in the presence of managnese dioxide exercises the great solvent action, as the same amount of chlorine was present in both cases. Essentially, the most favorable conditions for the solution of gold are those in which free chlorine may be liberated.

W. J. McCaughey (in "Journal of the American Chemical Society"), in studying the solubility of gold in ferric salt solutions, found that ferrous sulphate, even in very small amounts, had a marked effect in depressing the solubility of gold. Conceivably this may be a factor to be considered, and with this in view experiment 8 was performed, to determine whether ferrous sulphate, in the presence of sulphuric acid and manganese dioxide, would be quickly oxidized to the ferric salt, according to the following equation:

8. $2FeSO+2H_2SO_4+MnO_2=Fe_2\ (SO_4)_3+MnSO_4+2H_2O$.

After acidifying 100 cubic centimeters of 1.6 *N*. $FeSO_4$ solution with sulphuric acid, it was shaken vigorously with 5 grams of powdered MnO_2. After 5 minutes the solution was filtered off. No ferrous iron was detected by the ferricyanide test, showing that the iron had been completely oxidized to the ferric state. The experiments were not done in such a way that the velocity of the oxidation could be measured, but the result shows that ferrous sulphate in acid solution is quickly oxidized by manganese dioxide, hence the suppression of the

* Abstracted from The Journal of Geology.
† Stokes. Economical Geology, 1, 650.
‡ Normal as used in this paper refers to equivalent normal solutions. Tenth normal concentration was selected rather arbitrarily except for the fact that it is well within the range of concentration shown by mine waters. (See table of analyses.)

solution of gold by ferrous salts may be disregarded if manganese dioxide is present. It is interesting to note in this connection that the analysis of a sample of water from the Comstock lode (see analysis 3 in Table 1; Nos. 5 and 6 illustrate the same fact), showing the greatest amount of ferric iron, showed a considerable amount of manganese, but no ferrous iron; a condition in accord with the right-hand side of the above equation.

McCaughey's experiments on the solubility of gold in ferric salt solutions were made with stronger solutions than are known to occur in mine waters, but his results show that gold is attacked by ferric salts in the presence of hydrochloric acid.* While no loss of gold was noted in experiment 3, as carried out by the writer, it is possible that in a longer time some loss might be detected, as more dilute solutions act more slowly

TABLE I. ANALYSES OF VADOSE WATERS

	1*	2†	3‡	4§	55	6‖
Cl	12.40	186.40	127.60	19.00	tr.	tr.
SO_4	124.80	161.70	209,100.00	474.00	258.40	26.55
CO_3		1,513.44		20.45		
NO_3		1.60				
PO_4		tr.				
K		198.00		53.40		
Na		719.45	535.00	132.00		
Li		2.85				
Ca	46.40	146.41	1,286.00	100.00	121.40	72.48
Sr		1.95				
Mg	14.50	177.67	6,590.00	5.88	13.08	14.90
Al		1.06	9,760.00	1.37	1.49	.37
Mn		.57	885.10		4.72	4.12
Ni					tr.	tr.
Co					tr.	tr.
Cu		.02	147.50			
Zn	8.90	.34			2.82	47.40
Pb		1.35				
SiO_2	18.00	24.42	616.00	133.40	2.10	8.00
Fe^{++}						
Fe^{+++}	6.60	3.50	5,025.00	6.33	4.74	6.30

* United States Geological Survey Bulletin 330, 547.
† Ibid.
‡ Bulletin University of California, IV, 192.
§ Ibid, 189.
‖ Beck, Nature of Ore Deposits (Weed), II, 377. Analyses expressed in milligrams per liter. Where necessary they have been recast.

From the above experiments, it appears:

1. That at the dilution of natural solutions of ferric salts their solvent effect on gold is probably very slight.

2 That in the presence of manganese dioxide no increased solubility is found unless chlorides are present.†

3. That mixtures of ferric sulphate, sulphuric acid, and sodium chloride in concentrations common in mine waters will readily dissolve gold in the presence of manganese dioxide.

4. That free hydrochloric acid in the presence of manganese dioxide has a much greater solvent effect than the same amount of chlorine in ferric chloride solution.

5. That the influence of ferrous salts in suppressing the solubility of gold is negligible if manganese dioxide is present.

* If gold is dissolved by ferric chloride the reaction might be expected to be as follows:

$$Au + 3\,FeCl_3 = AuCl_3 + 3\,FeCl_2$$

This would seem to be a reversible action, as ferrous salts are commonly used as a precipitant for gold. McCaughey, however (op. cit., foot-note, page 1,270), failed to detect any ferrous salt after the action had gone on for 2 days. It seemed probable that ferrous chloride had been formed, but was oxidized by contact with the air. Accordingly the experiment was repeated in an atmosphere of carbon dioxide, care being taken to exclude, as far as possible, all contact with the air. Several possible sources of error were not eliminated and the experiment is only of preliminary nature. The results were as follows:

	Loss of Gold	Weight of Ferrous Iron	
		Observed	Calculated
(a)	.02904	.0240	.02476
(b)	.03247	.0282	.02769

This seems to establish the correctness of the equation given above. The result is not in accord with the statement of McIlhiney (Am. Jour. Sci., Ser. IV. II., 293) who found that gold dissolved in ferric chloride only in presence of air. The greater dilution at which he worked may account for this.

† The influence of manganese compounds in chemical reactions involving oxidation is noted in many cases. E. g., Moissan (Chem. Minerale, V., 617) states that fuming hydrochloric acid in presence of air will dissolve gold, especially if manganous chloride is present. The catalytic action of manganese dioxide in the decomposition of potassium chlorate and hydrogen peroxide are well known. Probably manganese compounds are of considerable importance in natural oxidations, even though they may be present in very small amounts.

6. That the solution of gold is practically limited to the oxidized zone.

In the agreement with these experiments is the fact that manganiferous lodes bearing pyrite, in areas of chloride waters, are leached to greater depths than lodes that do not carry manganese.

Six complete analyses of vadose waters from their tables are given in Table I.

THE ALCHEMIST IN MODERN INDUSTRY*

In a plea for a more generous publication of results obtained in modern industrial research laboratories, Prof. Wm. H. Walker declared that the spirit of the old alchemy—namely, secrecy—was still in force, and depriving the world of much knowledge that the discoverers could share without harm to themselves.

There is a heavy moral obligation on the part of large industrial organizations having fully-equipped research laboratories, said Professor Walker, to contribute their share to the advance of the world's knowledge. They have well-stocked libraries, and are provided with all the current periodicals; they profit by all the scientific work which has been done and is being done. This is as it should be, and such firms are to be commended for their progressiveness. But is this not a reason why such laboratories should do their part in adding to the sum of available knowledge? There is in every laboratory much work which could be published and yet conserve the interests of the corporation. First, there are the results which may not have proved valuable to the laboratory in which they were obtained, but which would be of immense value to some one else working in an entirely different field. Second, there are those results of value to the laboratory possessing them, but which could be published in an unapplied or "pure" form and which would make an important contribution to science, and at the same time the publication would work no injury to the company or corporation most interested. And finally, there are those results of operations and processes, machines and apparatus which, if the truth were known, are possessed by a number of concerns, but are held as valuable secrets by each. Every one would profit and no one be the loser by so far-sighted and generous a policy. Germany is very justly held up before us as a shining example of marvelous industrial progress and prosperity. A very great deal of the credit for her present position is due to her splendid educational system. But no small factor in her national progress is the helpful attitude which her industrial organizations take toward the publicity of scientific data. The individual does not suffer, while Germany, both from a purely scientific and an industrial standpoint is rapidly advanced. But too often with us the president and his board of directors are alchemists; they fail to see why if they pay the salaries of their research men, they should give to the public, or their competitors, any part of their results. They exclaim: "What has posterity done for me?" They would have their laboratories remain the secret chambers of the alchemists, and continue to improve their methods of changing baser materials into gold without regard to the obligations which they owe to their fellows.

It requires no extensive mathematical calculation to prove that the manufacturers themselves would be the ones to profit by such a liberal treatment of the results of scientific work. Of 100 manufacturing concerns, each one would give but 1 per cent. to the whole contribution, while he would receive the remaining 99 per cent. He could not in the long run be the loser. But of vastly more importance, he would feel and know that his organization was taking part in a world movement toward that increase of human knowledge upon which all real progress depends.

*Abstract of paper by William H. Walker, Professor of Industrial Chemistry, Massachusetts Institute of Technology, Boston.

CHEMISTRY OF THE CYANIDE PROCESS

*By W. H. Seamon**

The progress that has been made in solving knotty problems in treating different ores with the cyanide process formed the basis of a recent article by Mr. W. H. Seamon, in the *Mexican Mining Journal.* "Much as we know," he says, "of the chemical reactions involved, we have scarcely begun to learn anything about the complex."

The Necessity of More Knowledge of the Effects of Various Compounds Resulting in the Process

Thirty-three metallic cyanides have been isolated by chemists; 250 cyanides are more or less known; something like 450 ferrocyanides and ferricyanides, including their double salts, have been slightly studied, and quite a large number of sulphocyanates and cyanates are more or less known. Any one of these compounds may occur in the treatment of ores by the cyanide process. While their chief physical and some of their chemical properties are known, yet practically nothing is known of their possible effects in the cyanide process; no systematic study has been attempted with reference to their influence in the winning of gold and silver. It is very desirable that a study be made of all these compounds, but it is a question likely to be overlooked. Scientists have other questions that interest them. Practical cyanide men have to get results and cannot afford to spend 2 or 3 years in preparation of a foundation for the chemical understanding of the cyanide process. Unless cyanide men get together and put up the funds to employ an investigating force, this knowledge will be accumulated very slowly.

This article is intended to be suggestive, rather than an epitome of the chemical facts known. No attempt is made to give credit to the many authorities from whom the facts have been gathered, it being considered sufficient to acknowledge indebtedness to everybody who has written on the subject.

Gold, particularly when finely divided, dissolves in aqueous solution of potassium cyanide in the presence of air, forming aurocyanide of gold and potassium according to the equation:

$$H_2O+2Au=2AuK(CN)_2+2KHO$$

The presence of oxygen is absolutely essential for this reaction. Some practical men have stated that they have made a good extraction without blowing air through the solutions. It is certain that in these instances the oxygen has been supplied in some way unknown or unconsidered by the gentlemen. It may have come from the oxides in the ores, but more probably from the small amounts of oxygen dissolved in most natural waters. The question has been thoroughly studied in the laboratory, under exact conditions, and even though gold has been extracted in practical work without the use of an air current the truth of the statement that it is necessary is not affected. The following are the conclusions of Christy, based upon actual work with ores:

1. The gold and silver are capable of dissolving at ordinary atmospheric temperatures and pressures to a limited extent, without the additions of oxygen or oxidizing agents.

2. At ordinary temperatures the atmospheric pressure has an indirect influence on the constancy of the dissolving action.

3. For the continued dissolution of gold and silver at ordinary atmospheric temperatures and pressures, oxygen, or an oxidizing agent, is desirable, if not absolutely essential.

4. Temperature has a direct influence on the dissolution of the metal, independent of the oxygen absorbed by the solution.

5. The value of oxygen absorbed by the solution is dependent on atmospheric pressure and to some extent on temperature.

The oxygen supply is best furnished by blowing air through the solutions. Strong oxidizing agents have been employed very successfully in some instances, generally at the expense of a larger consumption of cyanide, which is a compound easily decomposed. When there are reducing agents in the ores a larger supply of oxygen must be furnished.

Commercial cyanide is likely to contain small amounts of alkaline sulphides and their presence not only proportionately diminishes the solvent power of the cyanide, but may require a larger supply of oxygen.

Metallic silver dissolves in potassium cyanide solution precisely as does metallic gold. Oxygen is necessary for the chemical reaction and must be supplied by the introduction of air when not furnished otherwise. The double salt, potassium silver cyanide is soluble in four parts of water at a temperature of 20° C., and is dissociated by acids and sulphuretted hydrogen. The presence of alkaline sulphides in the cyanide is more detrimental in the solution of silver ores than in the case of gold ores. This statement will be contradicted by many practical men, but the experimental work in the laboratory, under known conditions, demonstrates the truth of this statement and the fact that in certain cases in practical work it has been observed that there was no detrimental action from the presence of alkaline sulphides must be explained on the theory that there was in the complex constituents of the ores treated something that neutralizes the deleterient effect of the alkaline sulphides, expected from the results of laboratory experiment.

In the cyaniding of silver ores containing silver sulphide minerals, the soluble sulphides should be removed, and oxygen is an important agent in the decomposition of the sulphides even in the presence of lead salts. A chloridizing roast is always beneficial for most silver minerals, although it is not essential, and is seldom employed in the cyanide treatment of silver ores. A chloridizing roast will largely increase the extraction and obviate the presence of oxygen at the time of action of the cyanide solution.

In handling silver ores the reactions are more complex than in handling ores of gold, owing to the greater chemical activity of silver as well as to the fact that the solutions of cyanide used are necessarily much stronger. It is in the handling of silver ores that it is so important that more knowledge should be secured concerning the parts played by the various double cyanides, ferrocyanides and ferricyanides, in the solution of silver.

No simple cyanide of iron has, as yet, been isolated; they are, if they exist at all, very unstable and occur only in solution, decomposing when attempt is made to secure them in a solid state. It is well known that iron and steel and various iron oxides consume varying quantities of cyanide. The compounds of iron which most readily form ferrous salts consume the largest quantities of cyanide. While cyanides of iron are practically unknown, the ferrocyanides have been isolated and studied, and they are always formed in the cyanide process. As metallic iron and pieces of steel break off from the wearing parts of the machinery employed in preparing ores for cyanide treatment, it is important that a careful study should be made of all the possible influences of the ferrocyanides in the solution of gold and silver.

The sulphides of iron, as marcasite and pyrite, are also commonly occurring constituents of ores and they are large consumers of cyanide. All of the sulphides of iron decompose more or less rapidly in the presence of water and form soluble ferrous and ferric salts. When air is used in agitation there is a marked increase in the decomposition of the sulphides of iron. Some authorities advise preliminary washing out of all ferrous salts before adding cyanide solutions.

The salt $Zn(CN)_2$ is known and several double cyanides, corresponding in formula to the potassium zinc cyanide, $Zn(CN)_2 \cdot 2KCN$. These double cyanides are known to be solvents for gold, but, as yet, their influence in the extraction of gold has not been completely determined. They are sure to occur in varying amounts in the treatment and it is very important that more should be known of their properties and

* Abstract of article in *Mexican Mining Journal.*

effects. It has not been determined what effect the formation of the double cyanide of zinc has in preventing the precipitation of gold in the zinc boxes. It is well known that the salt helps dissolve gold when the solutions are reused.

Chemists know many double manganese cyanides and many more ferrocyanides, but their relations to the cyanide process are not known. The same may be said of aluminum, barium and calcium cyanides and ferrocyanides. It is known that calcium carbonate, a constituent of ores, is acted upon by alkaline cyanides, resulting in a consumption of cyanide. So long as the solutions are kept alkaline the loss is negligible.

All commercial cyanide contains more or less sodium cyanide, and so-called 125 per cent. cyanide is nearly pure cyanide of sodium. There is no scientific reason known why the sodium salt should not be as good a solvent of gold and silver as the potassium cyanide; it is a fact that pound for pound it possesses greater solvent power and can be produced more cheaply. These things make it especially desirable, and it is a matter of surprise that many cyanide men have found the sodium salt less efficient than the potassium. The problem has not been thoroughly worked out, but I am satisfied that when the facts are accumulated, it will be found in favor of the sodium cyanide.

Recently, there has been an awakening to the fact that many brands of cyanide are not so cheap as their price would indicate. For a long time but little attention has been paid by consumers to the impurities existing in different brands of cyanide. It will not be long before the sellers will have to sell on guaranteed analysis.

STANDARDIZATION OF BULLION ASSAYS*

Report of Committee "B," Institution of Mining and Metallurgy on Bullion and Assay Values

The committee for the standardization of bullion and assay values was formed for the purpose of recommending rules which shall, as far as possible, be followed by assayers, metallurgists, and mining men, in reporting the results of assays and of metallurgical, dressing, and other tests or which assays form an essential part, so that ambiguity shall be avoided and the possibility of misinterpretation or misuse of reports shall be minimized.

Reports of assay results, extraction tests, metallurgical, and ore-dressing trials, etc., are sometimes so worded as to permit a wrong interpretation, and the committee urges the necessity for definite statements as to the nature and condition of the sample as assayed, and, where advisable, an indication of the method of assay. Thus, in the case of such metals as copper, tin, lead, antimony, etc., where both "wet" and "dry" methods of assay are in use, results should be reported as obtained by "fire" assay or by "wet" assay, with further details as to method, if judged advisable. Similarly, it should be stated whether the sample was assayed in the condition in which it was received or "air dried" or dried at —°C.

In reporting that an assay for such a constituent as tin, arsenic, tungstic acid, etc., corresponds to so much "black tin," "white arsenic," "wolfram" or other recognized saleable product, the report should give the assay result for such constituent, together with a definite statement as to the figures used in calculating such saleable product. Thus, an assay showing 1 per cent. of metallic tin might be reported as 1 per cent. of metallic tin, equivalent, if all could be extracted as such, to 1.5 per cent. of "black tin" containing 66.6 per cent. (or two-thirds) of its weight of metallic tin. Similarly, in reporting results, where assay values might be mistaken as representing available produce, it should be stated that such are merely "gross" assay values, a statement or estimate of the actual "commercial" values being added whenever possible.

In such matters as ore dressing, ore-treating processes, furnace work, etc., where so much depends on the way in which results are reported, on the nature of the sample and on the method of taking it and its relation to other samples, the committee emphasizes the necessity for careful phraseology and for the inclusion of exact data, as also the necessity for care in preparing, describing, labeling, sealing and otherwise ensuring the correctness and authenticity of samples submitted for tests or assays.

The Bullion and Assay Values Committee submit the following recommendations for adoption by the Institution:

1. Assay reports shall state the exact condition of the sample as to dryness when assayed.

2. Assay values of gold and silver ores and products shall be represented in pennyweights and decimals or in ounces and decimals, and not in oz., dwts., and grains. They shall be expressed in terms of fine gold and fine silver, respectively, not as "bullion."

3. Assay values of alluvials shall be reported in grains and decimals of a grain of "fine" gold, or in pence (at 2d. per grain of "fine" gold), or cents per cubic yard.

It is recommended that, in the absence of specific information, one cubic yard of ordinary alluvial, excluding boulders, be taken as equivalent to 3,000 pounds (1½ short tons).

4. In reporting assay values of cyanide and other solutions, the results shall be given in parts by weight in a stated volume of the solution. (In the case of cyanide solutions, the use of the "fluid ton of 32 cubic feet" is recommended. It closely approximates to 2,000 pounds and is in common use.)

5. When it is necessary to state or estimate the money value of an ore, etc., (other than of gold), it shall be accompanied by the assay value, and the basis on which the former has been calculated from the latter shall be stated.

6. Laboratory sieving tests shall be made with the I. M. M. standard sieves, or, when other sieves are used, the widths of the apertures shall be stated.

With the above recommendations, if approved, will be included the following, which have already been adopted on the recommendation of the Weights and Measures Sectional Committee:

(*a*) The word "ton" shall represent a weight of 2,000 pounds avoirdupois (29,166.6 ounces troy).

Note.—It is advisable to abandon the use of the terms hundredweights and quarters, and to express fractions of a ton in pounds, or in decimals of a ton.

(*b*) The word "gallon" shall represent the imperial gallon measure of 10 pounds of water.

(*c*) Temperatures shall be expressed in degrees centigrade.

(*d*) Returns of gold and silver shall be expressed in terms of fine gold and fine silver, respectively, not as "bullion."

(*e*) Gold contents of ores, etc., determined by assay, shall be expressed in money values as well as in weights; and in this connection the value shall be taken (as a convenient constant) at 85 shillings or $20.67 United States currency, per troy ounce of fine gold.

*Before finally adopting the recommendations of the Committee, the Council will be glad to consider any suggested alterations or additions, which should be received by the Secretary The Institution of Mining and Metallurgy, not later than June 30, 1910.

COPPER REFINING

The largest copper refining plant in the world is located in Brooklyn, and is known as the Laurel Hill works of the Nichols Copper Co. At this plant there are copper smelting furnaces, converters for bessemerizing copper matte, and the necessary furnace paraphernalia in connection with casting copper anodes, wire bars, slabs, etc. There is also an extensive sulphuric acid plant that utilizes the excess sulphur from the copper ores.

THE CROWN MINES, LIMITED*

The property embraces, from the east, the Robinson Central Deep, Crown Deep, Crown Reef, and South Rand; Langlaagte Deep, Langlaagte Royal, and Pearl Central on the west, and in addition 1,278 claims of deep-level ground. In extent it is about 3 miles from east to west on the strike of the reef.

Electric Hoisting and Haulage. A Successful New Cyanide Plant of Large Capacity

There are on the property nine main shafts and seven crusher stations when including those at the Pearl Central and Langlaagte Royal, and the property is being so laid out, both on the surface and underground, that in the future there will be only two large crusher stations; one at No. 5 and one at No. 7 shafts. By a system of underground chutes and haulage, the rock from the Langlaagte Royal, Pearl Central, Langlaagte Deep and western areas will be concentrated at No. 7 shaft and hoisted there, while all the rock from the eastern section will gradually be concentrated at No. 5 shaft. These two shafts will have to handle between 9,000 and 10,000 tons daily, or the full requirements from the mine. The two shafts will be connected on the thirteenth level at a vertical depth of approximately 2,200 feet. This level will be a main haulage level, the main drive being 14½ feet wide and practically straight from one end of the property to the other. This drive will be served by electric haulage, so arranged that ore may be delivered to either shaft.

The No. 7 shaft is vertical to inclined, with three hoisting compartments and a pump way. Alterations are in progress to convert the hoisting at this shaft into double stage with four hoisting compartments in the vertical, for 4-ton skips, and four hoisting compartments in the incline for 5-ton skips, all hoisting to be done by electricity.

The No. 7 shaft crusher station is the western station of the Crown mines, and is being equipped to handle 3,500 to 4,000 tons of ore on one day shift only. The special features at this station are three 30-inch conveyer belts delivering into nine 30″×12″ Hadfield & Jack crushers. The return sides of these belts are utilized for waste, which is delivered to a transverse belt delivering to the waste bin and thence to dump. The rock from the crushers is caught in bins of 1,000-tons capacity, thence it is discharged through pneumatically operated ore gates into 40-ton hopper cars. These cars are hauled in trains by 50-ton electric locomotives, and may be delivered to any one of the mills of the Crown mines. The cars are constructed so that their discharge may be pneumatically controlled from the locomotive, thus reducing the labor required to a minimum.

The No. 5 shaft has a vertical depth of 3,400 feet. Three main haulage levels will be established in this shaft at distances of 600 feet apart, vertical. These main haulage levels will be the thirteenth, sixteenth, and nineteenth, and they will control all the ore down to the South Rand dike. On each of the levels electric haulage will be installed. At each of these levels the ore bins will have a capacity of 2,000 tons, and winding will be done by three 8-ton electric hoists, as large, if not larger, than any electric hoists in the world.

The No. 5 crusher station will be similar to that at the No. 7 shaft, except that its capacity will be from 5,000 to 6,000 tons per day of 10 hours.

The present metallurgical equipment consists of 675 stamps, 20 tube mills, and sand and slime plants made up as follows: Crown Deep, 300 stamps and 10 tube mills; Langlaagte Deep, 200 stamps and six tube mills; Crown Reef, 120 stamps and three tube mills; Bonanza, 55 stamps and one tube mill.

With the present plant there is a capacity of 165,000 tons per month.

A fifth mill is being erected of more modern type, which, when completed, will increase the total crushing capacity to more than 200,000 tons per month. The special features are the tandem classifications and the tube milling at Crown Deep plant.

The installation of a 300-leaf Butters vacuum-filter slime plant at Crown Reef plant is interesting not only on account of its being the first vacuum-filter plant erected on these gold fields, but that it should occupy the site of the first slime plant designed and erected on the Rand by Mr. J. R. Williams in 1894.

The Butters vacuum-filter plant consists of two stock vats equipped (temporarily) with air agitation to keep the slime in suspension. After the gold has been dissolved in Brown agitators, the pulp at a dilution of 2 to 1 is transferred to the stock-pulp vats from which it is fed to the filters as required. There is one wash-solution vat. All precipitated solution is delivered to this vat to be used subsequently for a wash solution in the filters. There are two filter boxes each fitted with 150 Butters patent vacuum filter leaves and one 14″×14″ Gould vacuum pump. Each filter box is an independent filtering unit. One 10-inch Robeson–Davidson slime pump handles the pulp and solution to and from the filter boxes and the stock vats, serving the boxes alternately. The piping is arranged around the pump in a loop, having four valves interposed so as to reverse the direction of flow through the pipes. The pump is set in either the emptying or filling position instantly by means of one pilot valve introducing hydraulic pressure into the cylinders of the four valves simultaneously. The motor for the 10-inch slime pump and the valves in the pump systems are operated from the switchboard. The kind of filtering and treating cyanide slime is called the "pumping system," and the process of filtration is performed as follows:

To form the cake, the filter box is filled with pulp to a point over the top of the filter leaves, and a valve opened connecting the vacuum pump directly with the filters. Clear cyanide solution is drawn from within the filter leaves and delivered to the clarifying vat for precipitation, the slime remaining as a cake on the outside of the filters. The filters are kept submerged by refilling the box at intervals, and jets of air are introduced at the points of each hopper to keep the slime in suspension. When a cake 1 inch to 1½ inches thick has been formed, the surplus pulp is pumped back to the stock-pulp vat, and the box is then filled with solution to wash the cake. During the time of forming and washing the cake, the vacuum is maintained at the highest possible point, but when the cake is exposed to air during the transfer of pulp and wash solution, the vacuum is reduced to 5 inches to prevent the cake cracking.

The cake being formed by atmospheric pressure, the resistance and permeability is equal over the whole surface of the leaf, thus producing an ideal condition for the recovery of the valuable moisture contained therein, which equals 30 per cent. to 35 per cent. Sufficient wash solution (about 2 tons per ton of slime) is drawn through the cake by the vacuum pump to effect a complete displacement of the original moisture. A portion of this solution goes to the clarifying vats for precipitation; the flow is then diverted to a stock vat to be used in making up new charges in the Brown or Pachuca treatment vats.

When the wash is complete, the vacuum is disconnected and a reverse flow of solution (by gravity from the vat on the roof) is introduced to the interior of the filters, causing the cakes to drop. The surplus solution is then pumped from the filter box through decanters and returned to the wash solution vat. The mass of thick sludge remaining in the box is diluted with water and agitated with air for a few minutes to make a homogeneous pulp of 1 to 1, which is then pumped to the residue dam by a second 10-inch pump, the delivery pipe of which is fitted with an automatic sampler. One complete cycle of operations occupies the following times: Filling box with pulp and forming cake, 45 minutes; transferring and washing and forming cake, 70 minutes; discharging and washing and forming

*From Journal of the Chemical, Metallurgical and Mining Society of South Africa.

cake, 20 minutes; total time of cycle, 2 hours and 15 minutes; tons treated per cycle, 1¼-inch cake, 50 tons each box.

Five leaves are removed from each filter box every day, thoroughly washed with a spray of water and air, and then immersed in a vat of 2-per-cent. hydrochloric acid solution, for the purpose of dissolving the calcium carbonate and keeping the leaves soft, pliable, and in good working condition. The *HCl* solution is passed through the leaves in reverse directions alternately by means of an automatic arrangement that requires no power or attention. The leaves are removed from the acid bath after 4 to 6 hours' treatment, again washed with the spray of water and returned to the filter box.

In January a special test was made on current slime, treated first by the Adair–Usher process, the residues from which were delivered to the filter for the purpose of determining what further recovery of dissolved gold was possible by filtration. In this test, as also in the former operations with accumulated slime, the efficiency of the filter was fully demonstrated, assays of the residues showing consistently during the whole period over 99 per cent. recovery of dissolved gold.

There is some dissolving of gold, incidentally effected, during the process of filtration; however, the Butters filter is essentially a slime-washing machine, purely mechanical in its functions, and might be termed a process of forced decantation.

The plant was designed to treat 500 tons per day. Subsequent operations proved that, owing to the exceptional adaptability of Rand slime to filtration, it has a capacity of 900 tons per day.

The plant was put in commission on August 12, 1910, and after a most exacting test over a period of 6 months, has been formally taken over by the Crown Mines, Ltd., as a part of their permanent plant.

THE A B C OF ELECTRICITY IN MINES*

In an electric motor, the thing that does the work is the current; and the entire power plant and system is calculated to generate and force through the motors a sufficient amount of current to make them do their work.

Definitions. Drop in Voltage. Calculation of Resistance of Conductors

The current of electricity is in many respects like a current of water. As water flows through a pipe, so the electric current flows through a copper wire. As the walls of the pipe retard the flow of water by friction, so a copper wire retards the flow of the electric current by friction or resistance. The great difference is that water flows through the hole in the pipe, while the electric current flows through the body of the metal itself. In order to guide the electric current to its destination, the copper wire or "conductor" is surrounded by a "non-conductor," or an "insulating" material; that is, a material through which the electric current will not flow. Air is a non-conductor, so that a bare wire in air will carry a current through its length. Water is a conductor, so a bare wire in water will allow the current to flow in any direction.

As the resistance to the flow of water varies with the length and size of the pipe, so the resistance to the flow of the electric current varies with the length and size of the "conductor" wire.

The pressure, called the "voltage," which forces the electric current to flow through a wire, corresponds closely to the pressure from the pumps of a water system. If the water pipes are full of water, but the water is not flowing, the pressure at the end of the line will be the same as at the pump. But as soon as the water begins to flow, the pressure at the end of the line drops, the amount of the drop depending on the amount of water flowing. So, if there is no current flowing in a trolley wire, the voltage at its far end will be the same as at the generator. If current is being used at the far end of the trolley line, the voltage will drop at the end of the line, the "drop" depending on the amount of current flowing.

A difference between a current of water in a pipe and a current of electricity is that the water flowing out of the end of the pipe is used and thrown away; while the electric current flows through the motor and then returns to the generator. That is, the same current flows round and round the circuit. So in order to get the total resistance of the circuit, the resistance of the rail return must be added to the resistance of the trolley circuit from the generator to the motor.

In order to be able to figure definitely how much copper is needed in a circuit, the resistance of the copper is measured in units called "ohms." 00 wire has a resistance of .0778 ohm per 1,000 feet, 000 wire of .0617 ohm per 1,000 feet, and 0000 wire of .0489 ohm per 1,000 feet.

The drop in voltage in the line is equal to the number of amperes flowing multiplied by the number of ohms resistance in the circuit. For instance, if a motor using 200 amperes current is running at a distance of 4,000 feet from the generator, and if the track return circuit has one-half the resistance of the trolley wire, which we will say is 0000, then the total resistance would be

$$4\times.0489+\frac{4\times.0489}{2}=.2934 \text{ ohm,}$$

and the voltage drop would be

$$200\times.2934=58.68 \text{ volts.}$$

If the generator holds 260 volts, this would leave 201.32 volts to run the motor. If a cutting machine using 100 amperes is running at the same time, the drop in voltage would be

$$300\times.2934=88.02 \text{ volts,}$$

and would leave but 171.98 volts for the motors. If the motors will not operate well on less than 200 volts, it would be necessary to add some feeder wire. Suppose a 0000 feeder wire be added. That would reduce the trolley resistance by one-half, and the total resistance would be

$$\frac{4\times.0489}{2}+\frac{4\times.0489}{2}=.1956 \text{ ohm,}$$

and the voltage drop would be $300\times.1956=58.68$ volts, and would leave 201.32 volts to run the motors. If the load be increased, still, it will become necessary to add more feeder wire.

To obtain the resistance of two conductors "in parallel," that is, placed side by side and connected at intervals so that they assist each other to carry the current in the same direction, divide the product of their separate resistances by their sum. Thus, the resistance of a 0000 wire ($R=.0489$ ohm per 1,000 feet) "in parallel" with a 500,000 C. M. feeder ($R=.021$ ohm per 1,000 feet) is

$$\frac{.0489\times.021}{.0489+.021}=.0147 \text{ ohm}$$

per 1,000 feet. The resistance for 4,000 feet would be

$$4\times.0147=.0588 \text{ ohm.}$$

Assuming the track resistance to be the same as above assumed, .0978 ohm, the total resistance would be

$$.0978+.0588=.1566 \text{ ohm,}$$

and the voltage lost in the wire and rail with 300 amperes flowing would be

$$300\times.1566=46.98 \text{ volts.}$$

In adding feeder copper to a circuit, it is very important that it be added at the right point. For instance, suppose the trolley resistance is 1 ohm and the track resistance is 2 ohms. If the trolley is 0000 wire and a 0000 feeder wire be added to it, the trolley resistance would be ½ ohm and the total resistance 2½ ohms If, however, the 0000 feeder had been added to the track, its resistance would then be

$$\frac{1\times2}{1+2}=\tfrac{2}{3} \text{ ohm,}$$

or the product of track resistance and copper resistance in parallel, divided by their sum, and the total resistance 1⅔ ohms. This change in location of the feeder thus reduces the resistance by ⅚ ohm, or 33⅓ per cent.

* By W. Kempton, Eng. Dept. Ohio Brass Co.

In short, the trolley and feeder resistance should be kept as nearly equal to the track resistance as possible. In the case cited in the second paragraph above, the next addition of copper should be made to the track, for its resistance is now more than the trolley.

To summarize, if a mine operator finds in operating his motors any difficulty on account of lack of power he should take a voltmeter into the mine and read the voltage between the trolley wire and rail while all motors are running at full load. Better still, take a voltmeter on the motor and read the voltage during its run, noting voltage particularly at points of poorest operation. If the generator maintains the voltage about constant and the drop is excessive, say 30 per cent., the trolley and track resistance is responsible.

To determine if the track resistance is excessive, two methods may be followed. One is to get the full load current rating of the various machines used in the mine from the makers. From this and the number of various machines and motors running at once from the trolley circuit, determine the largest amount of current carried by the circuit at one time. From the size and length of the trolley wire and feeders the resistance can be figured. Then by multiplying the number of amperes above by the trolley circuit resistance, the voltage lost in that part of the circuit will be obtained. This value subtracted from the total maximum drop will leave the voltage lost in the rail circuit. The rail loss divided by the current will give the rail resistance. If the rail resistance is more than .024 ohm per 1,000 feet, the bonds should be gone over and the poor ones replaced.

A more accurate method of getting the resistance would be to take an ammeter of sufficient capacity and connect it into the trolley line, say at a section insulator point with switch on the wall, and read the current and voltage at the same instant. The readings must be taken on the main entry at a point where all current used at that instant is flowing through the ammeter. The point selected should be as far from the mouth of the mine as possible, and the trolley and track length should be measured from this point to the generator.

The first method, though less accurate, will probably be the more practical for the reason that an ammeter of the proper capacity will be harder to secure than the voltmeter, as every mine having electric haulage should be provided with a portable voltmeter.

If it is found that the rail resistance is high, a bond-testing set should be secured and every bond tested. Those having too high a resistance should be replaced by good ones. The importance of good bonding is often overlooked; but when it is found that the difference between poor and good bonding often represents the difference between the addition or omission of an extra feeder, its importance will be appreciated.

PHOSPHORUS POISONING

The results of an investigation of phosphorus poisoning in American match factories are presented in Bulletin No. 86 of the Bureau of Labor of the Department of Commerce and Labor. This investigation was recently made by Dr. John B. Andrews, secretary of the American Association for Labor Legislation, in cooperation with the Bureau of Labor.

The manufacture of matches where white phosphorus is used, as in all American factories, exposes all employes who come in contact with the phosphorus or its fumes to the dangers of phosphorus necrosis. The phosphorus most frequently attacks the jawbones and not infrequently necessitates the removal of an entire jaw by surgical operation. It is the experience of all factories and all countries, that so long as white phosphorus is used the danger cannot be entirely eliminated, although it may be much diminished by thorough ventilation, and by the rigid enforcement of preventive measures.

Detailed investigation of 15 match factories showed that 65 per cent. of the employes were working under conditions exposing them to the fumes of phosphorus and the dangers of phosphorus poisoning. The employment of women and children is such that they are much more exposed to the dangers than are the men. Ninety-five per cent. of all the women and 83 per cent. of the children under 16 years of age were exposed to the dangers of the poison. The total number of employes in the 15 factories investigated was, according to the statements made by the manufacturers, 3,591, of whom 2,024 were men, 1,253 were women 16 years of age and over, while children under 16 numbered 314, of whom 121 were boys and 193 were girls.

A movement, which had been growing for a number of years, for the prohibition of the use of white phosphorus culminated in 1906 in the international convention of Berne, called on the initiative of the International Association for Labor Legislation. As a result of this convention the leading countries of Europe joined in a treaty to prohibit the manufacture, importation, and sale of matches made of white phosphorus.

Great Britain for a while held aloof from the movement, endeavoring to control the danger by the strictest kind of legal regulation. After a trial of a number of years the effort was admitted to be a failure, and legislation was, therefore, enacted which became effective the first of the present year, by which Great Britain joined the alliance of the countries prohibiting the manufacture, importation, or sale of white phosphorus matches.

A harmless substitute for white phosphorus, that is commercially practicable, has been found in the sesquisulphide of phosphorus, and is largely in use abroad in the manufacture of the "strike anywhere" match. For the safety match, red phosphorus, which is not poisonous, is generally used.

The United States is practically the only commercial country of any importance which has not taken any step to prevent the unrestricted use of this poisonous substance in the manufacture of matches.

While several states have enacted laws prohibiting the employment of children under 16 years of age in certain operations in match factories, no state has yet made any adequate provision for the protection of the health of workers over 16 years of age in the match industry, although for over 50 years the dangers of working with white phosphorus have been known and recognized in this country.

The owners of the harmless substitute for white phosphorus in this country have since the above was sent out by the Bureau of Labor offered the use of their patent free of royalty and without stipulation to all match factories in the United States.

TREATMENT OF COBALT ORES

There is still much to be discovered in the treatment of Cobalt silver ores. During 1910 the Canadian Copper Co. treated 4,900 tons of silver ore averaging 1,730 ounces to the ton. The product of this smelter is speiss and base bullion, the latter being retreated until it is ready to be shipped as refined silver. The speiss is milled and desilverized by a wet process. Crude cobalt material is obtained as a residue which is in the usual form purchased by cobalt manufacturers. As a result of improvements on this plant the Canadian Copper Co. is shipping refined silver, refined arsenic, and a marketable cobalt product. The changes have increased the capacity of the plant to over 800 tons per month, so that it has a monthly output of silver of over 1,000,000 ounces. Another result of the enlargement of this plant and the modifications in the process followed is the quicker return made shippers, the smelter people being able to pay for 70 per cent. of the silver in 30 days and for the remaining 30 per cent. in 60 days.

The Coniagas Reduction Co. consider that white arsenic and cobalt are as profitable as formerly. In other words, they state that at 2½ to 3 cents arsenic is unprofitable, but as they are obliged to handle it, it is a profitable by-product. The Reduction Works are paying 8 cents a pound for cobalt when ores assay 6 per cent. or more, 10 cents for 8-per-cent. ores, and 12 cents for 12-per-cent. or more.

PROSPECTING POVERTY GULCH CLAIMS

By Charles W. Henderson

(Continued from May number)

Surface Indications. Prospecting With Trenches and Underground. Materials Used. Cost and Progress

Surface Indications.—The veins in Cripple Creek, Colo., more particularly those of phonolite, have a similarity with the dikes in their strike and dip, thus suggesting some sort of relationship between them. Owing to soil and talus covering the deposits, surface outcrops are of rare occurrence, making it necessary to trench, and it is proposed to uncover two typical Cripple Creek dikes dipping in opposite directions, with the vein following their intersection at an angle of 23 degrees dip, though typical Cripple Creek veins are nearly vertical. The basic dike is exposed in Poverty Gulch where the railroad crosses, and also at the head of the creek which empties at the southwest corner of the Golden Cycle claim. The distance between these outcrops is 1,500 feet, and between these two points the dike is covered with soil from 10 feet to 15 feet deep. The dike at the outcrop is from 5 feet to 10 feet wide and seems to continue in a northerly

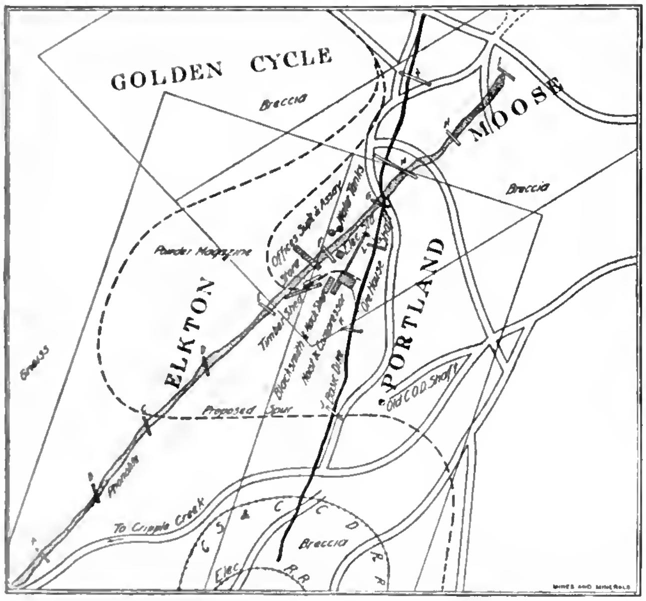


Fig. 1. Map Showing Location of Mine

direction. It is decomposed and not easily distinguished from andesite breccia, which is the country rock of Cripple Creek, for the most part at least.

On the road to Cripple Creek there is a hard compact phonolite dike striking N 40° E and dipping 60° in a N 50° W direction. This dike is readily identified, for which reason the first prospecting work is to prove its existence, by trenching and tunneling; work is therefore started at *A*, Fig. 1, and it is proposed to make trenches at intervals of 100 feet along the outcrop. During the course of the work itemized costs are to be kept for future reference and to show profit and loss of the undertaking.

Trench *A* 60 feet long is excavated 6 feet wide by 7 feet deep without any indication of bed rock. This work was done by 6 men with picks and shovels in 3.2 days and the following data were obtained:

$\frac{60' \times 7' \times 6'}{27} = 93.4$ cubic yards, material handled; or $\frac{93.4}{3.2 \times 6 \times 8}$ $= .6$ cubic yard per man per hour or 4.8 cubic yards per man in 8 hours. As men cannot throw dirt with a shovel to advantage more than from 4 to 6 feet from the trench, a staging was put in, the trench narrowed to 3 feet, and 3 additional men hired to rehandle the dirt. The capacity of the 9 men shoveling with two throws was equal to 6 men shoveling with one throw, but the cost was increased 50 per cent. for labor. The second stage was excavated in $\frac{60' \times 8' \times 3'}{27 \times 4.8 \times 6} = 1.85$ days.

The cost of excavating 146.7 cubic yards of earth from the trench was as follows:

Six men 3 days @ $3	$54.00
Nine men 2 days @ $3	54.00
Six round-point long-handled shovels	9.75
Three square-point D-handled shovels	4.00
Six 5-pound drift picks	8.00
Six 36-inch drift pick handles	1.90
172 feet lumber for staging	3.10
	$134.75

Cost per cubic yard, $.918.

The results obtained from this trench decided the feasibility of prospecting by a series of cross-cut drifts spaced 250 feet apart. The approaches were dug in the soil on the side hill until a face 10 feet square was obtained. The sides of this cut were given a 1 to 1 slope to prevent their falling in, and a drift set was erected at the face to prevent the portal caving. Three drifts were started with 9 men, three at each drift, who completed the approaches in 2 days, from which it was ascertained that 2 men with pick and shovel and 1 man wheeling could do about as much work as 2 men in a trench. When tunneling in soft earth a man can pick, shovel, wheel and waste, when the run is not long, say 25 feet, about $\frac{3}{4}$ as much as he can shovel and pitch from a trench, or $\frac{4.8 \times 3}{4} = 3.6$ cubic yards. When a rock face was reached, 2 men at each drift were put at work with single hammer and drills, while the third man of the party was occupied in mucking, carrying steel from the forge, etc. A blacksmith was hired at $4.50 a day, who also superintended the work and did the panning and sampling as the work progressed. Each tunnel was driven through the dike and through the ore in drifts *B* and *C*, and side drifts were driven 10 feet in the direction of the strike up the slope. There was 85 feet of drifting done in the prospect drifts *B*, *C*, and *D*.

In drifting, 11 holes were drilled in each face $2\frac{1}{2}$ feet deep, making an advance of 2 feet for each round. Each man was able to drill 1 foot an hour, and with six drilling, two in each tunnel, each 2-foot advance took 11 holes, $2\frac{1}{2}$ feet deep, or $27\frac{1}{2}$ feet. Two men in each tunnel drilled 16 feet a day, making a new round every day and a half by working from 7 a. m. to 1 p. m. in the morning shift, so that shots could be fired at the noon hour. Each shot broke down 2.6 cubic yards, which was readily mucked and wheeled by the laborers.

The following data were obtained from this work: $\frac{85'}{2'} = 43$ rounds of shots that demanded 27.5 feet of drill hole for each round. Since 6 men drilled 48 feet per day the advance required $\frac{27.5 \times 43}{48} =$ approximately 25 days. In all, there were 473 holes drilled, and each was charged with $2\frac{1}{2}$ sticks of $\frac{7}{8}'' \times 8''$ dynamite weighing .25 pound each or .625 pound per charge. The total quantity of explosives used was $\frac{473 \times .625}{50} = 6$ fifty-pound boxes.

The diameter at the mouth of a 6-foot hole need not be more than $\frac{3}{8}$ inch greater than at bottom, with $\frac{1}{8}$-inch reduction in 2 feet possible in very hard rock. The least admissible diameter of hole at bottom for dynamite is $\frac{3}{4}$ inch. The steel used was $\frac{7}{8}$ inch octagonal with $1\frac{1}{4}$-inch starting bits decreasing to 1-inch bits to finish. The drills were dulled every 2 feet of drilling with usually two starters required for the 2 feet.

The following lengths comprised a set of steel: 3 feet, $2\frac{1}{2}$ feet, 2 feet, $1\frac{1}{2}$ feet, and 10 inches; and each man in drilling 8 feet a day put in a little over 3 holes, using a set of steel in the operation. This made, for the six men working, six sets of

5 or 6 pieces each, or from 30 to 36 pieces of steel to be sharpened a day.

COST OF DRIVING THE CROSS-CUTS

One foreman 25 days @ $4.50	$112.50
Nine laborers 3 days @ $3	81.00
Three laborers 25 days @$3	225.00
Six miners 25 days @ $3.50	525.00
Three wheelbarrows @ $13.50	40.50
Six $4\frac{1}{2}$-pound striking bammers	10.80
Six 12-inch hammer bandles	.60
344 pounds, 132 feet $\frac{7}{8}$-inch steel	20.58
300 pounds dynamite @ $.101 per pound	30.46
1.5 tons charcoal @ $20.75	31.12
171 feet lumber for portals	3.42
One portable forge	42.00
One 90-pound anvil, steel faced	9.00
One set blacksmith tools	9.00
600 candles @ $.0145 each	8.70
172 assays, four for each round	57.00
473 fulminate caps, 85 cents per 100	4.02
1,419 feet double tape fuse @ 65 cents per 100 feet	9.22
	$1,219.72

When opening cross-cuts *B* and *C*, 10 feet were driven on ore that assayed 1.209 ounces of gold per ton and which amounted to 42 tons. This was hauled to the railroad, shipped to a smelter which purchased it at $18.13 per gross ton. The smelter return was as follows:

Gross Pounds	Moisture Per Cent.	Net Pounds	Ounces Gold Per Ton	Value Per Ton	Total Value
84,000	2	83,832	1.209	$24.18	$1,013.53
Charges freight and mill treatment, $6 per ton					252.00
Net					$ 761.63

From this is to be deducted the cost of haulage to the railroad or $18, leaving a profit of $743.53.

From the prospect trench and drifts, the following information was gleaned for future prospecting: Trench *A* showed no ore; neither did tunnel *D*; tunnels *B* and *C* opened a vein dipping 60 degrees, varying from 0 feet to 10 feet wide, and apparently 250 feet long between the two tunnels and extending some distance in the directions of *A* and *D*. The ore found assayed 1.209 ounces in gold per ton and this was sufficiently encouraging to warrant further prospecting, and owing to the flatness of the surface, trenches *E*, *F*, *G*, *H*, and *I* were started to cut the phonolite dike. Additional men were employed to push the work, and it was found that at *G*, where the basic dike comes in contact with the phonolite dike, there was a large vein of ore which was so favorable that it was decided to prospect the basic dike. No ore was discovered at *H*.

To excavate the five trenches required the labor of 5 men 20 days, and cost $630. On the basic dike, trenches *J*, *K*, *L*, *M*, *N*, *O*, *P* were cut, in all a length of 700 feet, that took up 47 days' time of 10 men and cost $1,495.

The assays derived from samples taken at *G*, *L*, and *M* were so favorable it was decided to prospect them deeper. At *G* the vein was followed down on the phonolite dike 60 feet in ore. Sinking was also carried on at *A*, *B*, and *C* in ore, and in all shafts at a depth of 60 feet on the dip levels were started.

The sinking was done by hand with the aid of a windlass and bucket, from which operation the following data were obtained.

Shafts 5 ft.×7 ft. required 12 holes per round, average $2\frac{1}{2}$ feet deep, drilled in 12.5 hours, $3\frac{1}{2}$ hours in two shifts were consumed in loading, firing, and cleaning up broken material.

On the first day two men drilled 30 feet of holes, which were loaded and fired. On the second day, two men loaded and hoisted for 3 hours, and were followed by two miners who drilled 25 feet of holes, which were loaded and fired. An average daily advance of 1 foot 7 inches was made in this way, and at every 6-foot advance a set of timbers was placed. It took 38 days to sink a shaft 60 feet in this way, and the cost which was $17.12 per foot was made up of the following items:

	Total Cost	Cost Per Foot	
Powder, caps, and fuse	$170.84	$ .95	
Labor (miners and muckers)	1,824.00	10.12	
Labor (blacksmith and bosses)	323.00	1.79	
Labor (timberman)	171.00	.95	$2.39
Timber	259.92	1.44	
Steel (depreciation)	9.36	.05	
Candles	26.45	.15	
Coke	98.60	.55	
Windlass, rope, and buckets	128.55	.72	
Shovels and picks	71.50	.40	
Total	$3,083.22	$17.12	
Eliminating windlass, rope, buckets, shovels, picks		1.12	
		$16.00	

Extra men were hired to drive the main drifts in 500 feet and make 7 cross-cuts averaging 15 feet. In *B* two drifts were carried 700 feet, and 9 cross-cuts, averaging 15 feet, were excavated. In *C* 4 drifts totaling 800 feet, and 8 cross-cuts, 25 feet each, furnished an advance of 1,000 feet. In all, 2,240 feet of drifting was accomplished. The estimation of 2,440 feet of track (using 12-pound rails at $47 per ton with fish-plates) is based on the cost per mile as follows: 364 pairs of splice bars; number of bolts, 1,456; weight of splice bars, .56 gross ton; weight of bolts, .15 gross ton; weight of spikes, .77 gross ton. Total weight of fastenings, 1.48 gross tons.

Weight of rails, 18.86 gross tons. Total weight of rails and fastenings, 20.34 gross tons per mile, then $\frac{2,440}{5,280}\times 20.34=9.4$ gross tons, at $47 = $441.80.

The 814 ties, spaced 3-foot centers, were 2 in.×4 in.×$2\frac{1}{2}$ ft., and cost $27.14.

In order to hasten the handling of broken rock and ore, and before drifting was commenced on the levels, three small electric hoists were purchased. This was necessary because the limit of a windlass is about 60 feet for economical work. The heaviest load to be hoisted at one time was estimated at 850 pounds, and hoisters were ordered capable of raising 1,000 pounds 200 feet in one minute, and motors having 8 horsepower were selected. The drums on the hoisters being 18 inches diameter, a $\frac{1}{2}$-inch steel wire rope was ordered with them.

It was found that 1 cubic yard of the Cripple Creek ore when broken occupied 1.73 cubic yards, also that while 12.4 cubic feet of solid ore weighed a ton, it took 21.5 cubic feet of loose ore to weigh a ton. The breccia is not as free drilling as granite, and sludge is made rapidly in drilling, however, the rock breaks better than granite. The number of tons of material excavated in driving the 2,440 linear feet of levels was:

$$\frac{2,440\times 7\times 5}{12.4}=6,890 \text{ tons}$$

In drifting, each man drilled 1 foot per hour, and with 11 holes in the face, each 28.5 inches deep, an advance of 22.5 inches was made each round. The material broken was ore and the amount to be handled was estimated to be as follows: $\frac{22.5''}{12''}\times 7'\times 5'=65.8$ cubic feet solid, or $65.8\times 1.73=113.8$ cubic feet broken ,per round. As each bucket holds 4.5 cubic feet, about 26 buckets will carry out the ore broken in one round. According to Gillette* one man can shovel 1.2 cubic yards per hour from the ground and 1.4 cubic yards per hour from steel, which gives one man 3.2 hours' time to shovel the material broken in one round. If 4 minutes' allowance is made for tramming and changing buckets per trip it will take one man 104 minutes to tram 26 buckets. From the above it is ascertained that an advance of 56.31 feet can be made in each drift per month, and that the 8 drifts are advanced $\frac{22.5''}{12''}\times 16$ hr. $=15.02$ ft. per day, consequently it will require $\frac{2,440}{15.02}=162$ days

*Gillette's Handbook of Cost Data.

to drive, hoist material, lay rails etc., making the cost of the work as itemized.

	Total Cost 2,440 Feet	Cost Per Foot	
Labor (drillers and muckers)	$15,552.00	$6.37	
Superintendence	1,377.00	.56	
Steel $ 164.75; Blacksmith and helper (labor) 1,296.00; Coke 420.19	1,880.94	.78	
Powder, fuse, and caps	2,062.78	.85	
Candles	300.67	.12	
Power	35.05	.01	$1.80
Hoistman	4,374.00	1.79	
Hammers $ 30.20; Steel plate 88.13; Rails, ties 468.94; Cars 192.00; Ore buckets 192.00	971.27	.40	
	$26,553.71	$10.88	
Leaving out cost of new steel and other supplies, and omitting depreciation	1,236.00		
Cost	$25,317.71	$10.37	
Cost (less supplies and less superintendence)		$9.81	

The appliances necessary for drifting and their cost are as follows:

Three head-frames (lumber)	$ 87.96
Three 24-inch sheaves and blocks	57.00
Three hoists	900.00
Three motors @ $230 f. o. b.	690.00
600 feet of ½-inch cast-steel rope	25.74
Freight on hoists 4,500 pounds at $.80 per 100	36.00
Freight on motors, 2,400 pounds at $1.75 per 100	42.00
Freight on sheaves, 360 pounds at $.65 per 100	2.34
Freight on rope, 234 pounds at $.55 per 100	1.29
Total	$1,842.33

The following points have been determined by the prospecting:

Shoot	Length Feet	Width Feet	Average Width Feet	Average Value
A	500	0–12	3	$15.00
B	700	0–10	3	$13.00
C	300	0–14	16	$20.00

Shoot *A* dips 50 degrees with the basic dike to the east; shoot *B*, 60 degrees with the phonolite to the west. At *C*, the junction of the two dikes, and the general dip of ore body is found to be at an angle of 22 degrees 42 minutes. The prospect shaft at *G* on shoot *C*, going down on the ore body which lies in the dip of the phonolite dike (60 degrees) runs into barren rock at a depth of 60 feet, though the drifts still show ore for 300 feet along each dike, the richest ore is at the intersection of these dikes, 150 feet from the foot of the shaft.

To continue the development work a winze was sunk from the first level (60 feet) at the end of the drifts, and at 126 feet the winze passed into barren rock, having passed through the intersection of the two dikes and finding ore the length of the winze.

The vertical distance down, now 176 feet, gives 514 feet along the intersection, and indicates 206 feet of backs of probable ore, 300 feet long, on 60 degrees or phonolite dike, and 231.5 feet of backs of probable ore, 300 feet long on the 50 degrees or basic dike. There are moreover, 60 feet and 69 feet, respectively, of backs of blocked out and proved ore, less 17.3 feet and 19.8 feet, respectively, to compensate for soil 15 feet.

The ore is estimated as follows:

$$\frac{[300\times(60-17.3)\times 16]+[300\times(69-19.8)\times 16]}{12.4}$$

$$=\frac{205,000+230,000}{12.4}=\frac{441,000}{12.4}=35,500 \text{ tons (proved and blocked}$$

out) above 60-foot level.

$$\left(\frac{300\times 126\times 16}{2}\times\frac{1}{24}\right)+300\times 82\times 16\times\frac{1}{24}=56,100 \text{ tons blocked}$$

out below 60-foot level.

$$35,500+56,100=91,600$$

$$\frac{(300\times 206-17.3\times 16)+[300\times(231.5-19.8)\times 16]}{12.4}$$

$$=\frac{906,000+1,015,000}{12.4}=\frac{1,920,000}{12.4}=154,900 \text{ tons,}$$

total blocked out and partly blocked out, including above 91,600.

$$\frac{(300\times 146\times 16)+(300\times 162.5\times 16)}{12.4}$$

$$=\frac{700,000+780,000}{12.4}=\frac{1,480,000}{12.4}=119,400 \text{ tons}$$

(probable ore between first and second levels).

This condition requires that 200 tons of ore be mined per day, which is a little under the operations of the Portland Gold Mining Co., but a good average production.

200 tons a day for 18 months or 540 days = 108,000 tons.

Allowing 10 per cent. for pillars, the 154,900 tons (blocked out) would give 139,410 tons.

The Portland Gold Mining Co. has produced its gold at a cost of 70 per cent. per ounce, and a profit of 30 per cent. per ounce. (The value per ton has been above 1 ounce, however.) The 91,600 tons of ore blocked out and carrying 1 ounce per ton, at a profit of 30 per cent. per ton, anticipates a profit of $\frac{(91,600\times \$20.6718\times 3)}{100}=\$56,801.16$, and 139,410 tons would give a profit of $(139,410\times \$6.20)=\$864.342$.

Whether this ore can be mined in time to make $864,342 a paying investment on the money invested; and also to find if the ore can be produced at a profit of $6.20 per ton, are two items for consideration.

The data collected during the operation of sinking the winze 126 feet with the help of one of the electric hoists placed on the 60-foot level, give the cost of the winze as follows:

Cost of Winze

Two men day shift, 80 shifts, at $4	$ 640.00
Two men night shift, 80 shifts, at $4	640.00
One hoistman and timberman, day shift, 80 shifts, at $4.50	360.00
One boss, hoistman, and blacksmith, day shift, 80 shifts, at $4.50	360.00
Powder, 12 holes, 80 rounds, 960 holes, each hole 2½ sticks (.625 pound.), 600 pounds, at $.155	93.00
Caps, 960 holes, at $.85 per 100	8.16
Fuse, 960 × 7 ft. = 6,720 feet, at $.65 per 100 feet	43.68
Candles, 1,914 at $.0145	27.76
Timber, 72 bd. feet per foot of shaft, 9,072 bd. feet, at $20 per thousand	181.44
Coke for blacksmith, 5 tons at $20.75	103.75
Power, 1.186 horsepower + .303 = 1.489 horsepower per minute used every 1¾ minutes, or 2.5 horsepower hours every hour, say used 8 hours out of 16 hours, 20 horsepower hours per day. Total, 80 days, 1,200 kilowatt hours, at $.013	15.60
Total, for 126 feet	$2,473.39
Cost per foot	$19.64

Most of the figures making up the cost sheets are Cripple Creek figures, including freight, etc. There were, however, 28,303.55 pounds of material bought in Denver, which cost $.65 per 100 pounds, and 2,044 pounds of freight at $.75 per 100 pounds, the total costing $199.30.

Expenditure for Prospecting

Surface prospecting	$ 3,505.37	
Underground prospecting	33,952.65	$37,458.02
Additional freight charges		199.30
Total		$37,657.32

Time occupied in surface prospecting was 100 days, and in underground prospecting 280 days, a total of 380, or 1 year 15 days.

During prospecting, ore was shipped almost from grass roots.

On shoot *A*, 45 feet of shaft and 500 feet of drifts were in ore. The ore in the shaft averaged 5 ounces gold per ton,

and in the 700 feet of drifts, the average was .725 ounce. The average width of the shoot was 3 feet.

On shoot *B*, the ore in the 45 feet of shaft ran 4 ounces in gold per ton, and the 700 feet of drifts averaged .628 ounce gold. The average width of the shoot was 3 feet.

On shoot *C*, 45 feet of shaft averaged 6 ounces of gold, going as high as 8 ounces; on the 60-foot level 600 feet of drifts averaged 1½ ounces, and in the 125-foot winze averaged 2 ounces. The width of the shoot averaged 16 feet.

Smelter returns for ore shipments I and V are given to show the forms used.

Shipment I, to U. S. R. & R. Co., at Colorado City
(Payment at rate of $20 per ounce)

Gross Weight Tons	Moisture Per Cent.	Net Weight Tons	Assay *Au* Ounces	Value Per Ton	Total Value
43.9	4	42.144	5.000	$100.00	$4,214.40
Freight and treatment, $8.50 per ton..................					373.15
					$3,840.25

Shipment V, to A. S. & R. Co., at Denver
(Payment at rate of $19.50 per ounce)

Gross Weight Tons	Moisture Per Cent.	Net Weight Tons	Assay *Au* Ounces	Value Per Ton	Total Value
976.0	4	936.96	5.000	$117.00	$109,624.32
Freight and treatment, $5.50 per ton..................					5,368.00
					$104,256.32

Receipts From Ore Sales

Shipment I,	$ 3,840.25	532.9 tons from *A*.		
Shipment II,	4,361.88			
Shipment III,	2,998.37	727.9 tons from *B*.		
Shipment IV,	4,827.40			
Shipment VI,	1,992.72	1,063.4 tons from *C*.		
Shipment V,	104,256.32			
Less hauling to railroad, 2,324.2 tons, or 1,850 cubic yards, in 1-yard wagons, 1 hour per trip, 1,850 trips; equivalent to 1 team and driver, 231 days, at $6..			$1,386.00	
Less umpire assaying, 930 assays, at three for $1..............................			310.00	
			$1,696.00	
Net receipts......................................				$120,580.94
Receipts of ore sales in prospect tunnels............				743.53
Total receipts ore sales............................				$121,324.47
Cost of prospecting................................				37,657.32
Cash on hand................................				$83,667.15

Plans are drawn for an incline shaft to follow the intersection of the dikes in the foot-wall. All work has been stopped on the shoots *A* and *B*; one hoist is used in the sinking of the incline shaft, while one is still used to take ore out of the prospect hole and to continue the exploration work.

It is assumed that, by the time the incline shaft has reached the first level, the plan for the surface buildings will be finished and their cost estimated, and that by the time the shaft has reached the second level, the contract will be let for the surface structures, and all mining equipment. This gives 210 days for surface buildings to be built and machinery to arrive while drifts and raises are being driven.

TAYABAS PETROLEUM

A sample of crude petroleum, typical of that found in the Tayabas Province, P. I., resembled Pennsylvania petroleum in containing a fairly high proportion of volatile hydrocarbons, in having a paraffin base and in being free from sulphur. The proportion of volatile constituents boiling below 150° C. was greater than in most petroleum oils: Sp. Gr. .831, initial boiling point 70° C.; first fraction (light oils) 70° to 150° C., 36.5 per cent.; second fraction (burning oils) 150° to 300° C., 48.75 per cent.; and residue, above 300° C. (by difference) 14.75 per cent. It contained 30 per cent. of unsaturated hydrocarbons which could be removed by acid. It might be described as essentially a paraffin petroleum. G. F. Richmond. (Philippine Jour. of Sci., v, I.)

DRIFTING WITH A STOPING DRILL

*By H. E. Moon**

It is interesting to note the ingenuity displayed by some miners, especially those who are leasing, in keeping down operating and equipment costs.

An example of this recently came to the writer's attention in the Cripple Creek district of Colorado. Two miners had a lease on one of the lower levels of the Granite mine. Their work was entirely stoping and raising, and their drill equipment consisted of air-feed stoping drills.

They drove the raises *A* and *B*, Fig. 1. Owing to heat and bad air and also to assist in opening the stope, they decided to drive the drift *C* from *A* to *B*, thus giving a chance for ventilation. They could not afford to purchase a drifting drill, but with the aid of a simple foot-plate they found that they could use a stoping drill to put in holes at all angles from the vertical to the horizontal. In one case they put in a hole 5 feet deep which dipped about 4 inches below the horizontal in that distance.

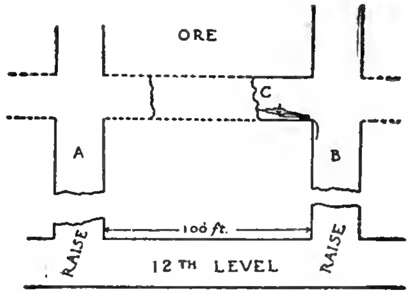
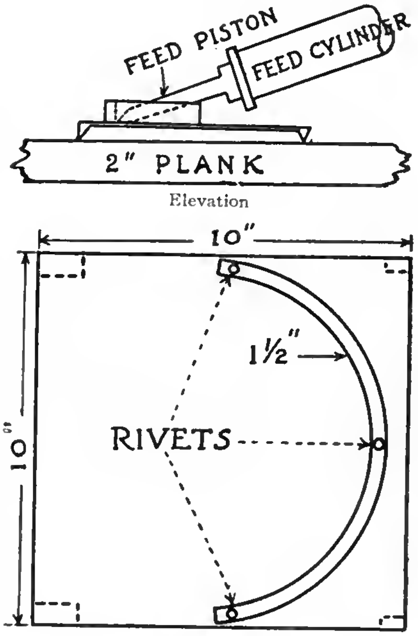


Fig. 1

This foot-plate, Fig. 2, was made of a sheet of ⅜-inch steel with ears turned down at the corners to act as spikes and hold the plate on to a plank. To this plate was riveted a piece of angle iron bent to a **U** shape.



Plan

Fig. 2

The drill used for this work was the new Sullivan stoper. The ground was a hard granite. The cutting speed was from 14 inches a minute, in vertical holes, to 8 inches a minute in horizontal holes. A blowpipe was connected with the air supply and used to clean the cuttings from flat holes.

[Sullivan stope drills are in use in the anthracite coal fields of Pennsylvania for work like that described above, for driving short tunnel headings. In this instance, hand-feed hammer drills are used to drill down holes in the face. The rock next the tunnel floor is then shot out, and the stoper used to drill upper holes.—Ed.]

* Reprinted from *Mine and Quarry.*

BLAIR'S ZINC-LEAD PIGMENT PLANT

Written for Mines and Minerals, by J. I. Blair

Before entering into a description of this plant attention is called to the principal considerations governing the design and construction. In this particular case the plant was intended as an addition to a regular lead smelter in order to treat the flue dust, blue fume, and a partly roasted complex zinc-lead ore that is common throughout the Western States.

A Plant Used in Addition to a Lead Smelter to Treat Flue Dust and Partly Roasted Ores

The capacity of the plant is to be approximately 144,000 pounds per day of 24 hours, of the material just mentioned and it is estimated that about 48,000 pounds of coal will be required per day. The word estimated is used because there has been introduced some carefully calculated data in the design of certain parts of this plant, although in the main it is the result of the experience in connection with sublimed lead and zinc oxide plants.

In the first design an attempt was made to take the hot fume from the trail coming from the blast furnaces and open hearths and, by drawing it through either an incandescent bed of coke or through an electric arc with the sulphurous fumes from the roasters, convert the fume directly into the desired white pigment. In this case the blue bag room would be converted into the white bag room and the cost of the plant would be probably 70 per cent. less than the cost of the regular plant as here outlined. The belief was that such a plan could be carried out successfully, from the fact that the fume in the "trail" of a lead smelter in the Middle States settled near the bag house in distinct layers, each one a fractional part of an inch thick, and throughout the bed of some 4 feet in thickness it was found that there were several layers of a relatively white fume between layers of lead-gray fume and blue dust. This arrangement showed that at some time during the smelting the conditions were realized for the formation of white fume. While it is to be hoped such a plant may become a reality, it has been decided that the better plan to follow, at least for the present, is the installation of the plant here described.

It has been claimed by experienced men in the sublimination of lead and zinc that it is practically impossible to produce the white fume from the blue fume alone. However, experiments show that if the requisite sulphur is added at the right time, basic sulphate will be obtained.

Another solution to the problem, that of briquetting the flue dust, was adopted by the St. Louis Smelting and Refining Co. with the result that only about 40 per cent. of the flue dust and blue fume was eliminated. The briquets were made, using the fluxes, lime or cement, as binders.

Fig. 1 shows the plan and side elevation of the zinc-lead pigment plant. To the left, not shown in the illustration, the coal bin is arranged, consisting of a simple shed and a floor made from the lumber that was used for scaffolding.

The railroad track at the coal bin is given a light grade and extends between 200 and 300 feet beyond the plant. The loaded cars are placed at the far end of the track and dropped down to the bin to be unloaded.

In choosing the type of furnace shown in Fig. 2, the considerations were first cost, cost of up-keep, and working costs. The slag-eye furnace although more compact is capable of treating more material and has the advantage of lower first cost than the furnace chosen, but has proved a troublesome furnace to run and requires considerably more skill in the furnace men to operate it properly. The blast necessary in this type of furnace is too great to be handled by a simple fireclay lined conduit here shown. Furthermore, the pigment is fused, choking up the mouth of the furnace and filling the conduit near the furnaces, making a shut-down, clean-out, and rebuilding of the combustion chamber imperative every few months.

For the small plant considered wheelbarrows are employed, and the charging is done by hand; for a large plant, a system of tram tracks override the plant and the charging is automatic from an overhead tram. The principal disadvantage of the furnaces shown in Fig. 1 is that considerable time and work is required to remove the clinker which remains on the grate after each heat. This source of loss could be reduced by using a movable grate bar worked by means of a lever, thus loosening the clinker and allowing it to drop into the pit below the grate, where it could be easily removed after the furnace has been recharged and in operation. The total grate area is 758 square feet, or 1 square foot per 1.5 pounds coal per charge of 4.5 pounds of material.

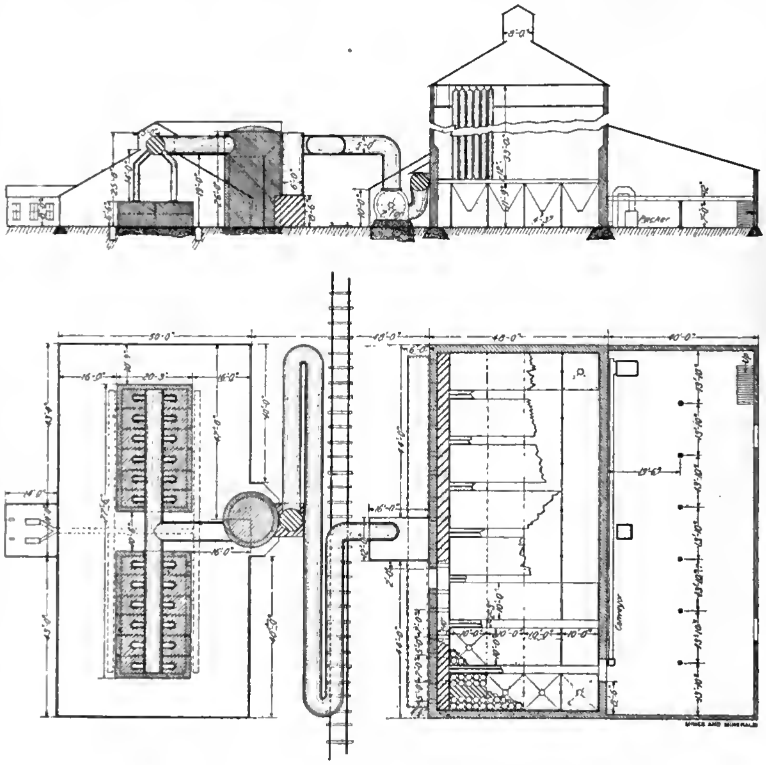


Fig. 1. Plan and Elevation of Zinc-Lead Pigment Plant

The vertical conduits directly over the furnaces are provided with "butterfly" valves (not shown in the drawing). These valves, which are opened during cleaning operations, are lined with a good fireclay. The conduits are made in sections so a burnt section can be removed and a new one inserted in a relatively short time. The size of the conduits is such that with the draft used the pigment and ash will be kept in suspension until they reach the tower, Fig. 3, where the heavy substances will drop out. From here the fume is kept in suspension until it reaches the large supply pipe situated along the side of the bag-house where, owing to the increase in volume and lower temperature, the remaining heavy ash and dust will fall into the short bags which are suspended from thimbles along the under side of the supply pipe (not shown in drawing), leaving the pure fume to enter the bags. This style of conduit has the advantage over a set of goosenecks, in lower cost and greater efficiency. It costs less because it is much simpler in design and requires less material. It is more efficient because of its horizontal position which admits of a greater area exposed to the cool atmosphere while with the goosenecks the heated vapor rises from the top of the hopper and from around the base of the

gooseneck and keeps the upper parts of the goosenecks surrounded with hot air, thus reducing the efficiency of their cooling effect. The overhead feature shown in the illustration makes it possible to use the ground beneath for other purposes. Thus the space occupied is reduced to a minimum. Small doors are provided along the conduit to admit air in order to regulate the temperature and velocity as occasion may require.

In determining the length of the conduit, calculations have been based on a study of the efficiency and properties of the larger type—600 feet—at Coffeyville, Kans., and also of the shortest—400 feet—at the new addition at Joplin, Mo., neither of which have given good results. The long trails cause too much pigment to settle out before it reaches the bags, while the short one burns up the bags and produces an inferior quality of pigment. With the increase in efficiency due to the peculiar arrangement of the conduit shown it is estimated that the 435-foot connection with the very low blast pressure adopted is about the length needed to furnish the desired results provided the exhaust fan is properly regulated. This, however, can be exactly determined only by actual operation.

By having the inlet to the tower on one side the inflowing gases swirl around the inclined walls and permit the heavier particles of ash to settle. The outlet to the tower is an arch. The tower is to be cleaned out each day during the shut-down for the bag shaking.

In designing the bag room it would have been optional whether the fume entered by way of the top by means of a set of pipes with thimbles and suspended bags as in the eastern oxide plants or through hoppers at the bottom, were it not for the quantity of sulphur gas evolved during the treatment of sulphide ores. The corrosive action of this gas on steel and iron work regardless of the protective coatings makes it advisable to use wood wherever possible, as it seems better adapted to the process than other materials. Each hopper is provided with a short sack to collect the pigment as it is shaken down. This is an improvement over the method of closing the mouth of the hopper in general use at some of the plants, which causes the pigment to cake and become lodged, and requires the men to pound on the hopper with pieces of wood to loosen and to cause it to drop. Treatment of this kind generally causes leaks in the riveted seams of the hopper. The suspended bags increase the filtering area of the bag room about 11 per cent. and to remove the pigment one has only to loosen the bag and replace it with an empty one.

Each row of hoppers is provided with a damper, making it possible to shut down a row during the repair of burnt bags. There is a suspended walk overhead where the bags may be tied, and a lower plank walk between the hoppers, shown in the illustration. Altogether there are 640 woolen bags 30 feet long by 20 inches diameter suspended by small ropes about 15 feet long, which permit of the bags being lowered as they are scorched and burned off around the thimbles. The total filtering area of the bags is 117,150 square feet, or 154.5 square feet per square foot of firegrate area. There are as few openings in the bag room as is possible, for light. This arrangement causes the ascension of the gases thereby creating a draft, which is very desirable. The white color of the pigment makes it possible to see how to get around with but little outside light. Openings in the room would permit the pigment to be blown out of the buildings by cross-currents of air, and considerable loss sustained.

It is surprising to know how far this fume will be carried in the air. Several samples of surface dirt at various places and at varying distances from the Pitcher Lead plant showed that the fume traveled many miles. The loss in this manner is almost beyond belief, thousands of pounds having escaped before it was suspected.

Adjoining the bag room is the packing room and cooperage, in which the screw conveyer is the most important feature. This conveyer is in a trough 10 inches square along the side of the partition wall. The conveyer blades do not fit the box but leave a space at each corner. There is a square section 2 in. × 2 in. removed from the outer edge of the blades which causes the revolving screw to mix as well as convey. The blades also work the air out of the pigment, making it possible to pack it by machine.

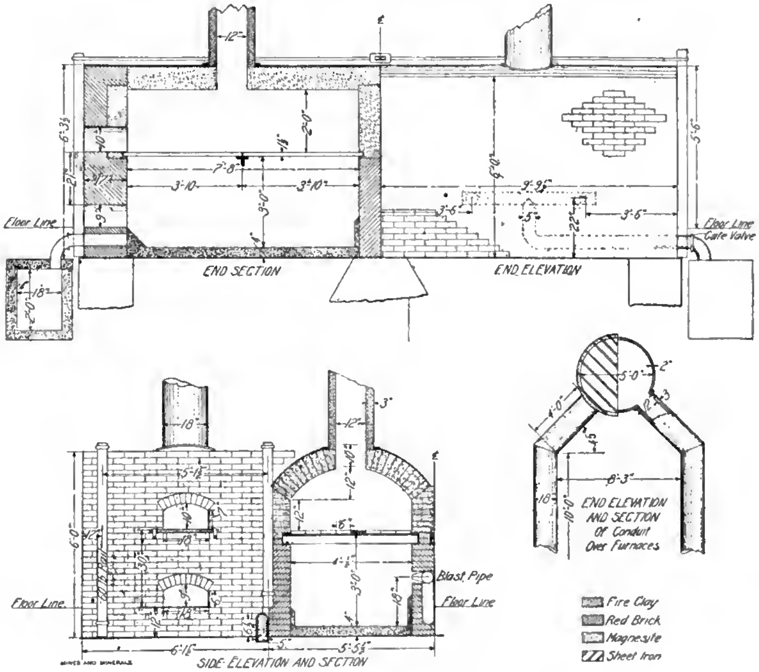


FIG. 2. FURNACE AT PIGMENT PLANT

The room over the packing department may be used as a cooperage, carpenter shop, "tailor shop," and storeroom for supplies. One or two men are kept busy repairing bags.

The switch between these rooms and the warehouse is up grade, to permit the cars to be moved by gravity to the concrete loading platform which is on a level with the car door.

The quantity of blast must be regulated to comport with the composition of the charge. For an ordinary charge using a coal containing 70 per cent. carbon the air necessary would be about 140 cubic feet per pound coal, or a total of $12{,}000 \times 140 = 1{,}680{,}000$ cubic foot per furnace. This for 24 furnaces would be 40,320,000 cubic feet of air for fuel combustion. For a charge carrying 45 per cent. lead, each pound of lead would require $.0168 \times 36{,}000 = 604.8$ cubic feet of air for the formation of the basic sulphate (the sulphur present goes to form the sulphate), a total of 40,320,604 cubic feet of air required per 5 hours, or 300 minutes = 134,402 cubic feet of air per minute.

This blast is delivered at from 1- to 2-ounce pressure by two No. 5 blower fans. The blast is kept underground, as shown in Fig. 1.

The 6-foot exhaust fan is driven by an electric motor with a resistance to regulate the speed between 150 to 250 revolutions per minute, depending principally on the outdoor temperature. As a rule the cooler the air is the faster the fan is driven.

The location for this fan was governed by the results of a practical demonstration which was carried out at a lead smelter in Collinsville, Ill. At first the fan was erected near the furnace, then it was moved about 100 feet toward the bag house, next it was moved to about three-quarters of the distance from the furnace to the bag house and finally it was set up at the bag house, where it gave satisfactory results for the first time.

The steel work for the trail and bag room can be put up for $15,000 according to the estimates of J. H. Blair, engineer, South Western Bridge Co.

In calculating the extra load for roof truss over bag room the bags are estimated to weigh 75 pounds each.

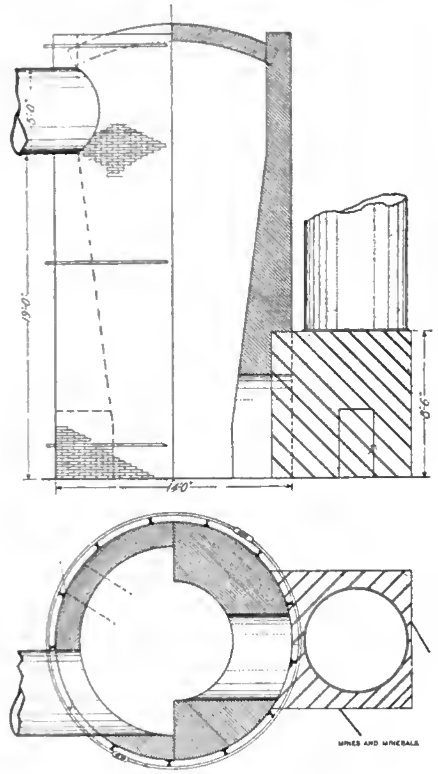


Fig. 3. Plan and Elevation of Tower

A general bill of material for a 50-ton-capacity plant is as follows: 700,000 brick; 35 cars crushed stone; 37 cars sand; 15 cars cement; 3 cars lime; 2 cars lumber; 800 bags, $5,200; 20,000 feet corrugated iron.

METAL MINING FATALITIES IN IDAHO

To those who have through experience become conversant with the difficulty of obtaining satisfactory statistics as to fatalities in metal mines, the recently issued report on the metal-mining industry of Idaho for 1910, by Mr. F. Cushing Moore, state mine inspector, will prove welcome.

The death rate for the past 8 years, the period covered by Mr. Moore's statistics, is remarkably low, but 2.487 per thousand employes. This compares favorably with the fatality rate of 2.45 in such old established coal-mining states as Pennsylvania and is a much better showing than that made by the coal mines of North America as a whole, 3.11 per thousand. It is also far better than the average (3.14 per thousand) for the metal mines of North America exclusive of Mexico as reported by the committee of the American Mining Congress to the Los Angeles convention.

We quote that portion of Mr. Moore's report bearing upon the subject in question: "The mining industry in the state for the past year, has been very fortunate, in that very few fatal accidents have occurred. The most prevalent cause of accident was that of falling rock in the working faces, of which there were seven fatal accidents, or about one for 1,000 miners employed. Explosions of blasting compounds was responsible for two deaths and two men were killed by falling down chutes or raises, another death was caused by contact with a high tension electric wire used in transmitting power to the underground workings. This, however, is a very creditable showing and is due to the persistent efforts on the part of the mine operator to caution the underground worker to protect himself, as the vast majority of accidents are due to the carelessness of the injured.

"The accompanying table shows the number of fatal accidents with the causes for the same, since the first of the year 1903; prior to that time no record of fatal accidents was kept in this office."

The above table emphasizes the fact that accidents are very largely unavoidale at times, for during the year 1908 only 10 accidents occurred, while there were 19 during 1909, and again during 1910 only 11, but as 10 of the 19 accidents in 1909 were due to falling ground, it appears that it is negligence on the part of the miners themselves, rather than upon the mining operators, as the same precautions were taken each year. This point cannot be too strongly emphasized, for no matter how careful a company may insist upon the back being examined, prior to the time the shift goes on, by the time that several holes have been drilled by a machine, other rocks have been shaken loose and only constant watching by the men at the breast can prevent fatal accidents.

COALING STATION IN SOUTHERN PACIFIC

The Campagnie Francaise des Phosphates de l'Oceanie has organized a coaling station at Tahiti, Society Islands, and is prepared to furnish coal to all war or merchant ships calling there. A 3-masted bark is used as a depot and is permanently anchored in 9½ fathoms of water in the harbor, where coal may be loaded from either side to other vessels at the rate of 300 tons a day. The stock will be 2,500 to 3,500 tons, to be increased according to demand, and is from Southern Australia, being "the best screened large coal." The prices will be $9 to $11 a ton, influenced by varying market prices and freight rates.

Fatalities in Idaho

Cause	1903	1904	1905	1906	1907	1908	1909	1910	For 8-Year Period	Percentage of Total Accidents for 8-Year Period
Fall of rock from working faces	4	3	5	4	5	4	10	7	42	33.33
Explosion of blasting compounds	7	5	5	6	2	1	2	2	30	23.81
Falling down chutes, raises, and other openings	3	1	3	4	7	1	4	2	25	19.84
Hoisting accidents—bucket, cage, or skip	3		2			1	1		7	5.56
Electrocution by contact with live wire			2		1	1	1	1	6	4.77
Car accidents			1	1	1	2	1		6	4.77
Tapping old workings	1	1			2				4	3.17
Caving bank in placer mine	1		1						2	1.59
Suffocation from gas or smoke	1								1	.79
Accidents from handling timber				1					1	.79
Falling staging while drilling				1					1	.79
Gasoline tank explosion			1						1	.79
Total	20	10	20	17	18	10	19	12	126	100.00
Average number employes	7,000	6,000	6,000	7,000	7,000	5,500	6,000	6,000	6,312	
Fatalities per 1,000 employes	2.85	1.66	3.33	2.42	2.55	1.75	3.33	2.00	2.487	

THE PORCUPINE GOLD FIELDS

*Written for Mines and Minerals, by Ralph A. Meyer, E. M., B. Sc.**

Until the last two or three years the Porcupine area was difficult of access and little prospecting was done on it. The information concerning it was practically all contained in the reports published by the geologists who accompanied Ontario Land Surveyor Niven in his base line work and the township surveyors in the years 1896 to 1899—1903 to 1905.

Districts of Nipissing and Sudbury, Ontario. Location and Geology

The first real prospecting in the area seems to have been performed in 1906 on the Wilson claim now known as part of the "Timmins" mine.

In 1909, during the month of June, interest was revived in the district by numerous discoveries having been made, following which the Government had an abstract report and brief examination made by Mr. James Bartlett, one of the Bureau's geologists.

From 1909 to present date, much enthusiasm has been displayed, approximately 10,000 claims having been staked.

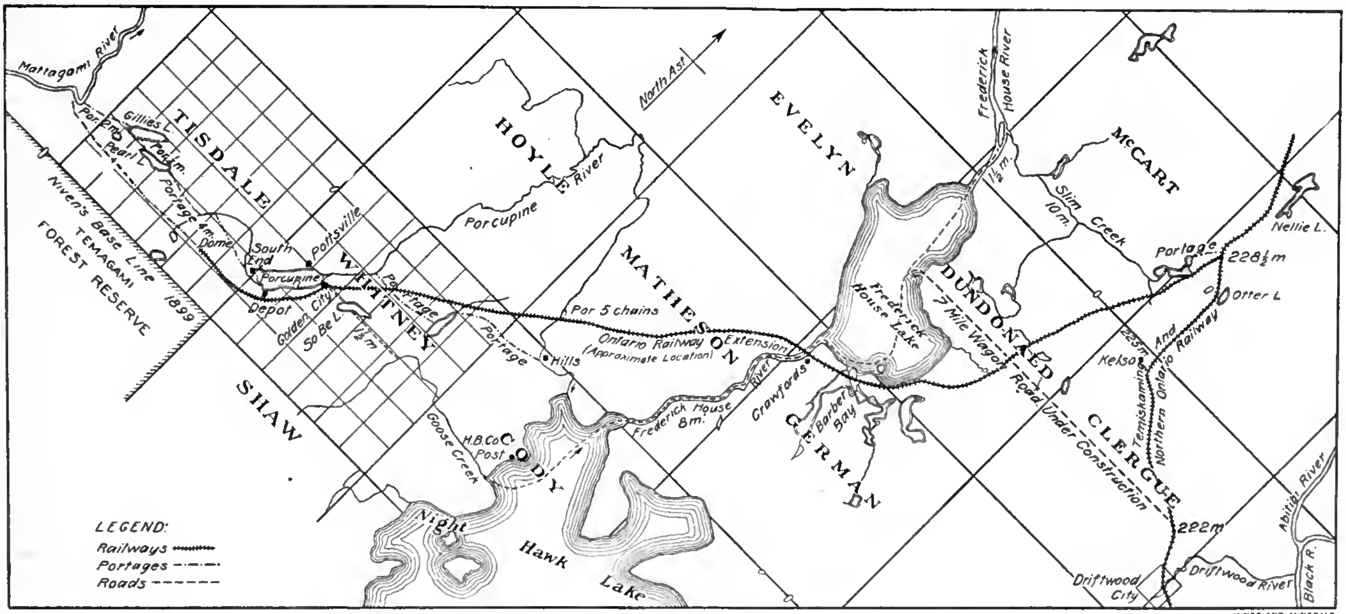
The tonnage will be obtained from the sulphide deposits, and even at the present early life of the camp, all indications point to a large tonnage. As a proof of such indications, both the Dome and the Timmins properties are installing mills of 40 and 30 stamps, respectively, equivalent to a daily capacity of approximately 350 tons.

Several of the smaller properties, as well as those that are in earliest stages of development, have or are installing small units, such as two Nissen stamps.

It is predicted that certain sections in the Porcupine area will contain free-milling ore that will make history in gold mining in the North American continent.

To reach the Porcupine gold camp, go to Kelso, on the Temiskaming and Northern Ontario Railway. From this point the balance of the journey is performed by driving; the total distance being 34 miles. This is known as the winter route. During the summer and spring months, the journey is performed by means of driving and launches.

The first portion of the journey, on leaving Kelso, is covered by a 12-mile drive which is subdivided into 7 miles on a fairly good government wagon road, and 5 miles along the shore of Frederick House Lake, on a gravel bed, to what is known as



Map of Porcupine Region

The greater amount of development work has been performed in the Township of Tisdale on the Timmins, Dome, Foster, Vipond, Crown Chartered, Armstrong-Gibbon properties, etc. There are also several properties located in the adjoining townships of Whitney, Shaw, Deloro (formerly known as the Temagami Reserve) Carmen, Langmuir, El Dorado, Denton, where phenomenally rich surface showings have been discovered and in which claims have been purchased and are at the present time being actively developed and prospected.

At the present time the general public is of the opinion that all the "pickings of the pie" are located in the Township of Tisdale. It does not follow because the Ontario Government has decided that each individual township shall be surveyed into 6 miles square that all mineral occurrences should cease within the near approach of a concession line. The uninitiated are prone to regard these concession lines as stone walls that cut off everything.

The general public should not be too easily captivated by spectacular free gold showings, as these alone do not convey much more than that the country is gold bearing.

Crawford's Stopping Place, located at the junction of Frederick House Lake and the Frederick House River. At this point the journey is covered by means of gasoline launches up the Frederick House River, across Night Hawk Lake in a northwesterly direction to what is known as Hills, a total distance approximating 14 miles. From the latter mentioned point, the balance is covered by driving on the government road to Porcupine townsite, better known as Golden City, a distance of 8 miles.

Both in the summer and in the winter, the journey from Kelso to Porcupine can be made in a day, and in fact in the winter season it takes a considerably shorter time, the entire distance being traversed by sleighs.

The Ontario Government is at the present time extending the railway from Kelso to Porcupine, in fact, 8.5 miles of the steel is already laid and the government expects to have the entire track installed by the end of June of this year.

Since the government has taken these steps, the camp has gained stimulus and prestige.

The railroad was advised by several geologists attached to the Bureau of Mines and by some of the most reputable engineers, such as William Frechevill, of the Anglo-French Exploration Syndicate; Louis Webb, of the South African Consolidated;

*Camborne School of Mines, Cornwall, England, and Royal School of Mines, Freiberg, Saxony, Germany.

W. J. Loring, of Bewick, Moering & Co.; Doctor Simon, representing French capital; and others.

Under present conditions one is able to leave New York and arrive at Kelso, within 34 miles of the camp, in a Pullman car, the journey being completed within 1½ days.

Such easy access to a gold camp from the large centers such as New York, Chicago, Pittsburg, Boston, and Toronto, has never been known before. This is very important to those who invest appreciable sums of money in gold mining as they are in the position to visit and keep in touch with their investments more satisfactorily than if their headquarters were located a thousand miles or more from the seat of operations.

There are three recognized townsites, located on Porcupine Lake, which is approximately 2 miles long, ½ mile in width, the elevation being 913 feet above sea level.

The townsite, known as the Provincial Townsite or Golden City, is situated at the extreme north of Porcupine Lake; that known as Porcupine Townsite, where the greatest amount of activity has been displayed, is located at the extreme northwest of the lake.

Since the location of the railway has been definitely decided, there is no doubt that the provincial townsite and the one at the extreme south end of the lake will be the most active, as there will be a depot at each townsite.

The recorder's office is located at Golden City in addition to several stores, etc.

A post office and hotel, known as the "Shuniah," also two banks are located on Porcupine townsite.

Three other hotels are under construction in addition to a post office and theater at the south end townsite.

There are five banks already in the camp.

The townsites are all on flat land partly composed of "muskes" (a local term designating swampy ground) and are connected with the outside world by telephone, two lines having been installed; one to Kelso and the other to Matheson. The former is operated chiefly for local calls, and the latter is used mainly for the transferring telegraphs.

At the present stage the properties generate their own power, but such will not be the case nine months hence, as a syndicate has leased the Sandy Falls, situated on the Mattagami River, in the Township of Mountjoy, distance 6 miles approximately from the main center of active operations.

The syndicate intends to generate 3,000 horsepower, which will adequately supply the electric energy as well as motive power for compressed air for the entire camp, so far as discovered up to date.

The greater part of the surface is low and wet, and this has hampered prospecting considerably during the past summer, but it does not signify that because a claim is situated in low ground it is worthless, for since the freeze-up, approximately one dozen most valuable discoveries have been made in the so-called swamps.

In other words, what is looked upon as swamp, is nothing more than the surface drainage from the surrounding outcropping ridges. There is no doubt that as soon as mining operations are in full swing the swamps will become drained. The lower flat surface is occupied by a banded clay, together with some sand and gravel. Overlying the clay is a layer of vegetable mold from a few inches to a foot or more in thickness.

Outcrops of compact rock occur irregularly over the surface covering, somewhat resembling the undulated surface of a calm sea.

Some of the outcroppings are small and difficult to locate; in other sections of the area the rocks rise into ridges, which extend across several claims.

In the majority of cases the rocks do not rise to a greater height above the general level than 50 feet, and it is exceptional that they rise to heights of 100 to 150 feet. The area lying between the mouth of Porcupine River and Porcupine Lake has a maximum elevation of about 970 feet; the southwestern portion of Tisdale is considerably higher, reaching an elevation of 1,000 feet.

The lakes are narrow and the rivers have cut deep.

The greatest depth of Porcupine Lake has been found to be 20 feet.

The oldest series, the Keewatin, is similar to rocks of this age found in various parts of the Province of Ontario from the Quebec boundary on the east to that of Manitoba on the west.

The series is here much more disturbed than it is in the Cobalt area, as is evidenced by its schistose appearance.

In the Porcupine area, some of the Keewatin rocks have escaped metamorphism sufficiently to show their general character. Most of the Keewatin in the Porcupine area, as elsewhere, consists of dark-colored or greenish schistose rocks of basic composition.

Quartz porphyry is however a quite common rock in the Porcupine area.

While it occurs in dikes it is also found in large masses. Associated with the Keewatin, especially in the southern part of the township of Whitney and southeastern portion of Shaw, there is an iron formation known as "jaspeylite," consisting of thin alternate bands of magnetite and silica. In certain instances the Keewatin rocks contain considerable calcite and dolomite.

A belt of Huronian fragmental rocks outcrops at intervals across the northern part of Whitney township and southwest through Tisdale township. The Huronian has been subjected to metamorphism and in instances has been rendered highly schistose.

Several dikes of olivine diabase, cutting the Keewatin, have been found in the area; also serpentine and sericite schists, which are mostly associated with the quartz porphyry and rhyolites. Some of the schists are impregnated with iron pyrites and ferruginous dolomite, which at the surface cause the rocks to weather to a rusty brown. The general strike of the schist is northeast and southwest. There is pronounced evidence of glacial action over the entire area.

Numerous outcrops of quartz are found both in the Keewatin and Huronian rocks. The age of the rocks does not seem to have any bearing on the vein either as to form or to gold contents. There is little doubt that the quartz deposits of the Porcupine area are connected with the great granite intrusions which took place in post-lower Huronian times. The quartz has been deposited from the impure waters, highly heated and under great pressure, which worked through the rocks after the granite intrusion.

The quartz veins vary in width from a few inches to 30 and 40 feet, the average width being from 3 to 4 feet. In many instances the ore bodies cross the strike of the schist, and in such cases they vary in width from point to point. In individual cases masses of quartz from 75 feet to 100 feet in width are rich in free gold.

A feature worthy of note is the persistency of the veins, which in the majority of cases can be traced for hundreds of feet, and in individual cases for thousands of feet.

The area over which these gold-bearing veins occur covers approximately 50 square miles, and each day reliable information is being received relative to gold discoveries made north, west, and southeast of the present center of operations.

Visible gold is seen at numerous points in the large masses of quartz and is associated chiefly with iron pyrite, arsenical pyrite, galena, zinc blende and small quantities of copper pyrite.

The schist near the veins is impregnated with iron pyrite and in many cases assays high in gold.

In South Whitney and southeast Shaw townships, deposits of iron pyrite occur, which carry gold. Samples from one of these sulphide deposits assayed $20.20, $34.80, and $80 in gold. It must be understood however that these figures were obtained from surface samples which had become enriched by oxidation and concentration to an average depth of 4½ feet.

Two pits were sunk to a depth of 9 feet and after systematic sampling of the vein which was 12 feet and 22 feet wide in the pits, respectively, an average assay of $7.50 was obtained.

It would appear that originally the iron pyrite and other associated sulphides were the "mother parent" and are the true gold carriers. Such can be assumed from the following reasoning: The chemical composition of iron pyrite is FeS_2. Where the FeS_2 comes in contact with certain mineral solutions, or the ordinary physical elements, the sulphides become changed to sulphites and thiosulphites, and a small proportion into sulphates. The sulphites are soluble and in turn are washed away. The iron becomes oxidized and forms the reddish-brown oxide or gossan. Any insoluble matter or elements that are unattacked by the above-mentioned agencies are left behind in the free state, which is in this particular case gold.

Note.—A small amount of gold is dissolved due to the presence of a certain percentage of iron sulphate.

There is little doubt but that the quartz veins in the Porcupine area are true fissure veins, and that the ore deposits in the said area are actually one of the many links of the great "Appalachian chain."

In conclusion, it may be said that there has been discovered a gold camp in Ontario Province that bears all the necessary indications of becoming one in the true sense of the word, and also one that is within easy access of the commercial centers of the North American continent.

Note.—The various quantitative data in addition to certain geological facts mentioned in this article have been obtained from the government reports of 1909.

METALLIC STRONTIUM

This metal was obtained by fusing pure strontium chloride in a hemispherical iron cathode vessel 25 centimeters in diameter, with walls .6 centimeter thick, with a carbon anode 8 centimeters by 8 centimeters. This allowed a low anode current density and avoided overheating. With a current of 125 amperes and 40 volts for 7 hours, 76 grams of metal were removed in small lumps up to 3 grams in weight. The metal is removed from the mixture by crushing on an iron plate and then sifting away the chloride. Analysis showed it to contain about 98 per cent. of metallic strontium, which is a light metal, with a silvery lustre when cut, gradually becoming yellow and finally white and non-lustrous. It is softer than calcium and can be cut with a knife. It alloys with iron. Hydrogen and nitrogen unite with the heated metal. The specific gravity is 2.55 and the specific heat .0742, corresponding to an atomic heat of 6.5. The iron alloy is rather hard, and decomposes water. This alloy contained 23 per cent. of strontium. *Journal of American Chemical Society.*

EXAMINATION OF STEELS BY CORROSION

A method is described for the examination of manufactured steel (forgings, etc.) in which the polished metal surface is treated either with a 4-per-cent. solution of picric acid in absolute alcohol, or, as a less active reagent, with an iodine solution of 10 parts of iodine and 20 of potassium iodide, in 100 of water. By the action of the solvent, the lines of flow induced in the metal by the mechanical treatment become visible to the naked eye; and thus it is possible to discover the method used for the manufacture of the piece under examination. The effects due to mechanical treatment are generally easily distinguished from initial defects in the metal and from changes induced by heating; in doubtful cases the microscope can be used. M. F. Cloup. (Rev. de Metall., vii, 605.)

ELECTRIC FURNACE FOR ZINC SMELTING

*By Francis A. J. FitzGerald**

A Description of the Furnace. Experiments Showing Temperatures and Efficiency

There is no branch of metallurgy which is apparently more suited to electric furnace treatment than that of zinc smelting. The regular method of zinc smelting is extraordinary in its crudity, inefficiency, and expense, hence the relatively high cost of heat generated electrically is not by any means so serious a consideration as in certain other metallurgical processes. Moreover, the electric furnace possesses certain characteristics which make it specially applicable to the conditions of zinc smelting. In the following paper it is proposed to describe briefly a new form of electric furnace originally designed for zinc smelting, although it has useful applications in other kinds of work.

It is not intended to discuss here the metallurgy of zinc smelting, but to appreciate properly the electric furnace which

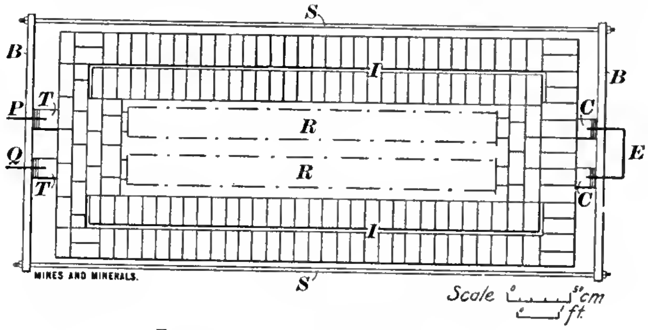


Fig. 1. Plan of Electric Furnace

will be described it will be necessary first briefly to consider the particular method of zinc production for which the furnace was designed. It has long been known that when zinc sulphide and metallic iron are strongly heated the following reaction takes place:

$$ZnS + Fe = Zn + FeS$$

but the reaction does not seem to be complete unless there is a relatively large excess of iron, or unless the temperature of the reaction is very high. Imbert, however, discovered† that by using suitable dissolvents this objection to the process is overcome. Imbert, for example, found that ferric oxide and iron sulphide mixed together in the proportion of one part and three parts, respectively, formed a very fluid bath at a temperature between 1,000° C. and 1,100° C., and that this bath would dissolve six parts of blende. Now when the blende is dissolved in a bath in this way the reaction with iron mentioned above takes place with the greatest ease, is complete, works at a comparatively low temperature, and as a residue produces two distinct substances—a slag consisting of the gangue from the ore, and a ferrous matte which may be used for the regeneration of iron, etc.

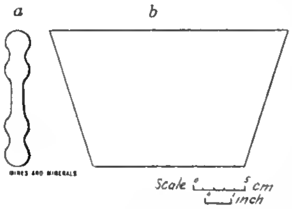


Fig. 2

A great many experiments were made with this process and the results were highly satisfactory, except that it was very difficult to construct a suitable furnace for the purpose. Obviously, working the process in the ordinary zinc-retort furnace would not be satisfactory, for the process should be carried out with a much larger unit than a zinc retort. When it comes to applying fuel heat to such a process numerous difficulties arise which are sufficiently plain without mentioning

* Consulting Chemical Engineer, Niagara Falls, N. Y. Presented before the Congress of Technology at the Fiftieth Anniversary of the Granting of the Charter of the Massachusetts Institute of Technology.
† U. S. Patent 875,579, December 31, 1907.

them in detail This naturally led to the idea of using an electric furnace, and many experiments with various kinds were made. Finally, John Thomson and the author designed a furnace which was used on a large scale in the working of the Imbert process. One of these furnaces of 150-kilowatt capacity was constructed and worked under the author's supervision in Hohenlohehutte, Upper Silesia, with highly satisfactory results.

In order to design a satisfactory furnace it was necessary to keep certain points in view: The furnace must be gas tight; the temperature must admit of careful regulation; the construction must be rugged so as to stand severe usage; the heat losses must be reduced to a minimum, since electrically generated heat is always expensive.

In Fig. 1, is shown plan of the furnace with the cover removed. The walls of the furnace are double, with air spaces *I* which are designed to prevent the loss of heat by conduction through the walls. The furnace is provided with carbons *T*, *T*, *C*, and *C*. The two former serving as terminals which are connected to the source of current by means of cable indicated by *P* and *Q*, while the two latter are simply connector terminals which form the other terminals of the two sections of the resistor *R*, *R*, and are connected by *E*. Bearing on the terminals *T*, *T* and the connector terminals *C*, *C*, are channels *B*, *B*, which are connected with each other by the tension rods *S*, *S*. The channels are, of course, insulated from the terminals. The furnace is lined with a suitable refractory and is provided with a tap hole. The resistor of the furnace is built

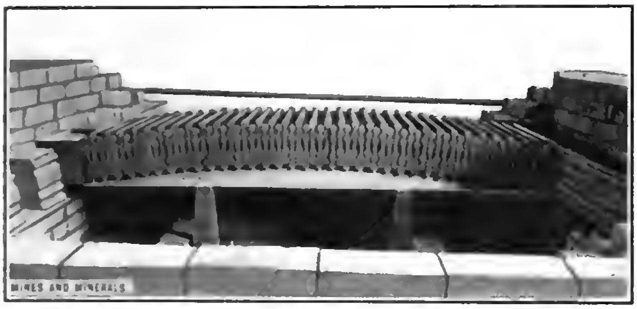


FIG. 3. END VIEW OF RESISTOR

up of a series of corrugated plates, as shown in Fig. 2; an end view of the plate is shown at *a*, while at *b* is shown the shape in which the plates are cut as viewed from the front. In Fig. 3 is shown a view of the plates set up so as to form a resistor. Considering one of the plates it is to be noted that the thickness is not the same from top to bottom, but increases from the bottom up, so that when put in place they form an arch of very long radius, as shown in Fig. 3. Because of the interlocking of the plates this arch form is not necessary, but seems to be desirable in the preliminary assemblage and is also utilized to produce a somewhat greater current density along the lower surface of the resistor. The cover of the furnace, which is not shown in the illustration, carries feeding tubes by means of which the ore mixture may be fed into the bath below the resistor.

The peculiar construction of the resistor plates has two purposes: to give a sufficiently high resistance to the resistor, and at the same time to form an interlocking device so that even if no arch form is given to the resistor yet it will not fall down. A furnace was built with plates having these dimensions: Length at top, 405 millimeters (16 inches); length at bottom, 255 millimeters (10 inches); width, 165 millimeters (6.5 inches). The two sections of the resistor contained 71 plates each. This, when cold, had a resistance of .200 ohm, and when running at the full capacity of 150 kilowatts, and with a temperature in the furnace of 1,400° C., the resistance was .0375 ohm. This resistance is due almost altogether to the contact resistance between the plates, for by calculating the resistance of the carbon itself we find that it would not amount to more than .00064 ohm.

In order to regulate the rate of generation of energy in the resistor there must be some means of varying the voltage at the terminals of the furnace. At Hohonlohehutte, as well as in the FitzGerald and Bennie Laboratories, where these furnaces have been worked, this is done by means of a transformer with several taps brought out from the primary coils which allow the voltage on the secondary circuit to be varied from 50 to 100 volts in 2.5-volt steps, and from 100 to 200 volts in 5-volt steps.

It will be seen that the weakest part in this furnace is the carbon resistor, due to the fact that if working in an oxidizing atmosphere the resistor will be destroyed. In the particular work for which it was designed, however, there would be no danger of this because the furnace is filled with vapor of metallic zinc. During the process of heating the furnace, or at any time when zinc vapors were not generated, there would be danger of burning through air leaking in; but this is easily overcome by keeping a reducing atmosphere in the furnace slightly above external pressure. It has been found by actual experiment that a furnace of this type running continuously for 2 months showed no appreciable wear of the resistor.

The regulation of temperature in this furnace is most satisfactory. In the Hohonlohehutte experiments thermocouples of pyrometers were placed in several parts of the furnace to study the temperature conditions carefully. It was found that the most accurate regulation of the temperature in the furnace was possible, the workman in charge adjusting the rate of generation of energy in the resistor so as to keep the needle of the pyrometer stationary.

The furnace is a highly efficient one. In one of the earlier models, where the heat insulation was far from being satisfactory, careful determinations of all heat losses were made. When working at temperatures between 1,250° C. and 1,260° C. the total heat losses were 33 kilowatts, and when working at temperatures between 1,400° C. and 1,450° C. the heat losses were 42 kilowatts. Consequently, when the furnace is working at full capacity—150 kilowatts—the thermal efficiency at 1,250° C. is 78 per cent., and at 1,425° C. is 72 per cent. No exact determinations of the efficiency of later models have been made, but it is known to be much higher than those given above.

The metallurgical end of the problem has not been completely worked out, but the satisfactory working of the furnace has been clearly demonstrated, and furnaces built on similar principles have been used experimentally with great success in the melting of aluminum, copper, brass, etc. This is thought to be of some interest, as a development in the use of electric furnaces using the heat generated by the passage of an electric current through a resistor. There is a tendency in electric furnace work to employ the arc, which is often a mistake, because of the difficulty in regulating the temperature. Finally, the furnace described above from its construction lends itself readily to adaptations which permit of using the combined heat effects of fuel and electricity, and it is thought that a great future is in store for furnaces of that type.

SODIUM CARBONATE AS AN OXIDIZER

When finely divided, silicon, tungsten, molybdenum, titanium, or such alloys as ferrotitanium, ferrosilicon, phosphor-iron, etc., are strongly heated with sodium carbonate, the metal becomes oxidized, and carbon monoxide is evolved. This is found to offer a convenient method of attacking many metals and alloys which are dissolved only with difficulty by acids. The addition of magnesium oxide to the sodium carbonate renders it more effective, especially for chromium and its alloy, and prevents fusion of the mass. The carbon monoxide is allowed to burn as evolved, in order to avoid the possibility of reverse action. E. Deiss. (Cehm. Zeit., xxxiv, 781.)

COBALT ORE BUYING BY SMELTERS

Written for Mines and Minerals

The Montreal Reduction and Smelting Co., at North Bay, Ontario, propose to eliminate the arsenic to 12 per cent., and smelt the roasted ore in a shaft furnace to silver bullion, speiss, and slag. The speiss is to be crushed, and roasted, and leached with sodium thiosulphate to remove the arsenic. Cobalt oxide, which at one time was almost entirely a product of Europe, is now produced in such quantities as a by-product in smelting Cobalt silver ores, that the restricted market is demoralized and the price of the refined product has fallen from $2.50 per pound in 1908 to 80 cents per pound in 1910. Smelters are not inclined to pay for the cobalt and nickel, and penalize the shipper for arsenic in excess of 5 per cent., and reserve the right to return ore at shipper's expense where nickel is in excess of cobalt. The charges for smelting are graded according to the value of the silver in the ore. The ores of the Cobalt district are smelted in Canada by the Canadian Copper Co., Copper Cliff, Ontario; Deloro Mining and Reduction Co., Deloro, Ontario; Coniagas Reduction Co., at Thorold, near St. Catherines, Ontario; Consolidated Smelting and Refining Co., Trail, British Columbia. Some high-grade ore is shipped to Beer, Sondheimer & Co., Hamburg, Germany; Quirk, Barton & Co., London, England; and Pennsylvania Smelting Co., Carnegie, Pa. Buyers of low-grade ores are, United States Metals Refining Co., Chrome, N. J.; and Balbach Smelting and Refining Co., Newark, N. J.

Bases on which the Different Smelting Companies Fix Prices for Ores of Various Content

The schedules on which payment is made for smelting vary with the kind of ore and the basis of payment adopted by each smelter. Taking as an example the Balbach Smelting and Refining Co.: They pay for the ore 30 days after agreement of assays, and at the silver quotation in New York City, on the date of payment. Their schedule is as follows: Provided ore contains over 1,000 ounces and not over 1,500 ounces of silver, per ton, the smelting charge is $4 per ton of ore, and 93½ per cent. of the assay value in silver is paid for, with 45 cents penalty for each per cent. of arsenic in excess of 6 per cent., and 6 cents for each per cent. of insoluble in excess of iron.

Should ores contain from 1,500 ounces to 2,000 ounces of silver per ton, the smelting charge is $20 per ton, with the same deductions for arsenic and insoluble in excess of iron for fluxing the insoluble matter.

Ores which contain more than 2,000 ounces per ton have 93½ per cent. of the silver paid for and the smelting charge is $19 per ton. The same deductions for arsenic and insoluble in excess of iron are made as in the lower-grade ores. Unless the tariff has changed, the freight to Newark, N. J., is about $9.20 per ton on a 3,000-ounce silver ore, containing 40 per cent. arsenic, 8 per cent. cobalt, and 6 per cent. nickel. The returns to the shipper for such ore would be calculated as follows:

Credits		
3,000 ounces silver at 50 cents per ounce....		$1,500.00
Debits:		
Smelter charges..........................	$19.00	
Smelter deductions, 6½ per cent............	97.50	
Arsenic deductions, 34 per cent.×.45........	15.30	
Freight..................................	9.20	
Assaying and sampling.....................	1.50	142.50
Net return..............................		$1,357.50

The Coniagas Reduction Co. purchases Cobalt silver ores containing but 20 ounces of silver. It would not, however, pay to ship such ore to them, as their charges are much in excess of the value of the ore. When ores contain less than 100 ounces of silver per ton the treatment charge is $10 for 2,000 pounds, unless the ore contains 12 per cent. or over of nickel and cobalt. Their schedule is as follows: Will pay for

	Ounces
75 per cent. of silver contents assaying over...............	20
84 per cent. of silver contents assaying over...............	200
86 per cent. of silver contents assaying over...............	300
89 per cent. of silver contents assaying over...............	500
91 per cent. of silver contents assaying over...............	750
93 per cent. of silver contents assaying over...............	1,000
93½ per cent. of silver contents assaying over..............	1,500
94½ per cent. of silver contents assaying over..............	2,000
95 per cent. of silver contents assaying over...............	3,000

Ore must be delivered in car-load lots Thorold Smelter via Grand Trunk Railway, and is subject to the shipper's risk until sampling is undertaken.

This company pays for cobalt in the ore as follows: When cobalt assays 6 to 8 per cent., 8 cents per pound; when cobalt assays 8 to 10 per cent., 10 cents per pound; when cobalt assays 10+ per cent. 12 cents per pound. When the ore assays less than 6 per cent. nothing is allowed for cobalt.

Beer, Sondheimer & Co. buy high-grade silver ores delivered in New York City, on the following basis dry weight. There is no refining charge. The smelting charge is $30 per ton; a deduction of 6 per cent. being made from the assay value of silver in the ore. Owing to draying, sampling, insurance, etc., this charge is about the same as the United States and Canadian smelters.

The Canadian Copper Co. makes all purchases of Cobalt ore through the Orford Copper Co., of New York City. Their schedule is as follows: The purchaser to make payment for

	Ounces and Over
75 per cent. of silver when same assays...................	100
84 per cent. of silver when same assays...................	200
86 per cent. of silver when same assays...................	300
87 per cent. of silver when same assays...................	400
89 per cent. of silver when same assays...................	500
90 per cent. of silver when same assays...................	600
92 per cent. of silver when same assays...................	800
93 per cent. of silver when same assays...................	1,000
93¼ per cent. of silver when same assays..................	1,300
93½ per cent. of silver when same assays..................	1,600
94½ per cent. of silver when same assays..................	2,000
94¾ per cent. of silver when same assays..................	3,000

Purchaser to make payment of $10 per ton when same contains 6 per cent. cobalt and over; $20 per ton when same contains 8 per cent. cobalt and over; $30 per ton when same contains 12 per cent. cobalt and over.

No payment will be made for cobalt in ores containing less than 6 per cent. cobalt, nor in which the nickel contents are higher than the cobalt contents. Further, the purchaser reserves the right to return at the shipper's expense ore whose nickel content is higher than cobalt content. Ore is to be delivered by the seller f. o. b. cars Copper Cliff, Ontario.

This company, and in fact all smelting companies, assumes no responsibility for ore until it has been taken into its sampler. This company samples free, if the seller's representative is present. Assays are to be made by Ledoux & Co., of New York City, at miner's expense.

Payments are made for 70 per cent. of the silver returnable to the seller, as per the scale, to be made at the New York official price for silver on the first settlement date, which shall be 35 days after the date of which sampling of the ore is completed, and the remaining 30 per cent. on the second settlement date at the New York official price of silver on that date, which shall be 90 days after sampling of the ore is completed. The company reserves the right to deliver upon either or both of the settlement dates above specified in lieu of cash, at its option, such silver bullion (commercial bar silver) as is due the seller in settlement upon these dates, such delivery to be made in New York City.

Settlement for cobalt is to be made 35 days after the completion of the sampling.

The company has put the 75 per cent. silver returnable on ore between 100 and 200 ounces of silver per ton as a penalty clause to apply where ores under 200 ounces per ton were shipped by mistake, and the company does not agree to accept

regular shipments of ore carrying less than 200 ounces of silver per ton of 2,000 pounds.

The Deloro Mining and Reduction Co. receives ore in carload lots f. o. b. Marmora Station, Canadian Northern Ontario Railway. Their tariff is subject to change without notice. Ledoux & Co.'s assays are accepted with the usual provisions as to umpire assays in case the assays of Ledoux and the company's assayer do not approximately agree. No payment is made for cobalt when the ores contain more nickel than cobalt, and no payment will be made for cobalt unless the ore contains 6 per cent. and more; then 10 cents per pound is paid.

This company charges $20 per ton for smelting; pays for 98 per cent. of the silver in ore; and makes a charge of 1 cent per ounce of silver in the ore for refining. Payments made at New York quotations, 75 per cent. in 30 days, and 25 per cent. of the net proceeds in 90 days from the time of sampling.

The Pennsylvania Smelting Co., of Pittsburg, Pa., with works at Carnegie, Pa., purchases cobalt ores ranging from 50 ounces per ton up. They pay for 95 per cent. of the silver less 1 cent per ounce refining charge. There is no payment made for cobalt or nickel, and sometimes arsenic is penalized. The treatment charges are $8 per ton; settling is done in about 20 days after the arrival of the ore at Carnegie.

The richest cobalt ore that the United States Smelting Co. purchases contains 400 ounces of silver to the ton. They do not care for richer ore than this at Chrome, N. J.

ORE MINE NOTES

Lead Production.—The production of refined lead in the United States in 1910 amounted to 470,380 tons of 2,000 pounds. This is an increase of 22,268 tons over 1909. In quantity of lead ore Missouri, as usual, came first with 161,659 tons; Idaho second, with 99,924 tons; Utah third, with 57,081 tons; and Colorado next with 35,685 tons. No other states showed production that reached five figures. The refined-lead product cannot be apportioned according to source of ore, owing to the fact that lead refiners treat smelted products whose origin may, as in custom refining, be unknown to them, the identity of the ore, and thus its original source, being preserved only as far as the smelter. Accordingly, the following table, showing source of lead smelted or refined in the United States, is based on smelter figures. It includes "pig lead" reported by all smelters using Mississippi Valley soft lead ores and "lead" produced at all other lead smelters in this country. The greater part of the product reported by smelters operated in conjunction with refineries is in terms of refined lead. A like part of the antimonial lead product is thus eliminated from the "lead" produced and appears only in the figures of production of antimonial lead. No lead ore from the United States was treated elsewhere during the period covered by the table. Source of primary lead smelted or refined in the United States in 1910 is furnished in tons by the United States Geological Survey as follows:

Total from domestic ores	372,227
Foreign ore:	
Africa	3,499
Canada	44
Central America	7
England	28
Mexico	70,159
South America	3,168
Other foreign	
Foreign base bullion:	
Canada	
Mexico	31,648
Total from foreign ore and base bullion	108,553
Grand total, derived from all sources	480,780

Rhodesian Chrome Ore Shipments.—In answer to inquiries, Consul G. A. Chamberlain, of Lourenco Marquez, Portuguese East Africa, furnishes the following information concerning the shipments of Rhodesian chrome ore through the port of Beira to the several countries in 1908 and 1909:

Country	1908	1909
	Tons	Tons
United States	4,225	11,470
Netherlands	3,612	5,102
France	3,290	3,717
Italy		1,000
Belgium	490	905
United Kingdom	306	677
Total	11,923	22,871

It will be seen that over one-half the quantity shipped in 1909 went to the United States—all to Baltimore.

Santa Rosa Exploration Company.—Volney D. Williamson, a mine owner of Spokane, Wash., who has just returned from the West Indies, announced on April 12 that the Exploration Co., of London, England, through which the Rothschilds conduct their mining operations, has taken over the properties of the Santa Rose de Mazapil Mining Co., and the Santa Rosa Development Co., situated in the state of Zacatecas, Mex. A corporation, known as the Santa Rosa Exploration Co., which is capitalized at $750,000, has been organized to acquire and develop the holdings.

Canada's Asbestos Industry.—Canada produces 82 per cent. of the world's supply of asbestos. The companies operating asbestos quarries and factories in the Dominion are capitalized at $24,290,000. In 1880 only 380 tons of asbestos were produced, valued at $24,700; whereas in 1909, the production amounted to 63,300 tons, valued at $2,300,000. In 1909, 3,000 men were employed in the asbestos industry, and received wages amounting to $1,350,000. Fritz Cirkel, M. E., says in his monograph on chrysotile: "It will not be long before the asbestos slate or shingle business, which is just commencing to be felt, will push its way more and more to the front. Indeed, it is not too much to say that the time is not far distant when fully 75 per cent. of all asbestos produced in the world will be used in the manufacture of asbestos slate and shingles. The asbestos slate business is only 4 years old, but during that short space of time the demand for this article has increased to such an extent that factories for this purpose are being established all over the world."

Treatment Rates Restored.—The rate war growing out of the competition between the United States Reduction and Refining Co. and the American Smelting and Refining Co. for Cripple Creek ore was of short duration. The former made a cut on high-grade ore that was promptly met by the latter. The new schedule, however, was in force only 2 weeks, and the old rates are now in force. None of the independent companies was involved; in fact, because the ore producers have been entirely satisfied, the independent companies have been able to renew contracts at remunerative rates, although the recently-issued annual report of the United States Reduction and Refining Co. makes the complaint that in spite of offering treatment at rates that are not remunerative it has been unable to get an adequate supply of ore.

Will Cyanide Tailing Dump.—John Q. MacDonald, vice-president of the United States Reduction and Refining Co., reports that the new tailing mill, at Florence, is ready to begin operations. It will have a daily capacity of 500 tons, and is expected to make a profit out of tailing that averages only $1.05 a ton. The estimated amount of tailing in the dump is 600,000 tons. As the ore was roasted when first treated, regrinding will be all that is necessary preliminary to leaching with cyanide solution. Tube mills will be used.

Preparations are being made for an active season by placer miners in the Libby Creek district in western Montana. The Libby Creek Placer Mining Co. is arranging for the installation of a saw mill to cut lumber for flumes and buildings about

20 miles south of Libby, Mont., where it holds a large area of ground. Operations have been started on the Rice placer claim in the Republic district, where there is abundant water this spring.

Tungsten in Summit County.—Tungsten ore of high grade, it is reported, has been discovered on Farncomb Hill, in the Breckenridge district of Summit County. The discovery was made by C. S. Laughlin on the old Key West property, a part of the large Wapiti holdings of which Judge Charles Cavender, of Leadville, is the chief owner. The three dredging companies operating on the river-bed auriferous deposits of Summit County are making regular shipments of placer gold. There is an excellent opportunity for a modern cyanide plant to handle Summit County ore, and when built there is promise of the same success that has followed the development of cyanide treatment for Cripple Creek ore.

The Reed Mine.—The first gold mine in the United States was known as the Reed mine. It is 5 miles south of Concord, in Cabarrus County, N. C. At this mine a gold nugget weighing 960 ounces was found, and afterwards, in 1838, the Cabarrus nugget, weighing 444 ounces. Since the latter date several other good-sized gold nuggets have been obtained.

Production of Gold and Silver in Georgia in 1909

Counties	Gold		Silver	
	Fine Ounces	Value	Fine Ounces	Value
Carroll	12.91	$ 267		
Cherokee and Dawson	1,189.93	24,598	19	$10
Hall and Paulding	113.20	2,340	8	4
Lumpkin	1,013.17	20,944	75	39
Murray and Raburn	22.83	472	6	3
White	169.36	3,501	25	13
McDuffie, Union, and others	411.09	8,498	69	36
	2,932.49	$60,620	202	$105

Production of Gold and Silver in North Carolina, in 1909

Counties	Gold		Silver	
	Fine Ounces	Value	Fine Ounces	Value
Burke	219.38	$ 4,535	37	$19
Cabarrus	79.14	1,636	10	5
Catawba and Gaston	605.66	12,520	141	73
Cleveland	30.57	632	4	2
Franklin, Granville, and Nash	100.67	2,081	4	2
Jackson, McDowell, and Rutherford	49.58	1,025	110	57
Mecklenburg	18.58	384	1	1
Montgomery and Randolph	142.76	2,951	43	22
Rowan	518.92	10,727	110	57
Stanley, Yadkin and others	180.87	3,739	39	21
	1,946.13	$40,230	499	$259

Tin Mined in Alaska.—The United States Geological Survey reports that in 1909 about 34,000 pounds of stream tin was mined in Alaska and shipped to England.

Hunter Mine, Idaho.—One hundred thousand dollars is being expended to make the Hunter mine, near Mullan, one of the best equipped plants in the Coeur d'Alene district. Among the improvements contemplated are the enlarging of the mine and the installation of an electric hoist and motor cars for hauling ore from the shaft to the bins on the tunnel level. The capacity of the mill also will be increased from 400 to 800 tons daily. The latter work is to be completed early in June. The electric hoist, upon which work is progressing, will have a lifting power equal to 5 tons. One hundred and fifty men are at work, the principal operations being on the 400-foot level.

Jack Waite Mine, Idaho.—At 1,200 feet, in the lower tunnel of the Jack Waite mine, in the Coeur d'Alenes, Idaho, a 12-foot ledge, half of which is clean shipping ore, has been exposed. More than 1,600 sacks of ore is on the dump. Assays from recent workings show these values: Lead, 78.5 per cent.; $70.65; silver, 5.5 ounces, $2.72; gold, .36 ounce, $7.20. Development work is progressing. The full face of the tunnel is in ore. The next development work will be a raise to the surface for stoping out the ore. This raise will be 550 feet, and the owners believe there is ore all the way.

Iron Ore in Idaho.—According to George Huston, of Mullan, Idaho, there is an excellent opportunity for the owners of the Bunker Hill and Sullivan, and the Morning and Gold Hunter mines to go into the manufacture of iron. Three milling plants in the Coeur d'Alene district discharge into the river 1,000 tons daily of siderite in their tailing. Here is a waste of natural resources that should stir the conservationists. The siderite can be depended on for years, he states, at 1,000 tons daily, because it occurs as a gangue with galena in the large mines of the district.

Accident Compensation in the Transvaal.—During the year ending June, 1910, there were 915 deaths due to accidents in the Transvaal mines of South Africa. This is at the rate of 5.72 per 1,000 employes in the mines, but even then the total number is 139 less than in the previous year. The average compensation paid for white miners' deaths is £528; for native deaths, £10; and for Chinese, £5. To compensate the schedule the Chinese are not getting killed as numerously as the others.

California Mineral Production.—In 5 years the annual production of petroleum in California has grown from $9,007,820 to $32,398,187. Gold has advanced from $19,197,043 to $20,237,570. Macadam production in 1909 reached $1,636,625. Copper amounted in value to $8,478,142, as against $2,650,605 in 1905. The summary of totals for all products for 5 years makes up a great exhibit as follows: 1905, $43,069,227; 1906, $46,776,085; 1907, $55,697,949; 1908, $66,363,190; 1909, $82,972,209. The advance in 5 years, from 1905 to 1909 was $39,909,209, or practically $8,000,000 a year improvement. This growth is now at a much greater rate than the average figures would indicate, being an advance of more than $16,000,000 a year, comparing 1908 with 1909.

Tennessee Marbles.—Tennessee produces a variety of marbles. Black marble streaked with white calcite; mosaic marble; red variegated marble; a fawn-colored marble; variegated by green, red, or white clouds over brownish red marble; dove-colored, pink, yellow, and onyx marbles. About 80 per cent. of the product is for interior decoration.

The Providencia Mining and Milling Co., of Guanajuato, Mex., whose principal property is the Tajo de Dolores, have completed their new stamp mill and cyanide plant and are expecting to start very shortly. The manager of this property is Mr. Wm. H. McCord. The mine has for many years been operated by use of electrical power, which is supplied by the Guanajuato Power and Electric Co., and the new mill and cyanide plant will also be entirely driven by electric motors. The mill comprises a battery of 60 stamps.

When all the machinery in The Veta, Colo., Mining and Smelting Co. is installed at the Parral, Chihuahua, Mex., plant, it will be one of the most modern. The mine and mill are run by electric power generated by the company. Over 40 motors are used in connection with the milling and cyanide work, totaling about 1,000-horsepower capacity. The hoisting and pumping is also done by electricity, making in all about 50 motors, totaling between 1,500 and 2,000 horsepower.

Platinum at High Price.—Platinum, which now is more costly than gold, has been advancing in price in the last few weeks. It is quoted in New York City at $43 an ounce for the hard and $41 an ounce for the soft metal. These are the highest prices ever reached and indicate an advance of about $10 an ounce in the last 6 months. The present upward movement was nearly equaled several years ago. In 1905 pure platinum was selling at $18.50 per ounce. In 1906 the prices steadily advanced until pure platinum sold at $38 an ounce, and hard platinum at $40. Then a decline started and continued until

1908, when the price declined to less than $20 an ounce. The present upward movement started soon afterward. The chief source of supply is the Ural Mountains in Russia, but some is also obtained from South America and Australia.

Tonopah-Belmont Mine Fire.—Some time ago a fire occurred in the Belmont mine, at Tonopah, Nev. It is said to have started in a pile of lumber in the workings of the mine at a depth of more than 1,100 feet below ground. It is supposed to have been started by some miner leaving a candle on one of the timbers in the mine when he came out from his shift.

When the day shift reported there was considerable smoke coming through the main shaft and the men went down into the mine through another shaft. The greater part of this shift of men were ordered out of the mine as soon as it was discovered that the workings were filled with the smoke, but a small force, presumably of volunteers, remained below to fight the fire. The shift boss, Frank Burke, with four other miners, remained to build a bulkhead, and Burke's body was found about 24 hours later in a frightful condition at the bottom of the 1,166-foot winze. The superintendent and foreman, who endeavored to push their way through the smoke were overcome, and when finally brought back to the surface they were unconscious.

Genuine heroism was shown in fighting this fire and in the rescuing of the miners below the ground. William Murphy descended twice through the smoke and brought back two cage loads of miners. The third time he went down he lost his own life, and his disfigured body was brought up later with several other bodies. Seventeen bodies were recovered from the mine, and it is believed that this represents the extent of the loss of life.

The damage done to the mine amounted to practically nothing. When the fire was first discovered no one at the Belmont mine seemed to have a realization of the danger from smoke and gas arising from fire in the confined space.

SOCIETY MEETINGS

The American Society of Mechanical Engineers hold their next meeting at Pittsburg, Pa., May 30 to June 2, 1911. An elaborate intellectual and social program has been arranged.

The Coal Mining Institute of America will hold their Summer Meeting, June 28 and 29, at Indiana, Pa.

The One Hundredth (100) Meeting of the American Institute of Mining Engineers will be held at Wilkes-Barre, Pa., Tuesday, June 6, and continuing through the week. Headquarters will be at Glen Summit a short distance from Wilkes-Barre on the Blue Ridge Mountains overlooking the Wyoming Valley. To the admirers of scenery this is ideal for a meeting. The Institute was organized at Wilkes-Barre, May, 1871, and has now seventeen of the original members living and active in the affairs of the Institute.

The annual convention of the American Institute of Electrical Engineers, will be held in Chicago on June 26 to 30, inclusive. Among the points of interest to be visited are the Ryerson Physical Laboratory of the University of Chicago, where the atomic theory of electricity has been demonstrated by most interesting experiments; the electric furnaces in the steel mills at South Chicago; the enormous electric plant at the Gary, Ind., steel works, driven by gas engines; the central stations of Chicago, famous for their size and modern design; the hydroelectric development of the Chicago Drainage Canal; "Underground Chicago," with its network of electrically operated freight tunnels; the latest large automatic telephone system; several of the largest manually operated telephone exchanges in the world; street railway and other substations of unusual interest; possibly the largest street railway shops in the world, with electric drive throughout, and many other notable electrical applications.

The sessions will be held in the handsome Louis XVI room in the new Hotel Sherman, which will seat 700 people and can be connected with adjoining apartments to seat 1,500 if desired.

While the list of papers to be presented at the convention is not complete, nevertheless there is a formidable list.

Louis A. Ferguson, 120 Adams Street, Chicago, is chairman of the arrangements for the convention. The Chicago members of the Institute are determined to make this the most entertaining and successful in the history of the society.

THE MENNE-OXYGEN MELTING PROCESS

The equipment included in contracts for the use of the Menne process does not cover the tanks of oxygen and coal gas. These are furnished by Linde Air Products Co. When the tanks are owned by the furnace company the cost of oxygen and coal gas is about 2 cents per cubic foot, much less than when the process was first demonstrated in the United States. Experience has shown that two tanks of oxygen and one of coal gas will open a frozen tap hole, cinder notch, or tuyere, in a few minutes. Two men are necessary to operate the apparatus. (Iron Age, lxxxvi, 8.)

NEW INVENTIONS

Complete specifications and drawings for any of the following patents can be obtained from the Commissioner of Patents, Washington, D. C., at the following rates:

Single copies ....5 cents each
Copies by subclasses...............................3 cents each
Copies by classes...................................2 cents each
An entire set of patents.............................1 cent each

PATENTS PERTAINING TO MINING ISSUED APRIL 4 TO APRIL 25, 1911, INCLUSIVE

No. 990,554. Cutter head for mining machines, John C. Hirst, Reynoldsville, Pa.

No. 990,773. Ore feeder for separating machines, Anders Ponten, Berkeley, Cal.

No. 990,633. Rock pulverizing machine, Alfred Godfrey Campbell, Sherbrooke, Quebec, Can.

No. 990,743. Well-drilling machine, Joseph W. Jennings, Quinton, Okla.

No. 990,205. Coal-mining drill, Richard C. Britton, Buxton, Iowa.

No. 989,819. Ore mill, Jonathan P. Smythe, Longbeach, Cal.

No. 989,868. Ore-roasting furnace, Curt Pfaul, Blasewitz, Dresden, Germany.

No. 989,302. Coke oven, Gustav Schwab, Chicago, Ill.

No. 988,995. Mining sulphur, Herman Frasch, New York, N. Y.

No. 989,184. Well-drilling apparatus, Samuel S. O'Connor, Santa Monica, Cal.

No. 989,257. Rope eye for well drills, Joseph J. Herndon, Sharon, Kans.

No. 988,321. Miner's and the like electric safety lamp, Charles Victor Albert Eley, and Thomas Patrick Brady, Birmingham, England.

No. 988,568. Safety brake for mine cages, Charles Hansen, East Rand, Transvaal.

No. 988,622. Mine door, Newton K. Bowman, North Lawrence, Ohio.

No. 988,948. Means for propping mines, Wilhelm Reinhard, Krefeld-on-the-Rhine, Germany.

No. 988,748. Ore-concentrating table, Enos A. Wall, Salt Lake City, Utah.

No. 988,749. Ore crusher, Enos A. Wall, Salt Lake City, Utah.

No. 988,505. Ore-crushing machine, Harry C. Quick, Los Angeles, Cal.

No. 988,396. Talc ore roaster, George A. Stanton, Chico, Cal.

No. 988,437. Apparatus for treating ores, Isaac A. Braddock, Haddonfield, N. J.

No. 988,458. Apparatus for treating ores, Edwin B. Goodwin, Ward, Colo.

No. 988,737. Process of concentrating ores, Walter Murray Sanders, Marion, Ky.

Mines and Minerals

Vol. XXXI—No. 12. SCRANTON, PA.—JULY, 1911—DENVER, COLO. Price, 25 Cents

WEBB CITY LEAD SMELTERY

Written for Mines and Minerals, by Lucius L. Wittich

Methods Used in Handling and Smelting the Ores and in Refining the Product

The completion of the Webb City Smelting and Mfg. Co.'s lead smeltery at Webb City, Mo., has materially increased the demand for the local output of galena, which, when pure, assays 86.6 per cent. *Pb*, and 13.4 per cent. *S*. The local production, however, averages 80 per cent., which is conspicuously higher than the ore from many other districts of the United States. H. O. Hofman, in the "Metallurgy of Lead," places the percentage of lead in the Granby, Mo., product, which is virtually the same as that produced throughout the entire Missouri-Kansas-Oklahoma district, at 84. This, he says, contains 1.25 ounces of silver to the ton. No effort is made to recover the silver content. According to Hofman, Northern England ranks second in the production of high-grade ores, the lead in that country assaying from 70 to 77 per cent. and carrying 8 ounces of silver to the ton. The lead ores of the Southeast Missouri district average 70 per cent. lead, and Hofman does not credit them with containing any silver. The purity of the Missouri-Kansas-Oklahoma district, or the Joplin district, ores, is due to the absence of contaminating metallic sulphides to any great extent. The most common impurities of the Joplin district lead are iron pyrite and blende.

FIG. 1. WEBB CITY SMELTERY

The total production of galena from the Joplin district is now averaging almost 1,000 tons a week, and of this the new Webb City smeltery is handling 250 tons. The capacity of the plant, which now is 300 tons, is to be increased to 500 tons. George W. Moore, of Webb City, president; R. A. Farnham, of Webb City, vice-president and general manager; and John B. O'Reilly, secretary and treasurer, were careful to arrange for a market for their product before undertaking the erection of a smeltery in competition with the lead trust. Only pig lead is produced and this sells on the East St. Louis pig lead quotations, f. o. b. destination. Cartridge works, shot manufacturers, and other users of pig lead in all parts of the East and in the Middle West appear in the books of the new company as purchasers.

As the crude ore will average 1,600 pounds of lead to the ton, and as the recovery of the plant is said to be 95 per cent., the company secures 1,520 pounds of pig lead from each ton of galena smelted. The price paid for the ore ranges from $53 to $55 a ton; the price received for the pig lead is $64.60, this being for the actual metal derived from the ton of galena. This leaves a margin of about $10 a ton, which the smelters declare is reasonable. It is the desire of the company to make a profit of $1 on every 1,000 pounds of ore handled. In the Joplin district, the ore is really sold on a basis of 1,000 pounds instead of a ton, but to simplify comparisons it will be well to consider the standard of weight a short ton of 2,000 pounds. The purchase of the ore is in the open market, competitive bidding among the various buyers being one of the distinguishing features of the Joplin district.

While the process of reducing crude galena into a merchandisable product, capable of being used in any one of a hundred different ways, is simple in principle, the many little details of the smelting really make the work complicated, and there is much more to the task than the mere heating of the ore until it fuses, the temperature of smelting at the Webb City plant being 1,500° F. in the Scotch hearths, and 2,500° F. in the cupola furnace, where the by-products are smelted. Before the ore finally comes out in the form of marketable pigs several treatments are required, and it also is necessary to utilize the by-products in order that the metal recovery may be as thorough as possible.

The Webb City plant, which was arranged with the view of expediting the production, and which also was constructed along sanitary lines, cost $75,000, and consists of about 12 buildings, occupying a desirable position near railroad tracks on the high prairie north of Webb City, and in the greatest lead-producing area of the district. Fig. 1 shows a view of the plant, looking north, while Fig. 2 shows the plan of the more important buildings, exclusive of the office, which is located to the right of the ore bins.

In the buildings used as the bag room, the trails, and the furnaces, Lamar sandstone, which is less subject to destruction from intense heat, is used. All of the yards are paved with concrete, and through this means the loss of lead ore is minimized. Water used at the smeltery comes from a well 943 feet deep, and is pumped to a tank elevated 30 feet above the ground and having a capacity of 12,000 gallons. In case of fire, complete apparatus is at hand; water mains from the city also extend to the smeltery and these would come into use in case of conflagration.

The mineral building, consisting of 18 ore bins, with a total capacity of 2,750 tons, is 24 feet wide and 72 feet long, and is connected with the jumbo furnace, or Scotch hearth, room by an aerial track along which is propelled the ore bucket having a capacity of 2,000 pounds. Concentrates, ready to be smelted, are dumped directly into the bins after being weighed on the office scales. As the bulk of ore purchased comes from the immediate vicinity, much of it is hauled to the smeltery in wagons; however, a portion of it is received by rail from Joplin, Mo.; Galena, Kan.; Miami, Okla.; Carthage, Mo.; Aurora, Mo.; and other smaller places. Some of the galena mined in the Joplin district occurs in chunks that find a ready market at the best prices. Before this can be smelted, however, it must be crushed,

and for this purpose a small crushing plant has been established at the east end of the mineral building. When filled to its capacity, the mineral building will hold enough ore to keep the smeltery running steadily for almost 3 months.

The aerial track runs the entire length of both the mineral room and the jumbo furnace room, the latter being 28 feet wide and 80 feet long. Passing directly in front of the six jumbo furnaces, as shown in Fig. 3, the ore bucket can be dumped into any one of the six concrete ore receptacles, one of which is in front of each hearth, as shown. At either side of the concrete bin used for the ore are two other bins, one for soft coal, which is used for smelting, and one for pulverized limestone, used as a flux. The ore bucket is equipped with a pair of scales which permits the furnacemen to satisfy themselves on the actual amount of ore they receive. In each shift of 6 hours a furnace crew will smelt approximately 7,000 pounds of the concentrate, which is distributed in seven consignments of 1,000 pounds each. Two men are employed at each hearth or eye, and between shifts the lead is permitted to "freeze" in the hearths.

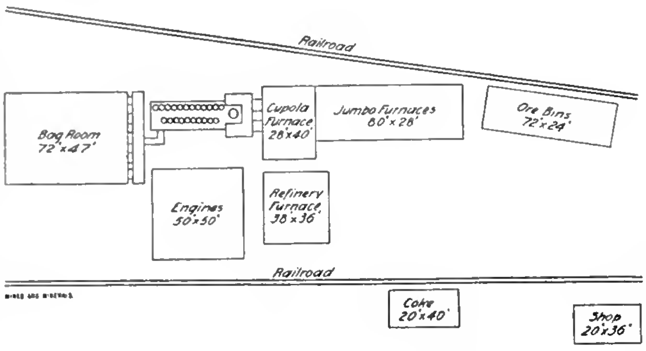


Fig. 2. Plan of Webb City Smeltery

Fig. 3. Jumbo Furnaces

Fig. 4. Refining Department

The furnaces, which are of brick and concrete, with hearthboxes or wells of cast iron, are 5 feet long, 18 inches wide, and the hearthboxes are 7 inches deep. In preparing for the initial charge a wood fire is started in the hearth box and to this is added coal and the blast started. A No. 9 Buffalo blower, driven by a 20-horsepower direct-connected motor, is employed, the air passing into a large pipe or air chamber that runs the length of the furnaces, and from which smaller pipes lead down to the firebox, causing a blast that quickly brings a raging fire. Dampers are so arranged in the smaller pipes that the air blast may be regulated. Ashes and clinkers are removed until the entire hearthbox is a glowing mass of flame. A small quantity of residue from a previous charge is spread over the back part of the fire and the first charge of ore is then given. During one shift approximately 400 pounds of lime flux to 7,000 pounds of ore will be used. As the first charge of ore becomes red hot, the molten lead will begin trickling through the blazing mass and will accumulate in liquid form in the well, the mass slowly lifting the fire until the latter is floating on the bath. When lumps form they are removed and the rich residue is returned to the furnace while the gray slag is thrown to one side to be smelted later in the cupola furnace. About two parts lime are mixed with 30 parts ore and the mixture distributed evenly over the fire and more fuel is added. At intervals the blast is discontinued and then turned on again. A groove from the hearthbox leads across the work stone in front of the box and permits the molten lead to flow into a kettle shown to the right in Fig. 3, from which it is ladled into molds, forming pigs weighing 90 pounds each. The gases from the furnace combustion pass upward through a chimney hood into the fume pipe, as shown, and are drawn by a 100-inch diameter Buffalo exhaust fan operated by a 25-horsepower motor, into the trail, consisting of 11 goosenecks, or 22 stacks, giving a total length of 1,150 feet in which the fumes may cool, before they pass into the bag house. The fume pipe also carries away the gases from the cupola furnace. The goosenecks are shown in Fig. 1 between the bag house on the left and the furnace house on the right.

Fig. 5

The stacks of the goosenecks, which are arranged in two rows, open at the bottom into brick ovens with doors which may be opened at any time and the ashes removed to be resmelted in the cupola furnace. From the end of the gooseneck trail the fumes pass into a last tower connecting with a distributing pipe of greater dimensions than the trail pipe. This runs along the end of the bag room and connects, by five pipes, with as many separate divisions running the length of the bag room. Partitions divide the apartments of the bag room, making it possible to enter any one when the fumes have been shut off and clean out the blue flume which has collected. The operations of the plant need not be impeded by this cleaning-out process.

Each compartment of the bag house consists of a long brick passageway, floored with concrete and ceiled with steel plate, through which the 144 bags of that compartment protrude. These bags are of wool, are 18 inches in diameter, 25 feet high, and are held at the top by steel rods. The

bags are the only inflammable material in the bag house, all the other construction being of steel, concrete, or brick. However, the walls of the structure are of frame. Considering the great distance the fumes must travel before reaching the bag house they are given every opportunity to cool sufficiently to eliminate the probability of conflagration.

Passing into the bag house, the gases, laden with dust, ascend into the hanging bags, where they are filtered, a big portion of the fumes falling into the compartment below. The gases escape through openings in the top of the bag house. Vegetation subjected to contact with these gases is destroyed, and the stifling odor may be detected for half a mile or more from the plant when the wind is in the right direction. As a portion of the fumes adhere to the bags, men with aspirators, dressed as shown in Fig. 5, pass through the building giving each bag a quick shake. The product accumulating in the spaces below is of a fine bluish-gray powder and is known as blue fume. It is possible to utilize this pigment in the manufacture of paint, but it is not put to this use at the Webb City plant. In order that this blue fume may be more easily handled in the cupola furnace it is first converted into porous cakes by burning. The fume is spread in a thin layer over the floor and ignited by burning oil waste. It does not flame, but smoulders for many hours, liberating much heat and some sulphur dioxide. The powder, after being roasted into a porous cake, is free from carbonaceous matter and lead sulphide.

Here the process differs from that employed in the manufacture of white lead at the Picher Lead Co., in Joplin, Mo., where to obtain the white lead powder a second filtration process is employed, the fume from the resmelted blue fume being passed through a second bag house, similar in construction to the first. The product thus derived is a white lead pigment used extensively in paints. As the Webb City Co. does not seek to produce a paint pigment, the blue fume from the cupola mingles with that from the Scotch hearths and passes through the same series of goosenecks.

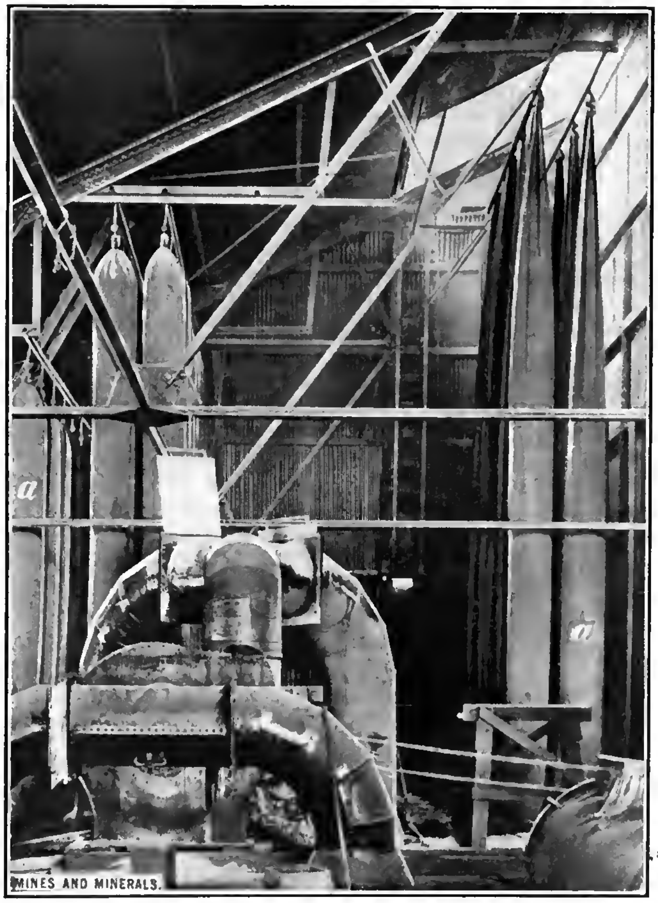


FIG. 6. INTERIOR OF A BAG HOUSE

The cupola is equipped with a Green pressure blower, operated by a 25-horsepower motor. The cupola, which is 16 feet high, smelts the gray slag, which is the original product from the Scotch hearths, and the blue fume, collected both from the bag house and from the brick ovens at the base of the goosenecks. The company estimates the fume to contain 70 per cent. metallic lead; the gray slag to contain 56 per cent. metallic lead; and the ashes or fume from the base of the goosenecks to contain about 56 per cent. metallic lead. From the cupola furnace is secured a black slag, which is the only visible waste of all the material that passes through the smelting process.

In the cupola furnace the heat required to melt the fume and slag is 2,500° F., and the blast fire is kept going continuously. Every conceivable kind of scrap iron, bed springs, rusty rails, tin cans, horseshoes and other junk is used as flux in the cupola. Coke is used as fuel. The company estimates the loss of metallic lead in the black slag at less than seven-tenths of 1 per cent., the other losses being in the waste of ore, smoke, fumes, etc. Of all the metal recovered it is estimated 65 per cent. comes on the first run through the Scotch hearths, and 35 per cent. from the cupola.

Pigs from both the jumbo furnaces and the cupola are remelted in the kettle refinery, Fig. 4, from which the finished marketable product is turned out. This department consists of a huge kettle capable of holding 9,900 pounds of lead. A coal fire is kept burning beneath the kettle and when the metal becomes liquid it is drawn through a tapper valve into the molds shown and in which it cools into 90-pound pigs of lead. Dross skimmed from the kettle goes back to the cupola for remelting, while foam, skimmed from the molds, is returned to the kettle.

As the smelting industry has been looked upon as unhealthful for those employed at the works, the Webb City Co. has constructed its plant with a view of making it as sanitary as possible. The ventilation is excellent and the jumbo furnace room is so arranged that it can be thrown wide open to permit of the free passage of air.

SOLID NITROGLYCERINE

In recent years the E. I. du Pont de Nemours Powder Co. has been manufacturing solidified nitroglycerine to replace liquid nitroglycerine in shooting oil wells. This solidified nitroglycerine is practically the same as blasting gelatin and a considerable quantity of it is used by the oil producers.

The strength of blasting gelatin or solidified nitroglycerine is now generally admitted to be greater than that of nitroglycerine. Manuel Eissler in his book on "Modern High Explosives," page 98, quotes the following from the "Revue d'Artillerie":

"Explosive gelatin is a peculiar description of gun cotton, entirely soluble in nitroglycerine and forming with it a gelatinous or gummy substance more powerful than nitroglycerine, scarcely affected by water, and giving out no trace of nitroglycerine under the strongest pressure."

On page 288 of P. G. Sanford's "Nitro-explosives" it is stated that results obtained by W. Walke, of the United States Artillery, in tests of different high explosives with Quinan's pressure gauge showed the relative strength of nitroglycerine and explosive gelatin as 100 to 106.17.

The du Pont de Nemours Company claim that solidified nitroglycerine is considerably stronger than nitroglycerine. Their blasting gelatin has been used for some time in hard rock tunnels of this country such as the Laramie-Poudre irrigation tunnel in Colorado and on the Roosevelt drainage tunnel at Cripple Creek, Colo.

MINING ON THE GOGEBIC RANGE

*By Percival S. Williams, Ramsay, Mich.**

A Description of the Method of Mining Iron Ore by the Top Slicing System

The method of mining generally practiced on the Gogebic Range is known as top slicing. It differs from the top-slicing system practiced underground on the Mesaba Range in that a pillar of ore about 8 feet in thickness is left over the back lagging on the sublevels during the development stage and during the driving and timbering of the slices in the sublevel pillars, whereas on the Mesaba Range the new slices are timbered up to the floor boards of the sublevels above, leaving no ore to pull back when retreating except that on the sides of the slice, sometimes a slice on each side of the opening slice, and again a slice on one side only.

On finding a new body of ore, drifts and cross-cuts are driven to determine its length and breadth, and 4′ × 7′ raises put up from the drifts at intervals of 25, 50, or 100 feet to prove its height. As raises *a*, Fig. 1, are advanced they are cribbed with 6-inch round timber, opening sets being placed at the sublevel openings *b* so that the work on these sublevels can be started at any time without delay. Poles are laid on top of these opening sets to support the cribbing put in the raise as it is continued above the sublevel. These opening sets consist of two 7-foot posts and a cap on each end of the raise, and are put in the raise 16 to 18 feet apart from floor to floor. This allows for a pillar of ore 7½ to 9½ feet thick from the back of a sublevel to the floor of the next sublevel. Whether or not these sublevels are developed as soon as reached with the raises depends to a large extent on the condition of the ore reserves in the mine. If the ore is needed they are developed at once, but if the management does not need the ore from the section of the mine in which this development work is being done the sublevels would not be developed until a short time before the ore was to be mined. This method of putting up the raises and placing opening sets at the sublevels until the height to which the ore body goes up is reached is by far the most satisfactory, as it permits reaching the top of the ore sooner and makes unnecessary the extensive repair work on the sets of timber that would have been put up in the sublevels if the development work had been done long before slicing on the sublevel began. This feature of additional cost is particularly serious in case the mine is closed temporarily for some unforeseen reason. Frequently, however, ore-reserve conditions are such as to make it advisable to develop the sublevels as rapidly as possible. This is a problem to be worked at each mine, but it is generally conceded that development work on the lower sublevels should be retarded until the time is reached where the ore can be mined to the best advantage.

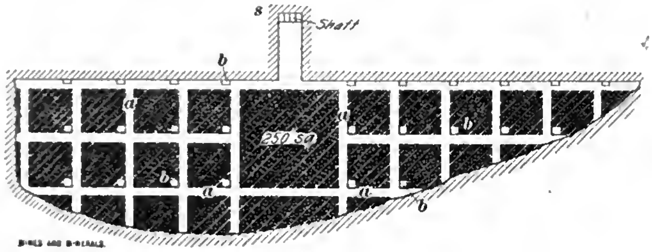


Fig. 1. Plan

It is apparent that the top of the ore body has to be reached and a sublevel driven as near the top as possible before the actual slicing can begin. Sublevels are developed in much the same manner as the main levels, except that the ore is generally blocked out in 50-foot pillars instead of 100-foot pillars as some of the main levels are developed. In one of the mines on the range the main levels are developed with parallel drifts spaced so as to leave 35-foot pillars between them and only occasional cross-cuts are put in. Electric haulage is used on these levels. Raises are put up 25 feet apart. Where the main levels are blocked out in this manner the sublevel pillars are only 25 ft. × 35 ft. The advantage gained by blocking out the ore in this way is in having the raises at more frequent intervals on the sublevels and in producing less curves and crossings on the main level tracks. The cross-cuts on the sublevels are directly over those on the level so as to connect with the raises put up in the ore bodies from the main level cross-cuts. The sublevel drifts are also driven so as to connect with the raises put up from the main level drifts or cross-cuts, and when fully developed the plan of this sublevel would show the ore body cut into blocks about 40 feet square with raises at the corners through which to dump the ore to the main level. A plan of this method is shown in Fig. 2.

* Proceedings Lake Superior Mining Institute, August, 1910.

The sublevel will have a more or less irregular outline on the hanging side, depending upon the manner in which the hanging rock or capping occurs. Small cars of about 1-ton capacity are taken up to the sublevels on which they are to be used. Timber and lagging are usually lowered from a timber raise in the shaft pillar to the sublevels. It is sometimes necessary to hoist the timber and supplies from the main lower level; however, in such cases small "puffers" run by air are installed. From 8- to 16-pound rails are laid in the drifts and cross-cuts on the sublevel floors on which to run the cars, and in some cases only cross-cuts have rails in them, turn sheets being placed at the intersections between the drifts and cross-cuts, and 2″ × 4″ hardwood lumber is used for rails when slicing out the ore.

When the top sublevel is fully developed, the next sublevel below is developed to such an extent that when the slicing is well under way on the top sublevel it may be commenced on the next sublevel below. The ore is sliced out in nearly the same manner at all of the mines, slicing starting at the contact between the rock and the ore on the hanging-wall side and progressing toward the foot-wall side. Two gangs of miners usually work on each pillar, starting on opposite sides and running drifts or cross-cuts parallel and next to their previous openings toward each other until they meet. The openings made by them are timbered with 7-foot timber and the back and side against the ore are lagged. After meeting, the gang starts to retreat in the openings made by them, breaking down the ore over the caps to the floor of the sublevels, where either the miners, or laborers furnished them, shovel it up into cars and tram it to the nearest raise, rarely over 50 or 75 feet away. After reaching the original drift or cross-cut from which the miners started, and having previously covered the floor of the mined-out portion of the sublevel with boards or lagging to enable cleaner extraction of the ore on the sublevel below, they start another slice as before, either leaving about 6 feet of ore

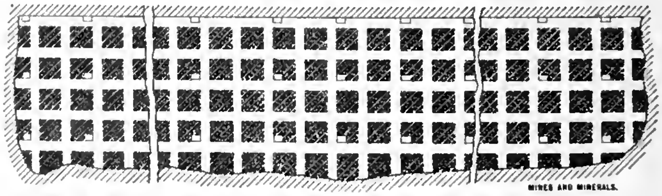


Fig. 2. Plan

between them and the space mined, this ore to be mined when they start to retreat and take the ore over them, or another slice is cut from the side of the ore pillar. Where the latter method is practiced, the ore over the top of the caps in the first slice is usually left to support the material above the ore until the second slice is driven, after which the ore over the space formed by removing the first slice is broken down. The ore over the second slice would be broken down after the third

slice had been driven, etc. When mining in this way, the timber and side lagging put in place with the previous slice is depended upon to hold the top long enough to remove the ore over the slice against the side of which the cave is pressing. In this manner the pillars are gradually pulled back to the safe points of final retreat from the sublevel until the entire sublevel is mined out. In the meantime, however, mining operations are well under way on the lower sublevel, it being the object to mine

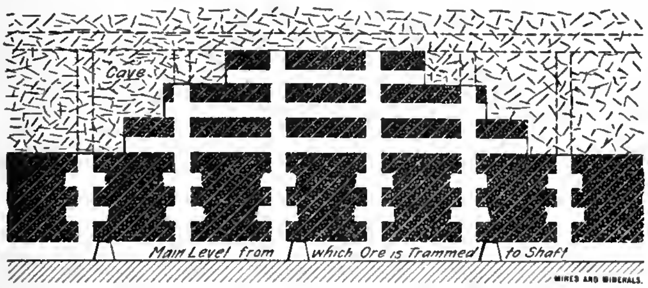


FIG. 3

back the sublevels with the same progress, always having the miners that are working on the next sublevel below working under the cave produced by the men working above them. In this way a longitudinal projection or a cross-section would show the ore in place at any time, forming a succession of steps covered over with gob or waste material, as shown in Figs. 3 and 4.

Any rock encountered in mining out the slices is thrown into the mined-out portion. The timber is usually broken with the blast and also left behind. In this way a gob accumulates, which in the course of time aids materially in holding up the cave while the ore over the slice on the lower sublevel is being mined out. It is advantageous to have the rock capping break down of its own weight on the cave soon after the immediate ore underneath is removed, but at times this capping hangs up for a long period and it is necessary to depend upon the cushioning effect of the accumulation of rock and timber left in the opening during mining operations to prevent a heavy fall of hanging rock from crushing the pillar and timber used to protect the miners. Some ore deposits have extended close enough to the surface to make the surface follow down with the cave, but frequently with smaller ore bodies the surface does not follow readily. This system of mining never exposes a miner to an open stope into which there is danger of his falling, or to being injured by a fall of rock from overhead. Raises occur at frequent intervals into which men might fall from the sublevel. The ore compartment of these raises, however, has six pieces of steel spaced about 10 inches apart, across the opening while the ladder road has a trap door at each sublevel which makes it unusual for a man to be injured by falling down a raise. Accidents will happen, however, so long as there are mines, and the best thing to do is to be on the alert to see that all precautions are taken to prevent them.

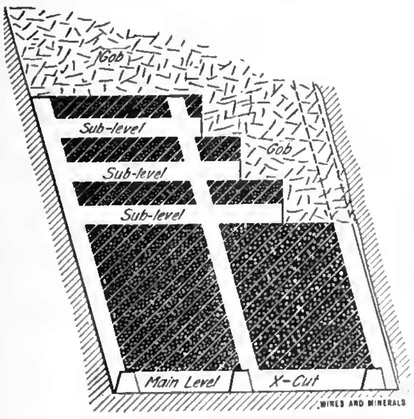


FIG. 4

There is a growing tendency among operators to shorten the distance between raises, in order to reduce the shoveling of ore into cars on the sublevels. The practice of covering the sublevel floors with lumber to help support the cave when mining the sublevel underneath, seems to be growing in favor, even where the gob above is compact enough to enable a good extraction of ore without the use of boards. One thing certain is that by exposing the floor boards, miners on the sublevel below know definitely when to stop work toward a cave above them, and this helps the bosses to keep up the grade of ore by preventing the miners running waste material into it.

While 95 per cent. of the Gogebic Range ore is extracted by means of the top slicing system, certain conditions are met in places which make other mining methods more feasible. Where the ore deposits are narrow, say from 6 to 20 feet wide, developing the ore with sublevels has proved too expensive. Where it is not necessary to make more than one grade of ore, the deposits have been worked by the back stoping method, the underhand stoping method, or a combination of the two. No timber is used in any of these methods. The main level is first drifted on and raises put up about 15 feet apart to the first sublevel which is driven so as to leave 10 feet of ore between the back of the main level and the floor of the sublevel. The raises are widened at the floor of the first sublevel to enable them to accept ore from more territory. Up to this point all three methods are alike. In back stoping, the ore is broken by blasting and drawing enough ore from the chutes to let the miners have room to work. The stope is carried up in this manner within 6 feet of the level above, where if there is waste material resting on the floor of the upper level, the ore is drawn out before proceeding further. In case no waste material is resting on the upper level, the miners go to the upper level and take the 10-foot pillar above the level, and at the same time break the 6-foot pillar under them a little at a time, starting at the point farthest from their means of escape and retreating toward the shaft or limit of the stope. In case waste is being held up by the floor pillar, the waste should have been supported by stulls, otherwise when the pillar finally falls to the chutes below it, the ore will be mixed with waste and some lost in consequence.

FIG. 5

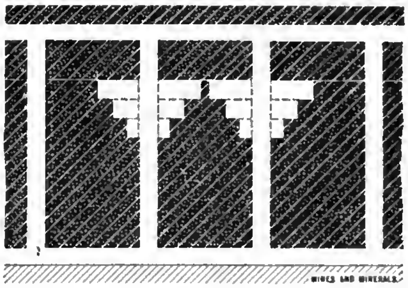


FIG. 6

In the underhand stoping method raises are continued through to the level above, usually 100 feet. Miners start to drift on each side of the raise, leaving about 6 feet of ore between them and the level above. As soon as they are 10 or 15 feet away from the raise other miners start to break the ore that formed the floor for the first gang of miners. These latter miners are in turn followed by others a step lower down, and the stope is gradually worked out, as shown in Fig. 5, the broken ore running into the funnel-shaped raise without much

shoveling in the stope. The hanging wall must be very strong and absolutely self-supporting in order to make this a safe system of mining without the use of timber or stulls. This method is more susceptible to successful ore grading than the back stoping method, but both are much inferior to the top slicing method in this respect. The pillar left under the floor of the upper level is blasted down in the stope from the upper level after the stope is mined out, and serves merely as a protection to the men while working in the stope from anything that might fall from above it, as shown in Fig. 6.

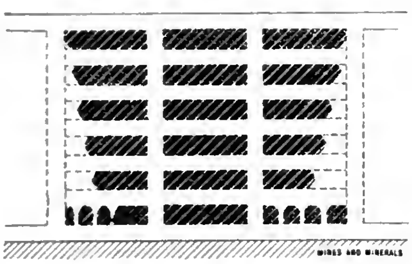


Fig. 7

A combination of these methods has been used after the narrow pillar between the main level has been developed or cut up with sublevels, when it was not known but that the ore might widen out and become softer thereby necessitating the use of the top slicing system. The ore was hard enough to stand without timber in the sublevels, and after its boundary had been established the miners at the sublevels started at the ends farthest from the safe means of retreat and drilled holes which would break half of the pillar above the sublevel and half of the pillar below the sublevel on which they worked, the gang above taking the balance of the pillar above them, and the gang below taking the balance of the pillar under them. The lower sublevels were carried back a little ahead of those above them, in order that the broken ore might have a direct drop to the first level to be trammed out through the chutes, as shown in Fig. 7.

The ore in the Gogebic Range is generally quite soft in structure, making it unnecessary to use drilling machines except occasionally. Holes are bored into the ore with hand augers to a depth of about 5 feet, and 27 per cent. dynamite is generally used to shoot the holes. The foot-wall lies to the north at 60° to 65°, and the hanging rock usually lies in the plane parallel to the foot-wall. The strike is northeast and southwest.

LAKE SUPERIOR IRON ORE NOTES

The Chandler Mine, at Ely, Mich., one of the pioneers of the Vermillion Range, has shipped 9,500,000 tons of iron ore. Two years ago the United States Steel Corporation abandoned the lease, and it was acquired recently by other interests. Explorations have revealed ore comparatively close to the surface; in fact, 30,000 tons have been taken out and stocked during last winter, and it is estimated there are several times that amount in "sight," with prospects of further discoveries.

The Kennedy Mine of the Rogers Brown Co. will be the only Cuyuna shipper this year, for the reason that this range is in its prospecting stage. While there is much to learn about the deposits, enough is known to state that the district will be a factor in Lake Superior ore production. Lean ore high in silica exists in abundance, and the problem consists in finding merchantable ore in quantities. Many of the Lake Superior iron mining concerns, while indirectly interested in the Cuyuna, have adopted the policy of letting others do the pioneer work. However, there is considerable exploration being carried on that is likely to produce several shipping mines in the next two years.

Plumer, Wis.—Prospecting is proceeding in the vicinity of Plumer, Wis., at the western end of the Gogebic Range. It is expected that while the Lake Superior iron ore region as a whole will show decreased shipments in 1911, the Gogebic district will produce as much as last year, namely, 4,315,000 tons.

The Newport Mine of Ferdinand Schlesinger, of Milwaukee, shipped 1,180,000 tons in 1910, which is more than twice its output for any single season with the exception of 1909, when for the first time it attained the million-ton mark. A few years ago this property was considered of so little value that orders had been received to close the mine on the day that a diamond drill, boring from the bottom of the shaft, struck a great deposit of high-grade ore. Since that time as much as 5,000 tons of ore have been taken from that shaft in a single day, this from a depth of 2,000 to 2,400 feet, and this rate can be maintained for some years. It is the Newport that gives its employes each Christmas time an amount equal to their earnings during the month of December.

The Norrie Group.—The steel corporation's Norrie group, on the Gogebic Range, comprises the Norrie, East Norrie, Aurora, Old Pabst, and New Pabst properties, from which a production in excess of 1,250,000 tons is expected this season. The Norrie group has extensive ore bodies—which are still many years from the point of exhaustion—and in addition advantage is being taken of the developments at the Newport. The ore mined in the latter property extends into the steel corporation area and the New Pabst is now being opened on that particular formation. The Norrie group has to its credit a production of 25,000,000 tons.

The Ashland Mine of the Cleveland-Cliffs Iron Co. is counted on this season for an output approximating 225,000 tons. Next to the Newport and the Norrie, it is the largest shipper on the Gogebic Range. The Cleveland-Cliffs Co. have so rejuvenated this mine that it is one of the finest in the Lake Superior region.

The American Mine on the Marquette Range, west of Ishpeming, that was idle from 1893 to 1907 has been acquired by J. R. Thompson and the M. A. Hanna interests. Because the old workings had been robbed, it was necessary to sink deeper in order to reach good productive ground. West of the American, large bodies of ore have been found by diamond drilling and its development is to be started shortly. Part of this land which is controlled by the G. J. Maas interests is owned by the county of Houghton, Mich. Mr. Maas has requested of Houghton County a 50-year instead of a 30-year lease, with the privilege of renewal for a similar period provided that by that time $1,000,000 has been paid in royalties.

Palmer Lake Mine.—A new shipper in the Marquette Range this season will be the Volunteer Ore Co.'s Palmer Lake mine. Thomas F. Cole, of Duluth, formerly in charge of the steel corporation's Lake Superior interests, is interested in the Volunteer company, and while on a recent visit to the property he took occasion to deny any knowledge of the prospective formation of a new steel and iron combine.

The Cleveland-Cliffs Co. is to erect a number of modern mine buildings at its Negaunee property, including a miners' change house, which will be one of the largest and most complete in the Lake Superior district, an office building and a laboratory. With the new structures completed and the new concrete shaft in commission, a considerable portion of the surface will be permitted to cave into the underground workings. The company has started the construction of its dam on the Carp River near Eagle Mills. The dam, it is estimated, will develop 7,500 horsepower.

Lake Superior Southern Railway.—Some little interest has been created in Marquette and Menominee Range circles on account of reiterated reports of an intention to put through the Lake Superior Southern Railway project, the line to extend from Huron Bay, Lake Superior, southerly to Madison, Wis., there connecting with the Illinois Central Railroad tapping en route both the Marquette and Menominee iron districts.

Iron River District.—An interesting development in the Iron River district of the Menominee is the discovery of ore at a depth of 212 feet on the Lindahl property at Beechwood.

SPECIFIC GRAVITY ESTIMATION OF PULP

*By F. B. Hyder**

In order to properly control operations and to be able to make those comparisons between results actual and theoretical, upon which depend success and improvements in cyaniding ores, it is necessary to know the weight of ore treated.

Formulas for Determining the Specific Gravity of Slime Solutions and of Dry Slimes

Where cyanide plants are not arranged so that it is possible to determine by weighing the quantity of ore treated per day or the amount fed into a particular tank, resort is had to the estimation of the contents of the tanks filled, from the specific gravity and volume of the pulp.

For this purpose, the inside measurements of each tank are taken, the volume per foot of depth, and the total volume to one or more convenient levels, marked on the inside of the tanks by painted rings, are calculated and all these data observed and calculated, are recorded on the working drawings of the plant. The volume of the pulp in any tank at any time may then be readily calculated by reference to the drawings of the plant, after measuring from its surface to one of the standard levels.

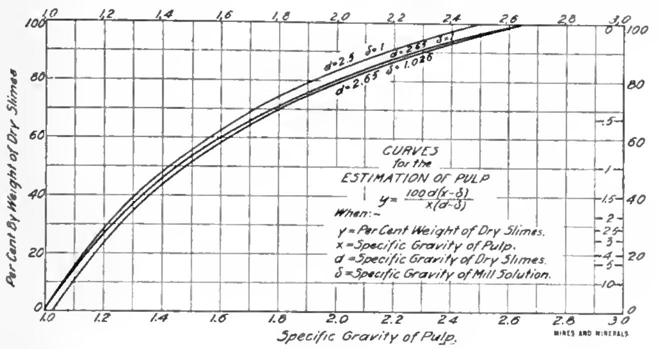


FIG. 1

A liter flask filled to the mark with a sample of the pulp, taken under conditions ensuring, so far as possible, its being representative, is weighed. This sample is best obtained by means of a narrow-necked flask attached to a long pole, so that it may be submerged during the agitation of the contents of the tank. If the pulp is transferred to another vessel before weighing, it should be kept agitated during the process. The weight expressed in kilograms, less the known weight of the flask, is the specific gravity of the pulp.

In the same way, the specific gravity of the mill solution in use may be determined. In practice it has been customary to ignore the fact that the specific gravity of cyanide mill solutions is materially different from that of pure water, and apparently the extent of the error introduced is not generally appreciated. The writer has not been able to find published figures for the range of density of mill solutions. Perhaps some member of the Society can supply them from his experience. Julian and Smart†, however, give as the average density of cyanide solutions (mill solutions) 1.026. From the curves referred to hereafter, it will be seen that the error, assuming this figure to be correct, amounts to from 2 to 3½ per cent. of dry slimes per ton of pulp, in the ordinary range of pulps. The ore tonnages calculated by the usual formulas would therefore be in error from about 4 per cent. for a 1 to 1 pulp, up, increasing rapidly with the dilution of the pulp. The importance of this point is evident.

The average specific gravity of the dry slimes being treated should also be determined. This also is often neglected. It may be done in two ways, which should, taking proper precautions, give identical results.

1. Place a kilogram of the dry slimes representative of the ore being treated in a weighed liter flask and add pure water, mixing thoroughly, until the flask is full to the liter mark. Bring to a boil, adding water if necessary to keep flask filled, then weigh. The weight in kilograms less (1 plus the weight of the flask), is the amount of water added. Call this w. Then:

The specific gravity of dry slimes $=\frac{1}{1-w}$, a simple and more convenient form of the general formula:

$$\text{Specific gravity dry slimes}=\frac{\text{Weight of dry slimes in air}}{\text{Weight of dry slimes in air}+\text{weight of (flask + water)}-\text{weight of (flask + water + slimes)}}$$

2. After determining the specific gravity of a sample of pulp representative of the ore being treated, evaporate a liter of it to dryness. Determine the percentage of dry slimes from their weight. Then:

$$\text{Specific gravity dry slimes}=\frac{\text{Specific gravity of pulp}\times\text{per cent. of dry slimes in pulp}}{\text{Specific gravity of pulp}\times\text{per cent. dry slimes in pulp}-100\text{ (specific gravity of pulp}-1)}$$

W. A. Caldecott* finds that when slimes have been dried, the pulp afterwards formed in the first method must be boiled to expel air or the result obtained will be low.

Having thus briefly indicated the methods of determination of the constants involved, we will proceed to the derivation and use of the formulas.

Then let x = specific gravity of the pulp;
d = specific gravity of the dry slimes;
y = per cent. by weight of dry slimes;
δ = specific gravity of mill solution.

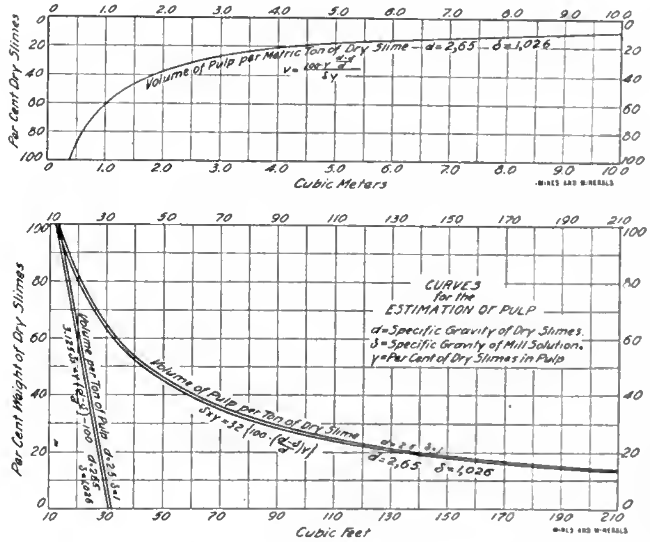


FIG. 2

The formula of the second method of determining the specific gravity of dry slimes may then be expressed:

$$d=\frac{xy}{xy-100(x-1)}$$

Since the volume of unit weight of pulp equals the sum of the corresponding volumes of the dry slimes and of the mill solution, we have:

$$\frac{y}{100\,d}+\frac{100-y}{100\,\delta}=\frac{1}{x}$$

* Colorado Scientific Society, Vol. IX, page 417.
† "Cyaniding Gold and Silver Ores," page 433.

* Reprint Proceedings Chemical and Metallurgical Society of South Africa, pages 373-375.

or
$$y=\frac{100\,d\,(x-\delta)}{x\,(d-\delta)} \qquad (1)$$

and
$$x=\frac{100\,\delta}{100-y\dfrac{d-\delta}{d}} \qquad (2)$$

The percentage by weight of solution in the pulp is then:

$$100-y=\frac{100\,\delta\,(d-x)}{x\,(d-\delta)} \qquad (3)$$

and the ratio of solutions to one of dry slimes is—

$$\frac{\delta\,(d-x)}{d\,(x-\delta)} \qquad (4)$$

The weight per cubic foot of pulp in pounds is the product of the weight of a cubic foot of water by the specific gravity of the pulp:

$$\frac{6250\,\delta}{100-y\dfrac{d-\delta}{d}},\text{ or } 62.5\,x \qquad (5)$$

Or expressed in tons of 2,000 pounds avoirdupois:

$$\frac{3.125\,\delta}{100-y\dfrac{d-\delta}{d}},\text{ or } \frac{x}{32} \qquad (6)$$

The volume of 1 ton of pulp in cubic feet is the reciprocal of (6):

$$\frac{100-y\dfrac{d-\delta}{d}}{3.125\,\delta},\text{ or } \frac{32}{x} \qquad (7)$$

The weight of dry slimes in 1 cubic foot of pulp will be from (1) and (3):

$$\frac{62.5\,d\,(x-\delta)}{d-\delta} \qquad (8)$$

The number of cubic feet of pulp containing 1 ton of dry slime will be from (7):

$$\frac{100-y\dfrac{d-\delta}{d}}{.03125\,\delta\,y}=\frac{32\left(100-y\dfrac{d-\delta}{d}\right)}{\delta\,y} \qquad (9)$$

From (7) and (1) we have for the volume in cubic feet per ton of dry slimes in terms of the specific gravity of the pulp:

$$\frac{32\,(d-\delta)}{d\,(x-\delta)} \qquad (10)$$

For the case where $\delta=1$, the above reduce to some of those given by W. A. Caldecott in the Reprint of the Proceedings of the Chemical and Metallurgical Society of South Africa, page 837, in a table of slime pulp formulas which is of great value.

For $d=2.5$ and $\delta=1$ the above formulas become:

$$y=\frac{1{,}000\,(x-1)}{6\,x} \qquad (1a)$$

$$x=\frac{1{,}000}{1{,}000-6\,y} \qquad (2a)$$

$$\frac{1{,}000\,(2.5-x)}{6\,x} \qquad (3a)$$

$$\frac{2.5-x}{2.5\,(x-1)} \qquad (4a)$$

$$\frac{62{,}500}{1{,}000-6\,y} \qquad (5a)$$

$$\frac{31.25}{1{,}000-6\,y} \qquad (6a)$$

$$\frac{1{,}000-6\,y}{31.25} \qquad (7a)$$

$$104.16\,(x-1) \qquad (8a)$$

$$\frac{3.2\,(1{,}000-6\,y)}{y} \qquad (9a)$$

$$\frac{19.2}{x-1} \qquad (10a)$$

which are formulas in general use but which give only approximate results.

Where metric units are used we have the weight per cubic meter in metric tons is numerically equal to the specific gravity x, and in terms of the per cent. of dry slimes:

$$\frac{100\,\delta}{100-y\dfrac{d-\delta}{d}} \qquad (6b)$$

and this weight may be read off directly from the plot of formulas (1) in Fig. 1 for any desired value of y.

The volume of 1 ton of pulp in cubic meters is the reciprocal of (6b):

$$\frac{100-y\dfrac{d-\delta}{d}}{100\,\delta} \qquad (7b)$$

The volume of pulp containing 1 metric ton of dry slimes is in cubic meters:

$$\frac{100-y\dfrac{d-\delta}{d}}{\delta\,y} \qquad (9b)$$

or, in terms of the specific gravity of the pulp:

$$\frac{1}{x} \qquad (10b)$$

For every-day use it will be found advisable to plot the curves corresponding to these formulas, in which the proper values, determined as indicated above, have been substituted. This may be done on ordinary cross-section paper, as shown in the accompanying diagrams. These curves are preferable to tables calculated from the formulas, such as the excellent ones calculated for $d=2.7$ and $\delta=1$, given by W. A. Caldecott, in the reprint of the Proceedings of the Chemical and Metallurgical Society of South Africa, page 374, both for facility of use and calculation, while the smoothness of the curves is an excellent check on the calculations. In operation and design they will be found useful in a variety of ways.

Fig. 1, showing the curves corresponding to formulas (1) for $d=2.5$, $\delta=1$; $d=2.65$, $\delta=1$; and $d=2.65$, $\delta=1.026$, shows clearly the effect of variation in the value taken for either d or δ. It will be seen that in the range of pulps in practice, the error due to the assumption of the specific gravity of the mill solution to be that of water, may be greater than that introduced by the assumption without determination of the mean specific gravity of the ore. A variation of .15 in d makes a difference equal to that caused by a change of only .026 in δ for a 1 to 1 pulp. For more dilute pulps, the effect of a variation in d rapidly decreases while that due to a variation in δ increases.

It is to be noted that the above formulas apply not only to slime, but to any pulp; i. e., to any mixture of solid particles and a liquid where there is sufficient liquid present to at least fill the voids between the solid particles. At what point this condition no longer obtains is in part dependent on the sizing of the solid particles and is a matter for experiment. Beyond this point the formulas fail. I am not prepared to discuss this further than to say that they hold good for any slime obtained by decantation and therefore will fill the needs of practice.

VANADIUM IN CAST IRON

Vanadium entered into practical metallurgy about 1903, when rich deposits of it were discovered in the Andes of Peru. Vanadium exerts a very powerful influence on steel. One or two-tenths of 1 per cent. raise the elastic limit of mild carbon steel about 50 per cent. or more, without impairing the ductility, and vanadium steels have a very high dynamic strength. Vanadium cleanses the cast iron from oxides and nitrides, eliminates porosity and produces sound castings. In chilled cast iron vanadium produces a deeper, stronger chill, and one less liable to spall or flake. It greatly diminishes the wear on cast iron cylinders. In malleable cast iron the tensile strength is improved about 12 per cent. Geo. L. Norris. (Iron Age, lxxxvi, 5.)

STOPING-DRILL STEELS*

It is essential to the most efficient results in stoping with hammer drills, that everything about the machine itself be kept in the best possible condition. It is just as essential that the drill bits be properly formed, sharpened, and tempered for the work to be done, and that an abundance of sharp steel of the right gauges be constantly available. Poor blacksmithing and the use of wrongly shaped steels or those whose gauge is worn down, will cut down drilling speed, and waste air power and labor to a surprising degree. The following suggestions for the use and care of stoping-drill bits are reprinted from a circular of instructions sent out with the Sullivan stoping drill.

Result of Tests to Determine the Proper Shape and Temper of Bits

The form of drill steel which has been found to be the most practical for use with this machine is the 1-inch heavy ribbed, cruciform solid steel. This type not only by its shape provides greater clearance for the cuttings to fall from the hole being drilled, but also provides greater bearing and wearing surface for the part which enters the bushing than steels of any other shape. The shape of the bit which will give the best results in any particular kind of ground must be determined by experience, for a bit which proves its efficiency under certain conditions may be found inadequate in others, and a very slight change in its shape may bring about surprising results. For the foregoing reason it is impracticable to advance any hard and fast rule to follow in shaping stoping-drill bits, but, as a general rule, the cross bit, sometimes modified to the x-shaped, gives the best results, and such other modifications as the length and thickness of the wings, the angle of the cutting edge, the shape of the end, whether rounding, pointed, or flat, may be determined only by experience in the particular kind of ground in which they are used.

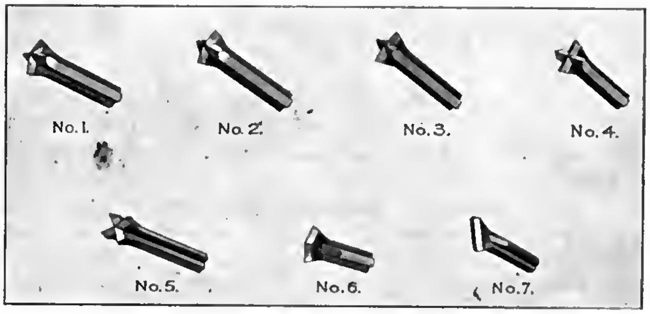
The length of the wings is dependent to a great extent upon the hardness of the ground. If extremely hard, long wings are desirable, as they are capable of withstanding the blow much better than short, flaring ones; while, on the other hand, short wings give the bit more clearance and enable the drill to be rotated easier for a given amount of reduction in the size of the gauge. The accompanying table gives the results obtained by experiments on the seven different styles of bits shown in Figs. 1 to 7, which were operated under the same conditions, so far as consistent, in a moderately hard granite.

Shapes of Hammer Drill Bits and Relative Drilling Speeds

Description of Bit	Duration of Runs Minutes	Diamond Hole Drilled Inches	Average Depth Drilled per Run, 70 lb. Pressure Inches	Average Depth Drilled per Run, 85 lb. Pressure Inches	Remarks
No. 1 25° Diamond point + .	2	1 5/8	6 5/8	8 1/2	Spots easily. Hole rifles slightly. Rotation easy. Gauge does not wear rapidly.
No. 2 25° Diamond point x.	2	1 5/8	6	7 3/4	Spots easily. Drills round hole. Rotation easy. Gauge fairly durable.
No. 3 Convex + ...	2	1 5/8	6 1/8	8 1/4	Spots easily. Hole rifles slightly. Rotation easy. Gauge does not wear rapidly.
No. 4 Flat + ...	2	1 5/8	6	7 1/2	Difficult to spot hole. Hole rifles badly. Rotation hard. Gauge fairly durable.
No. 5 4° Concave + ...	2	1 5/8	5 3/4	8 1/4	Difficult to spot hole. Hole rifles badly. Rotation hard. Gauge fairly durable.
No. 6 25° Pointed Bull – ...	2	1 5/8	5 3/4	7	Spots easily. Drills round hole. Rotation easy. Gauge wears rapidly.
No. 7 Flat Bull –	2	1 5/8	5 13/16	7 1/4	Difficult to spot hole. Hole rifles badly (3 flutes). Rotation hard. Gauge wears rapidly.

Under all conditions the faces or outside edges of the wings should be made flat, with square corners, and never rounding like a figure 8. This is necessary to maintain the gauge of the hole and greatly facilitates the removal of the steels if the hole is drilled at a flat angle.

Before starting, the proper steels needed to complete the hole should be picked out. It should be carefully determined just how much work each drill bit will do before the gauge begins to wear; and the operator should never try to use a bit the second time if the gauge is worn, even if it is sharp on the cutting edge. Furthermore, a drill bit should never be used



Figs. 1 to 7

if the cutting edge is dull, for not only is the drilling speed of the machine reduced, but the cushioning effect which it otherwise gets, due to the penetration into the rock for every blow delivered, is lost, with consequent greater rebound and jar on the moving parts.

A proper gauge for each length of steel should be established and rigidly adhered to, so that the following bit will work freely in the hole. When this system of gauges has been formed, a set of corresponding templates should be made up to insure uniformity in the bits.

Tempering.—The bits should be tempered to suit the particular character of the ground they are used in. The rear or striking end of the drill steel should also be slightly tempered and the face should be kept ground off flat with the edges slightly rounded. A drill steel should not under any circumstances be inserted in a machine if the shank end is rounded or otherwise out of shape, for it will have a tendency to hollow out the striking plug with consequent breakage of this part.

* *Mine and Quarry.*

LAKE SUPERIOR COPPER NOTES

The Mass Consolidated Mining Co., at its annual meeting added two members to its directorate, Francis L. Maguire and Elton W. Walker. It is expected that the rock taken from the mine during the present year will average 17 pounds per ton, which is considerably better than 14.59 pounds, the 1910 average. After another year of development work the mine will be in shape to greatly increase its output. Diamond drill work has started in the Evergreen tract of the Mass company to search for the Algomah vein. The company plans to sell a tract of timberland that is outside of the mineral range. The

Mass Consolidated made a profit from its 1910 operations of $217,798, which compares with a loss of $45,005 in 1909. The company sold 1,321,885 pounds of copper in 1910, for which it received $363,477.

The Tamarack Mining Co.s' annual report shows a loss from mining operations in 1910 of $172,246, or, including interest paid, a net loss of $191,349, compared with $128,546 for 1909, and $235.606 for 1908. The production of copper was 11,063,606 pounds.

The Victoria Copper Mining Co. shows a loss for 1910 of $37,925. Receipts were $152,193 and expenditures $190,118. The company produced 1,164,564 pounds of copper, a slight increase over the previous year. It is evident that shareholders will be called upon for further funds should the management decide to continue operations along the lines now being undertaken.

The operations of the *Atlantic Mining Co.* in 1910 resulted in a loss of $32,162. Receipts totaled $24,671 and expenses $56,834. But little encouragement has attended exploratory work in section 16 of the property, and the future of the company depends principally on what the old portion of the property will reveal.

The Quincy Mining Co. in 1910 made a profit of $725,871, compared with $787,504 in 1909. It produced 34,177,380 pounds of copper last year, against 35,025,225 in 1909. The diamond drill at the Michigan mine, in Ontonagon County, is believed to have passed through the Baltic lode, some copper being taken out in the core. This is of the utmost importance to the Michigan mine, which has about 5,000 acres of mineral lands.

Cost of Milling.—A great deal of interest is caused throughout the copper world by the report that 12 Lake Superior copper-mining companies are joining hands in an endeavor to mill rock at an outside cost of $1 per ton and to produce metal at an outside cost of $1.25 per ton of rock handled. If the proposition can be carried through successfully, rock carrying 10 pounds of copper or more to the ton can be treated at a profit, with copper selling at 12½ cents per pound, and rock carrying 9 pounds of copper to the ton can be milled at a profit with copper selling at 14 cents. A great deal of the rock that is now thrown aside as being of no value could then be treated at a profit.

The Keweenaw County Mines.—At the 1,500-foot level of the Ojibway mine the showing is equal to that of any of the other mines on the famous Kearsarge lode. The rock is full of fine grade stamp copper for the entire width of the lode, which averages 18 feet. Considerable epidote also shows in the rock. In a new opening on the 1,100-foot level the ore is said to be even better than on the 1,500-foot level. News from the Mohawk, which, like the Ojibway, is one of the newer of the Keweenaw County properties, states that the ground tributary to shafts No. 5 and No. 6 has been improving with depth and at No. 6 some of the finest ore found on this property has been exposed.

At the Ahmeek, a Keweenaw property, the openings in No. 3 and No. 4 shafts are more than fulfilling expectations. An abundance of fine stamp rock is being found and developed.

The annual report of the Centennial mine emphasizes the statement that the Kearsarge lode in the Centennial ground is not a paying proposition, with metal selling at present prices.

Because of the state of the metal market local business houses have started a campaign of retrenchment. Many workers have been laid off and others will follow.

During the last six months thousands of people have moved out of the district. More are leaving every week and more will follow. Many are taking land in Minnesota, Washington, Oregon, Florida, and Texas. A large colony was recently organized, almost all copper country miners, in Michigan, to raise fruit.

COPPER CONVERTER FLAMES

*By Donald M. Levy**

There are two main stages in the process of Bessemerizing copper mattes. The first is essentially the elimination of the iron sulphide; the second, the final sulphur elimination.

The Successive Stages in the Bessemerizing of Copper Mattes as Indicated by the Converter Flame

The first stage of the process is known as the "slagging stage," and is characterized by a green flame caused chiefly by the formation of iron-silicate slag. The reactions during this stage are well known; the oxygen of the air which is blown in yields oxides of copper, iron, and sulphur, the former of these immediately reacts with the iron sulphide still remaining and reforms copper sulphide, with the production of more iron oxide. The iron oxide combines with the silica present, chiefly in the lining, and produces a ferrous silicate slag. The corresponding sulphur escapes as SO_2.

The chief product of this first stage is a white metal, practically pure copper sulphide.

When the slagging stage is completed, the slag is poured off and the white metal blown up to blister copper. This constitutes the second main stage of the process.

The chief reaction now is the elimination of the sulphur, and the production of metallic copper—brought about by the action of some copper oxide, first produced, on the copper sulphide present. The flame during this period is smaller, thin, fairly non-luminous, and red-purple to bronze-purple in color.

In addition to these two main color stages, there is a rich red-brown color on first blowing, and a blue-white color at the end of the slagging stage.

The first of these is partly due to the coal thrown in, also to foreign ingredients in the matte which are the first to be oxidized; possibly also to an undue proportion of sulphur burnt out at the lower temperatures which obtain on commencing the blow, and also to unslagged iron oxide.

The blue-white flame, which marks the finish of "slagging," is dealt with later.

The progress of the blowing from copper matte to white metal, and thence to blister copper, is indicated and controlled at the smelter by means of the appearance of the flame issuing from the nose of the converter. The successive changes in their aspect are quite noticeable, and after some practice the stages of the operation can be readily followed. The changes in form and color are gradual, but the skimmer at a glance is able to tell how his converter charges are progressing and to give a shrewd judgment as to the temperature, composition, and nature of his metal and slags.

In general character, the color sequence of the converter flames does not vary much in different localities, but the body and luminosity depend very largely on the nature of the charge and working conditions. The colors are intensified by very hot metal, high blast pressure, and rapid working; also to a great extent by the presence of what might be called secondary constituents in the matte, such as zinc, lead, or arsenic, which, liberating heavy white fumes, increase the luminosity considerably.

There are usually four distinct variations in the appearance of the flame, and these are indicated as follows:

At the commencement of the blow—oxidation of secondary constituents, burning of iron, sulphur, and coal; dark reddish-brown flame, accompanied by much smoke.

Slagging stage—iron sulphide oxidation and formation of iron-silicate slag; vivid apple-green flame.

White metal stage—copper oxidation and formation of copper silicate; white-blue flame.

Blowing to blister copper—sulphur oxidation; thin red-purple flame.

* Institution of Mining and Metallurgy, November, 1910.

The system of working off charges of matte in the converters varies at every plant, and even at the same plant, depending on the rate of production, grade and supply of matte, and its temperature; condition of converter lining, and the state of affairs at the casting and refining furnaces; so that all the cycle of changes may not be seen in regular succession at one converter, which might be kept blowing to white metal with successive additions of fresh matte, or blowing to blister copper with continual additions of white metal from other converters.

At the Washoe smelter, Anaconda, Mont., the normal working is to blow the matte to white metal, pouring off slags when necessary; then to blow up to blister copper, and pour the resulting metal in regular sequence.

The matte here averages 45 per cent. copper; about 8½ tons are charged at an average temperature of 900° C., with the converter in an upright position and with the blast of 16 pounds per square inch. A few lumps of coal are thrown in before and during the running in of the matte.

During this charging, copious and heavy white fumes and smoke and a full red to red-brown flame are emitted—this effect being, as previously stated, partly due to zinc, lead, or arsenic fumes, to sulphur and iron burning, and to the effect of coal on the flame.

The converter being turned slightly back, the blow proper commences. The flame has dropped to some extent, and continues red to red-purple for from 2 to about 8 minutes, after which green commences to show in the red smoky flame, when the first or slag-forming period properly begins.

The green becomes more and more prominent until the flame is altogether of a vivid apple-green color after about 15 minutes blowing. It is also brighter, fuller, and is accompanied by plenty of smoke; slagging of the iron being now in full operation and the metal very hot.

After some 40 to 45 minutes' blowing (depending largely upon the working conditions) flashes of blue appear occasionally in the flame. These gradually increase in number and finally the flame becomes white-blue.

This is a sign that most of the iron has been slagged off, that the white metal stage has been reached and that copper silicate is being produced, to which latter fact the color of the flame is to be attributed. The copper oxide now being formed appears to produce the silicate in presence of much slag rather than attack the copper sulphide, especially if the temperature be high.

The fact that the marked green and blue color effects are due to the formation of silicates of iron and of copper is interesting; these colors are not produced unless the respective oxides are being slagged.

When this white metal stage has been reached, the slag is poured off until an iron rake held under the stream begins to show signs of metal (which gives an appearance of spots of grease on the blade).

If considered necessary, a "dope" charge of scrap, cleanings, and silicious materials, etc., is now added—partly to cool the charge—then the converter is turned up and blowing resumed. Very often a preliminary skimming is carried out to prevent too great an accumulation of slag before the white metal stage is quite reached and while the flame is green, but giving indications of blue. Blowing is afterwards continued, the pure apple-green flame results, and white metal is eventually obtained. The first stage of the blow to white metal occupies about 60 minutes.

The second part of the blow commences with a vivid red-brown flame and smoke; but this gradually decreases and a thin red-purple flame of some brightness with thin smoke results and continues with very slight change to the end of the operation.

The temperature of the operation can be judged by the flame, a red-brown indicating the correct temperature. If too red, the metal is too cold and coal is thrown in; if the tint be too orange, the metal is too hot and "dope" is charged.

The end of the blow is most difficult to judge, the size and color being some criterion; but a very important guide is the emission of little shots of copper which no longer stick to the hood over the converter, but rebound from it. This is the stage where the judgment and skill of the skimmer is most tried. The metal is further tested by judging the appearance of a small quantity poured on to the floor; a rugged and uneven surface is satisfactory. The second period of the blow takes from 60 to 75 minutes.

Below are given observations made during a blow:

Converter A.—Matte charge. Charged 2:25 P. M., flame red; 2:29 P. M., green appears; 3:00 P. M., green becoming brighter; 3:17 P. M., vivid green; 3:18 P. M., skimmed, no metal reached; 3:21 P. M., blue appears in flame; 3:27 P. M., flame chiefly blue. Converter turned down to skim. Doped.

Converter B.—White metal charge. Charged 4:13 P. M., doped; 4:14 P. M., converter turned up; flame red; 4:16 P. M., flame red-purple; 4:18–5:20 P. M., flame peach to bronze-purple color.

The changes in composition of the metal during a blow have been shown graphically in a paper* by Mr. E. P. Mathewson, superintendent of the Washoe smelter at Anaconda, Mont.

The changes in the appearance of the flame during Bessemerizing are striking, and each has its own significance in connection with the various stages of the process. Their reproduction in color may assist in the realization of such effects which a verbal description could not convey.

The colors mentioned are representative of the reactions which occur in converting a fairly pure iron-copper matte when blowing with a high pressure. With low pressure the density of the colors is much lessened, and in addition, the presence of lead and zinc increases the luminosity of the flame.

Mr. W. A. Heywood says that "if nickel be present the color of the flame is bright green from start to finish, yet the man in charge of the converter is able to determine the finish of the blow by the appearance of the flame, regardless of the color."

Mr. Charles Olden says that "the coloration of the flame differs at various operations" and suggests that in addition to judging the end of the blow by flame coloration and the emission of shots of copper, rod sampling by the usual method should be followed.

"This would be as efficient and less troublesome than pouring a small quantity of the molten mass on the floor.

"The three tests would tend to prevent 'burning' and much wear and tear on the converter."

PREVENTION OF SCALE IN BOILERS

It has been discovered by a man in Hanover, Germany, that if hard water, previous to being fed into a boiler, were allowed to run over a plate of aluminum set at an angle and exposed to sunlight or diffused daylight, no scale would be formed through the action of such water on the boiler plates or tubes. The solid matter contained in the water deposits as a fine mud, which is easily removable on the lowest part of the boiler, consequently the tedious process of chipping off the scale is obviated. This process was tried at the Proprietary Company's Port Pirie Works, Broken Hill, Australia. Two plain aluminum sheets, each 4 feet by 2 feet were fixed in a frame inclined at an angle of 59 degrees, facing the sun. Water from the main was then allowed to flow over the plates from a perforated pipe fixed at their upper edge. A sufficient quantity of the water was run over the plates in the day time and stored to supply a Lancashire boiler for 24 hours. Mr. J. H. F. Hill, responsible for this treatment, states that the cause of the phenomena described has not been ascertained.—*Australian Mining Journal.*

* Bull., A. I. M. E., 1907, 7.

THE MAMMOTH CAVE OF KENTUCKY

Written for Mines and Minerals, by James H. Gardner

The Geological Formations and Natural Conditions By Which the Cave Was Formed

The Mammoth Cave is in Edmonson County, Western Kentucky. This cavern is one of the most widely-known objects of general interest to Europeans traveling in America, and it is said that in this country the percentage of people who have heard of the cave increases directly with the distance from it. It is a fact that natives living within 10 miles of the cave cannot direct one to it and others living much closer have never gone into it.

This section of Kentucky is noted for its natural rock houses, sink holes, and subterranean caverns. It is a limestone region and one especially favorable to the process of cave making. The drainage of this part of Kentucky is chiefly beneath the surface and one may at places travel 20 or 30 miles without crossing a surface stream. The prevailing topography consists of funnel-shaped basins 40 to 190 feet deep and draining 5 to 2,000 acres. Water collecting from gullies meandering out over these circular divides is drained by means of sink holes 3 to 15 feet in diameter, which communicate downward with underground channels. By this means the water is led off through subterranean streams to the major water courses where it emerges either at the edges of the valleys or else rises in the form of a spring beneath the main stream.

VIEW IN MAMMOTH CAVE, SHOWING ACTION OF WATER

The Mammoth Cave is in what is generally known as the St. Louis limestone of the Mississippian series, which, farther west in the state, Ulrich has divided into the Spergen, the St. Louis, and the Ste. Genevieve. In the Cave district, these limestone beds aggregate a thickness of about 500 feet. On the higher hills and ridges the limestones are capped by outliers of Chester sandstone, which is in turn overlain unconformably by a conglomerate sandstone corresponding to the Mansfield formation of Indiana and possibly in part with the Lee conglomerate of Virginia, Tennessee, and East Kentucky. The St. Louis is the lowest of the formations exposed in the section under discussion. It has a gentle westerly dip and outcrops over a wide area northeast, east, and southeast of the Mammoth Cave; an area abounding in smaller caverns. The relation of these formations to one another and to Green River which drains the district has an important bearing on the origin of the cave.

Origin of the Cave.—There is little doubt that the cave has originated in a most quiet and orderly manner and that the natural forces now at work enlarging it are the essential ones that have produced it. This view is at variance with the opinions advanced many years ago by Prof. Alexander Winchell and Prof. E. D. Cope. An abstract of theories made by Professor Winchell and given in Packard and Putnam's publication on the fauna of the cave,* 1872, is as follows:

"The country of the Mammoth Cave was probably dry land at the close of the coal period, and has remained such, with certain exceptions, through the Mesozoic and Cenozoic ages, and to the present. In Mesozoic times, fissures existed in the formation, and surface waters found their way through them, dissolving the limestone and continually enlarging the spaces. A cave of considerable dimensions probably existed during the prevalence of the continental glaciers over the northern hemisphere. On the dissolution of the glaciers, the flood of water which swept over the entire country, transporting the materials which constituted the modified drift, swept through the passages of the cave, enlarging them, and leaving deposited in the cave, some of the same quartzose pebbles which characterize the surface deposits from Lake Superior to the Gulf of Mexico. Since the subsidence of the waters of the Champlain epoch, the cave has probably undergone comparatively few changes. The well, 198 feet deep, at the farther end of the cave, shows where a considerable volume of the excavatory waters found exit. The Mammoth dome indicates probably, both a place of exit and a place of entrance from above. So of the vertical passages in various other portions of the cave."

Again Professor Cope's "general observations" in his article on the Port Kennedy fauna is quoted as follows:

"The origin of the caves which so abound in the limestones of the Alleghany and Mississippi valley regions, is a subject of much interest. Their galleries measure many thousands of miles, and their number is legion. The writer has examined 25, in more or less detail, in Virginia and Tennessee, and can add his testimony to the belief that they have been formed by currents of running water. They generally extend in a direction parallel to the strike of the strata, and have their greatest diameter in the direction of the dip. Their depth is determined in some measure by the softness of the stratum, whose removal has given them existence, but in thinly stratified or soft material, the roofs or large masses of rock fall in, which interrupt the passage below. Caves, however, exist when the strata are horizontal. Their course is changed by joints or faults, into which the excavating waters have found their way.

"That these caves were formed prior to the Postpliocene fauna is evident from the fact that they contain its remains. That they were not in existence prior to the drift is probable, from the fact that they contain no remains of life of any earlier period so far as known, though in only two cases, in Virginia and Pennsylvania, have they been examined to the bottom.

* Packard, A. S. Jr., and Putnam, J. W., editors *American Naturalist*, "The Mammoth Cave and Its Inhabitants," Naturalist Agency, Salem, 1872.

No agency is at hand to account for their excavation, comparable in potency and efficiency to the floods supposed to have marked the close of the glacial period, and which Professor Dana ascribes to the Champlain epoch. An extraordinary number of rapidly flowing waters must have operated over a great part of the Southern States, some of them at an elevation of 1,500 feet

Entrance to Mammoth Cave, from Inside

and over (perhaps 2,000) above the present level of the sea. A cave in the Gap Mountain, on the Kanawha River, which I explored for 3 miles, has at least that elevation."

These views are excusable on the ground that at that time the boundaries of North American glaciation and the areas covered by the drift were not known. But glaciation little more than crossed the drainage of the Ohio, and outwashed material never reached the drainage of Green River this distance above its mouth, hence the views of these eminent men have now become untenable. The quartzose pebbles found in the cave by Professor Winchell and mistaken for glacial pebbles are those which are secondary from the conglomerate sandstone and have been washed into it from the immediate vicinity. His mistake was a natural one: that of drawing a general conclusion from insufficient observation. The same may be said of the conclusion of Professor Cope.

The rocks in the vicinity of Mammoth Cave are very slightly disturbed and such a condition is necessary for the formation of large caverns. A thick massive terrane of limestone beds, slightly above drainage and to some extent jointed or fractured, presents the proper relations for the genesis of such a cavern as the Mammoth Cave. It is at once evident from a study of the cave and surroundings that dynamic forces have played their part in the creation of the cave only by producing the initial fractures or joint planes through which acidulated waters found their way downward to the level of Green River. The first stages in the growth of Mammoth Cave were brought about by percolation of ground water through joint planes probably developed by those forces which produced the Cincinnati Arch or perhaps to some extent by the subsequent settling of that structure.

In Kentucky and the adjacent states of Ohio and Tennessee, the writer has observed that the rocks having dips normal to the Cincinnati arch usually show a decided joint structure. The lead- and zinc-bearing veins of the "Blue Grass" section of the state occupy fissures roughly parallel to the axis of the fold and the writer has observed that the major jointing on both sides of this anticline for a long way back from the axis has as a rule a trend slightly east of north and west of south. These fractures are especially noticeable in the Devonian shales of Eastern Kentucky and at places are equally prominent in the rocks of the Mammoth Cave region of Western Kentucky.

It was from meteoric waters slowly traversing these joints in the massive limestones that the cave had its origin. The same holds true for the thousands of other caves in the area of outcrop of the St. Louis and Ste. Genevieve limestones west of the Cincinnati arch. The Newman limestone occupying a similar stratigraphic and structural position east of the arch is also a cavernous formation.

Geologically speaking, the Mammoth Cave is comparatively young, Green River, in modern geologic time, has cut through the Chester sandstone to a depth of about 200 feet into the underlying limestone. The cave action began when the river channel was lowered into the limestone. Ground water passing down through the smallest joints of the stone, then slightly above the level of the river, began the solution of what is now the uppermost levels of the cavern. These waters congregated along planes of least resistance to form Echo River which by both solution and erosion has been the secret of the immensity of the cave's development. This stream now flows in the deepest levels of the cave and empties into the waters of Green River. Echo River is now about 195 feet below the highest levels of the cave, which is approximately the vertical distance that Green River has cut into the limestone. The present entrance of the cave is in the east bank of Green River near the top of the limestone and is probably the original exit of Echo River. From time to time the course of Echo River has been changed by its waters finding a lower outlet, thus causing the stream to abandon its former levels. In this way the various avenues in the higher and dryer portions have been formed. In conjunction with the solvent and erosive action of the river are the effects of various tributary rivulets and floods from torrential rains. The various pits and domes are the products largely of the erosive action of flood water fed into the cave from the basin-shaped sinks on the surface. The destructive effects of this water are in some cases increased by the grinding action of pebbles inherited from disintegration of the conglomerate of the surrounding hills, chert fragments resulting from disintegration of the limestone. But it is the slow action of perennial waters that has been the chief agent in the growth of the cavern. The district is covered by a luxuriant growth of vegetation and it is from the damp humus of this forest that water charged with carbon dioxide finds its way into the subterranean channels, converting the limestone into soluble bicarbonate of calcium, the greater amount of which is carried

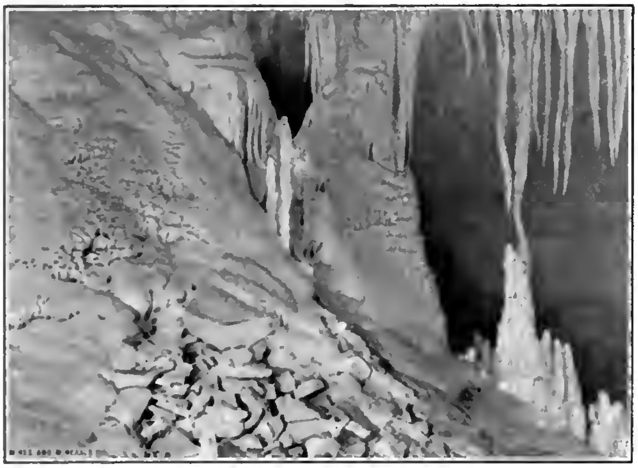
On Echo River

out via Echo River, but a portion of which is precipitated in the form of encrustations, stalactites, stalagmites, and pillars. This precipitation is found chiefly in the higher and drier portions of the cave where the waters are stronger in acid content and where evaporation is greatest.

The time required for the cutting of Green River to a depth of 195 feet into the limestone represents the age of the cave.

At the rate of 1 foot in 6,000 years (the estimate of removal of sediment from the Mississippi Valley) this gorge would have been cut in slightly over 1,000,000 years. If from this is deducted the more rapid cutting in upland streams, the calculation is brought to considerably less than 1,000,000 years, which according to Professor Shaler's estimate of geologic time, would probably not extend beyond the Pliocene.

Fauna.—The fauna of the Mammoth Cave is undoubtedly recent. For a time the subject of the blind fishes, insects and crustacea of this and similar caverns led to heated arguments in the application of the Darwinian and Lamarckian theories of evolution. But it is now generally conceded that these aberrant forms of life are specialized derivatives of the ordinary fresh-water or land fauna. The organs affected by the life of darkness are those directly concerned with the phenomena of the outer world, chiefly those of sight. The important modifications brought about by this change from light to darkness have led to the origin of new species by change of environment. The resemblance of the underground forms to those of the outside compares too closely for application of the theory that supposes these species to be the result of internal tendencies. Such organs as the eyes of fishes have been developed through the geologic ages from beyond the Silurian, yet in the comparatively short interval of the cave's formation these organs have



Stalactites and Stalagmites

not only lost their functions but nearly disappeared. There are two kinds of blind fishes in the Mammoth Cave, the smaller of which is slightly over 1½ inches long, the *Typhlichthys subterraneus.* The larger, slightly over 4½ inches long, the *Amblyopsis spelaeus.* In 1853, Professor Agassiz took a tour of the southern and western states and at that time wrote a letter to Prof. D. James Dana stating that he had discovered a new fish genus. This genus he named "*Chologaster*," giving the specific name "*cornutus*." This fish has eyes and is not a cave type, being found among the ditches of the South Carolina rice fields. It is much like the blind fishes of caverns, and a species named by Putnam, "*Chologaster Agassizii*," appears to be an intermediate type. The latter was obtained from wells of Tennessee and Alabama. There is little doubt that these types are closely related and show the transition in specific character from those of open surface waters to those living in constant darkness.

The crawfish, *Cambarus pellucidus*, first described by a German, Doctor Tellkampf, has very rudimentary eyes in the adults but larger in the young. This seems to be an evidence that the embryo develops like that of other species and that blindness is due to their physical surroundings and adaption to changed conditions of life. Specimens taken of the crawfish, *Cambarus Bortonii*, are not blind but have well-developed eyes and, one may assume, have been recently introduced into the cave.

What is commonly known as the cave cricket, *Hadenoecus subterranea*, offers another plain case of the effects of environment. The wingless locust, *Ceuthophilus maculatus*, which inhabits the outer world and lives ordinarily under stones is probably the ancestral type of *Hadenoecus subterranea.* Its legs have grown longer because of its life of clinging to walls, its antennae and palpi have elongated and become more sensitive organs of touch and hearing, and its body has partially bleached.

A small isoped crustacean, *Coecidotea stygia*, inhabits the cave and is closely allied to certain species which are all marine with the exception of two species which inhabit deep lakes. This similarity may be accounted for by the somewhat similar condition as regards light and temperature in caves and deep waters.

Within the cave are also small blind spiders, blind beetles, and a blind, hairy myriapod, *Pseudotromia Vudii.* Besides the blind myriapod here named, there are other species of "thousand legs" present that have eyes.

Hymenoptera and Memiptera are not represented in the cave. The flies or Diptera are represented by a few species with eyes, living in sections of semidarkness. The total of the cavern fauna includes, so far as known, 4 genera of infusoria, 2 vermes, 5 crustacea, 12 arachnida, 17 insecta, 1 mollusc, and 8 vertebrates.

GEOLOGIC PHENOMENA OF THE CAVE

Air-Currents.—At the entrance of the cave one notes a very perceptible air-current. This current flows out of the cave in summer and into the cave in winter. Its explanation is simple and is the result of equalization of air pressure due to differences in temperature of the outside and inside air. In summer the air in the cave is cooled by contact with the rocks and therefore heavier than the air outside and by gravity rushes out of the mouth. This has a tendency to produce a vacuum which is filled by warmer air drawn into the cave through its highest openings communicating with the surface. In winter the movement is reversed because the air within the cave is warmer than the exterior air and flows out of the highest opening to be replaced by cold air rushing in at the lower ones. The temperature of the cave is uniform at about 54° F. the year around, which is the same as that of Wyandot and Luray caves.

Calcium Nitrate Deposits.—Near the Mammoth dome, not far from the entrance of the cave, are deposits of calcium nitrate formed by leaching of water containing calcium carbonate in solution, through guano of bats. In this portion of the cave thousands of bats congregate to spend the winter. This calcium nitrate was used in the manufacture of gunpowder in the war of 1812. The nitrate was dissolved and leached through wood ashes in order to convert into potassium nitrate which was crystallized and sent by pack mules to the Ohio River.

Stalactites and Stalagmites.—Water charged with bicarbonate of calcium precipitates in the upper levels of the cave through evaporation, forming calcite onyx and structures common to limestone caverns. Drops of water so charged remain by adhesion on the ends of stalactites long enough to permit of evaporation which is stimulated by the currents of air through the cave. When a drop falls to the floor it is broken into spray thus increasing evaporation and further precipitation which builds up stalagmites from below.

Gypsum.—The origin of the gypsum crystals which line the walls and sides of certain portions of the cave is explained by the presence in the unweathered limestone of iron pyrites nodules. By exposure to air and moisture, the iron and sulphur are oxidized producing sulphuric acid and iron oxide; then a double decomposition takes place; the sulphuric acid acts on the limestone, forming gypsum and liberating carbon dioxide, which unites with the iron oxide producing iron carbonate. On further exposure the iron carbonate parts with carbon dioxide leaving a brown coating of iron oxide. These crystals of gypsum grow outward from the pyrites nodules, often forming beautiful crystal rosettes.

THE DESICCATION OF FURNACE AIR

Written for Mines and Minerals

Comparative Results of Refrigeration Methods and by Use of Calcium Chloride

In a general way the readers of Mines and Minerals are familiar with the Gayley dry air blast which has accomplished such economies in iron blast-furnace practice by the device of refrigerating and drying air. Many thousand tons of air are forced into a blast furnace daily, each pound of which contains a small weight in moisture depending on the weather conditions. The moisture thus carried into the furnace decreases the heat of the furnace and necessitates the use of more fuel to perform a given amount of useful work. Mr. Gayley, by passing the air through a refrigeration device precipitates the moisture, after which the air is passed through the hot-blast stoves and then to the furnace. By this arrangement not only is there an economy in fuel, but the output of the furnace is materially increased.

In the April issue of the *Compressed Air* magazine there is an article on "Successful Air Cooling and Drying" from which is abstracted the data and diagram relative to the Gayley process.

The data here presented are furnished by Mr. Gayley from a western plant for the month of July, 1910. The summer months represent the period of greatest humidity and highest temperature, and for that reason they furnish the best opportunity for demonstrating the efficiency and value of the process. While the atmosphere, as shown by the figures, varies widely in both temperature and moisture, the variations in the dry air are within narrow limits, and approach as close to uniformity as seems possible to obtain in a mechanical device that is treating 40,000 cubic feet of air per minute.

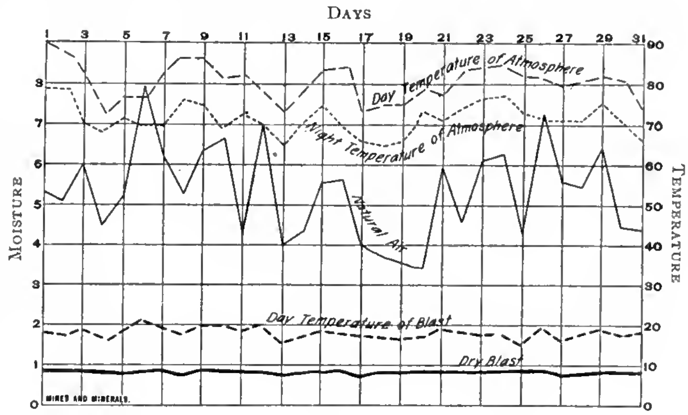


Fig. 1

The amount of work done by a dry-air plant in midsummer in removing moisture is ordinarily not fully comprehended by simply expressing the moisture content in grains per cubic foot of air. Taking, for example, a very humid day, July 6, when the moisture as shown in Table 1, averaged 7.90 grains for the day and night, there would have entered the furnace under natural air conditions 7,797 gallons of water in the 24 hours. This would be the equivalent of 185.6 barrels. The dry air on the same day contained only .86 grain, and the quantity of water entering the furnace was accordingly reduced to 849 gallons, thereby eliminating 6,948 gallons, and saving the fuel necessary to dissipate it. Taking again the day with the lowest humidity, July 19, when the moisture for day and night averaged 3.45 grains, the furnace would have received 3,410 gallons of water, but the dry air carried is only 809 gallons, representing an abstraction of 2,601 gallons. Thus, even on days of relatively low humidity, the quantity of water extracted is very large.

The amount of water extracted through the dry-air process expressed in gallons as seen in Table 2 is very impressive, and will be particularly appreciated by practical blast-furnace managers who are familiar with the cooling effect produced in the furnace hearth from a small leak from a tuyere or bosh plate. The diagram, Fig. 1, shows graphically the grains of moisture in the atmosphere and the resulting dry air. The uniformity of the dry air speaks for itself, as to its value in a process so delicate in adjustment and so variable as a blast furnace.

When the results of dry air were first made public in 1904, it was thought by some that the economy obtained was not due so much to low moisture in the dry blast, as to creating conditions of uniformity in the moisture—that is, if the moisture was maintained uniformly at 2 to 2.50 grains or 3 grains, the results would be practically as good as if it was reduced to 1.5 grains. This, however, does not appear to be borne out in actual practice, as the best results are obtained when the moisture is reduced below 1 grain per cubic foot of air, and is markedly greater at .75 grain than at 1.50 grains. The reasons for this do not seem at present to be clearly understood, although it has been demonstrated in practice.

The conclusion reached from the experience of the past 6 years is that dry-air blast conservatively considered will effect a saving of 10 per cent. in fuel, with an increase in output of 12 per cent., and the product can be increased beyond this at the expense of fuel saving, and vice versa. The tendency in some cases is to increase the output at the expense of fuel saving.

At one blast furnace the saving in coke was 7.5 per cent. on dry air, but concurrently the output was increased by 23 per cent. Thus the dry-air blast not only reduces the cost of pig iron, but it also creates uniformity in the furnace operations, and any cheapening of the pig iron cost is reflected to a greater extent in the finished steel product.

It is not necessary to offer any explanation of the diagram, which speaks for itself, except to call attention to the fact that the moisture scale is at the left hand and the temperature scale at the right hand.

Owing to the success met by Mr. Gayley, it was but natural that others should turn their attention to accomplishing similar results in some other way. Messrs. Felix A. Daubine and Eugene V. Roy conducted experiments on the drying of air by calcium chloride,* the results of which were so satisfactory that their process was installed at the Differdange Steel Works, Luxemburg. In a recent paper read before the Iron and Steel Institute a description of the Luxemburg plant was given and here reproduced.

It is inherent in the very nature of bodies that every substance the solutions of which, when highly concentrated, possess very low vapor tensions, should be, for that very reason, a hygroscopic substance, and capable of serving for the extraction, up to a certain state of equilibrium, of the moisture contained in atmospheres possessing higher vapor tensions than those of the solutions in question. The word "solution" should, in this connection, be understood in its most general sense—a solution of a given strength of concentration may be solid or liquid, this being a matter of temperature—and it will be seen later that one of the characteristics of the process is the maintenance of the solutions obtained in this solid "phase."

The more rapidly equilibrium is attained, the more suitable is the substance for playing its part as a desiccating agent. A highly hygroscopic body, at a given degree of concentration, would possess very low vapor tensions, differing greatly from the vapor tension of the air to be desiccated. This is precisely the case as regards chloride of calcium and certain of its hydrates, the very low vapor tensions of which permit rapid desiccation. In this connection the properties of calcium chloride have been known for a very long period.

It should also be noted that every external influence which is capable of lowering the tensions of the hydrates favors, by this very fact, the desiccation; and as, for any given hydrate,

* Journal of the Iron and Steel Institute, Vol. I, 1910, pages 613-615.

these tensions vary with the temperature, and in the same proportion, it follows that the lowest temperature possible should always be selected. Further, as the vapor tensions increase in proportion as the molecules of water absorbed

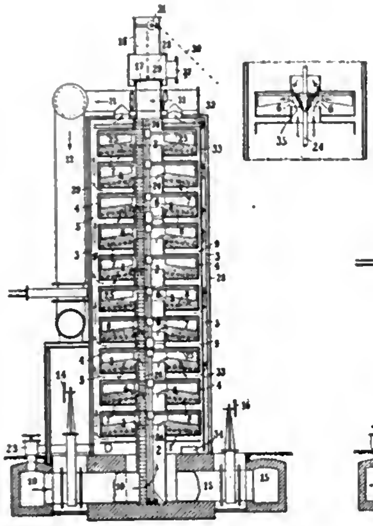


Fig. 2

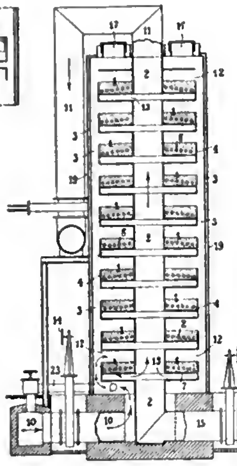


Fig. 3

render the dilution of the hydrates greater, it becomes equally necessary always to maintain the solution at a sufficiently high degree of concentration.

Consideration of these principles, taken together with numerous experiments, has enabled the authors to determine the economic conditions of the evolution of calcium chloride during the course of its periods of hydration and regeneration.

Table 1

1910	Grains of Moisture Per Cubic Foot Air				Temperature (Degrees F.)			
	Atmosphere		Dry Blast		Atmosphere		Dry Blast	
July	Day	Night	Day	Night	Day	Night	Day	Night
1	5.37	5.46	.85	.84	91	79	18.5	19.0
2	5.09	6.10	.86	.86	89	79	18.0	19.5
3	6.06	4.96	.85	.83	82	71	19.0	18.5
4	4.43	4.56	.84	.81	72	68	16.0	16.0
5	5.14	5.99	.82	.85	77	71	17.8	19.0
6	7.96	7.84	.86	.86	77	70	21.0	21.0
7	6.24	6.56	.86	.85	82	70	19.5	20.0
8	5.31	6.59	.79	.83	86	76	18.0	20.0
9	6.47	6.97	.86	.82	86	74	20.0	19.0
10	6.76	5.37	.84	.82	81	69	20.0	18.0
11	4.25	4.89	.82	.77	82	73	19.0	18.0
12	7.03	5.54	.84	.79	79	71	20.5	19.0
13	3.91	4.20	.78	.80	74	65	16.0	15.3
14	4.40	5.15	.79	.80	78	70	18.0	17.6
15	5.57	6.24	.81	.83	83	75	19.0	19.0
16	5.68	5.51	.85	.81	84	70	18.0	18.0
17	4.03	4.31	.79	.85	74	66	17.0	17.7
18	3.73	3.54	.81	.83	75	65	17.0	14.7
19	3.54	3.37	.80	.84	75	66	16.0	15.0
20	3.48	4.35	.81	.85	79	73	17.0	18.0
21	6.07	6.33	.82	.82	78	71	19.0	17.5
22	4.52	4.96	.83	.82	84	75	18.5	18.0
23	6.04	5.87	.83	.82	85	77	18.0	17.0
24	6.21	6.61	.84	.85	85	77	18.5	19.0
25	5.28	5.16	.83	.83	83	72	16.0	17.0
26	7.28	5.79	.85	.83	82	71	19.2	18.0
27	5.58	6.10	.80	.84	80	71	17.0	18.0
28	5.31	5.74	.82	.82	81	71	18.3	18.7
29	6.47	7.06	.82	.84	82	75	19.2	20.0
30	4.43	4.74	.83	.79	81	70	18.0	18.5
31	4.40	4.29	.81	.78	75	66	18.5	17.5

The problem at the Differdange works was to desiccate the air blast for a furnace of 150 tons daily capacity and at the same time realize the economies obtained in the Gayley process. Two forms of apparatus were designed, both of which are shown in section in Figs. 2 and 3. In the first design, Fig. 2, the blast enters through a central well and is distributed through openings to the various compartments containing calcium chloride. On escaping from these compartments the blast is collected in an annular chamber, whence it is led to the place where it is to be utilized. In the second design, Fig. 3, the blast enters an annular chamber and is collected in a central reservoir after traversing the layers of calcium chloride.

In order to regenerate the calcium chloride for reuse, the central reservoir or annular chamber is connected with hot-air pipes that admit gases to remove the excess moisture from the chloride of calcium. It having been found by experiment unsatisfactory to have a dilution of more than 8 parts of water this part of the apparatus requires attention.

Table 2

July, 1910	With Natural Air Gallons	With Dry Blast Gallons	July, 1910	With Natural Air Gallons	With Dry Blast Gallons
1	5,349	834	17	4,116	809
2	5,517	849	18	3,588	809
3	5,438	829	19	3,410	809
4	4,432	814	20	3,915	819
5	5,488	844	21	6,119	809
6	7,797	849	22	4,678	814
7	5,428	844	23	5,876	814
8	5,873	799	24	6,327	834
9	6,633	829	25	5,152	819
10	5,981	819	26	6,450	829
11	4,511	785	27	5,764	809
12	6,203	804	28	5,453	809
13	4,002	779	29	6,184	819
14	4,713	785	30	4,525	799
15	5,828	809	31	4,288	782
16	5,522	819			
Total, July				164,560	25,254
Equivalent in barrels of 42 gallons each				3,918	601

In Figs. 2 and 3 the same numbers refer to the same parts. The compartments represented by *1* contain the $CaCl_2$; *2* is a central shaft; *3* is a cover, which forms with the external shell; *4* a number of compartments; and *5* is the annular chamber.

Each compartment *1* has a circular inlet orifice *6* and a grid *7*, on which the chloride of calcium rests. An outlet *9* is provided below the grid.

Toward the end of the desiccation the air is drawn by means of the fan across the main lead *10* into the central shaft *2*,

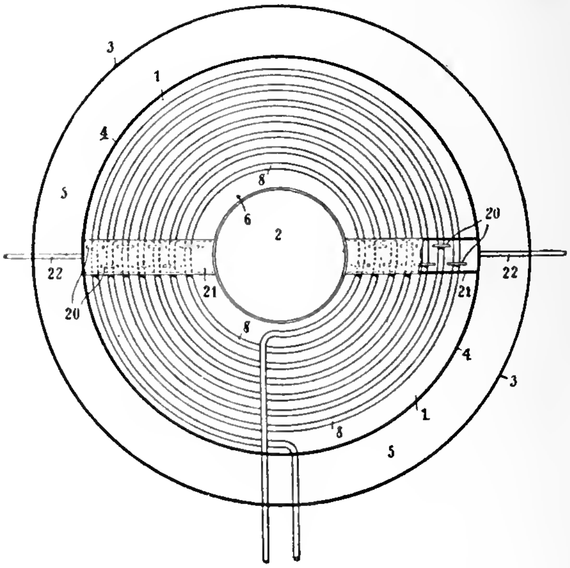


Fig. 4

and distributed to all the compartments. It traverses the chloride of calcium from above downwards, depositing its moisture; it then escapes into the annular chamber and reaches the conduit pipe *11*, which takes it to the place of utilization.

The chloride of calcium warms during the desiccation, and

this must be rectified by the cooling. It is with this object that water is made to circulate in the spirals, shown in Fig. 4, provided in each compartment. These spirals are arranged to have a certain inclination to provide for their being rapidly emptied. The joints are effected in a special box securing a water-tight separation for the chloride of calcium, with a discharge vent in case the joint becomes defective. The chloride is thus completely protected against such an event occurring.

When the chloride of calcium has absorbed the requisite amount of moisture, the apparatus is cut off from the inlet and outlet of air by the slide valve *14*, and communication is made by means of the slide valve *16* with the main *15* which conducts air or warm gases. As the temperature should only be increased gradually, the fan aspirates cold air at the same time as it takes in the gases to allow of the temperature being regulated. By diminishing the quantity of air, and increasing that of the gas, the temperature is gradually raised to 235°. The object of this is to maintain the hydrates in their solid form and in the course of this regeneration the temperature should not exceed 235°. When the calcium chloride has been regenerated, the apparatus is first cooled by means of a cold current of air entering chamber *5*, by the conduit pipe *23*, and escaping by the outlet *17*, and then water is introduced in the spiral cooling coils until the temperature has been sufficiently lowered. The apparatus is then ready to desiccate the air blast anew.

In the installation shown in Fig. 5 three apparatuses are employed, one of which dries the blast, while in the second the chloride of calcium is being regenerated, and the third is undergoing cooling. During the period of the passage of the blast or at the end of the period of cooling, it is possible to obtain access to the appliance either by the central shaft by means of the ladder shown in Fig. 2, or to the annular chamber by the ladder *33* running on rails at the top of *32*.

In the application of the process at Differdange, which is shown in Fig. 5, the conditions require the drying of 30,000 cubic meters* of air hourly. The apparatus was constructed in accordance with Figs. 2 and 3 and possess the following features:

Total area presented to the passage of the blast, 100 square meters per apparatus. Number of compartments, 10.

Depth of the layer of calcium chloride in each compartment, 24 centimeters.

It may be well to explain that 240 kilograms of calcium chloride spread out in a layer, 24 centimeters in depth, on a square meter of surface, will desiccate 300 cubic meters of air, per hour for 4 hours under average conditions of 15 grams of moisture per cubic meter.

Apparent density of the chloride of calcium.

Weight of chloride of calcium contained in each apparatus, 24,000 kilograms.

Weight of chloride of calcium in all three appliances, 72,000 kilograms.

Cooling surface of the spirals in each apparatus, 170 square meters.

These appliances have been designed to work in the most unfavorable conditions, that is to say, to remove during the summer months 15 grams of moisture per cubic meter of air during a period of 4 hours.

* 1 cubic meter = 35.315 cubic feet.

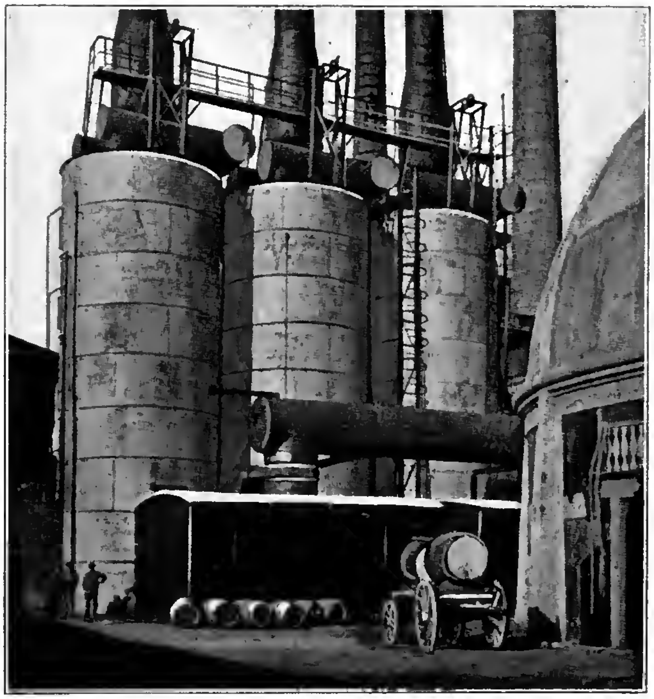


FIG. 5. AIR-DESICCATING APPARATUS

At the time of writing this paper the appliances had been working normally for 6 weeks, but as the season is the end of winter, and as the moisture in the air is not very abundant, it has been found unnecessary to make as many reverses as were contemplated. Each apparatus receives the blast from 6 to 8 hours. The air which contains 6 to 8 grams of moisture before its passage only contained from 1 to 15 grams per cubic meter on emerging from the apparatus, and this figure remains practically constant from the commencement to the conclusion of the period.

Regeneration requires 4 hours for its completion, and is carried out by means of the waste smoke gases from boilers and from Cowper stoves. These gases, cleaned to the extent of .4 gram per cubic meter, pass directly through the mass of chloride. The temperature is regulated at 30° to commence with, and thereafter gradually raised in conformity with a certain ascertained law up to about 200°. In the summer the temperature will be carried to 275°.

Cooling takes 3 hours.

It is interesting to compare these results of drying the blast with those which are capable of being given by the freezing method used by Mr. Gayley.

In Table 3 the authors reproduce the vapor tensions of ice and the amounts of moisture which saturate a kilogram of dry air at temperatures below 0° C.

From this table it may be seen to what temperature the air must be cooled to obtain results identical with those obtained at the Differdange Steel Works.

At 15° an amount of humidity of 1.5 grams per cubic meter corresponds with the content of 1.22 grams per kilogram of dry air. This figure, as may be seen, is well within the limits of the foregoing table. To obtain this result, calculation shows that it is necessary to cool the air effectively to −15°, which requires even considerably lower temperatures in the refrigerating tubes.

TABLE 3

Temperatures Degrees C.	Vapor Tensions in Millimeters of Mercury Per Cent.	Weight of Water Vapor in Grams in Saturated Air Per Kilogram of Dry Air Per Cent.
0	4.60	3.80
− 1	4.27	3.50
− 2	3.95	3.20
− 3	3.66	3.00
− 4	3.39	2.80
− 5	3.13	2.60
− 6	2.89	2.40
− 7	2.67	2.20
− 8	2.46	2.00
− 9	2.27	1.85
−10	2.09	1.70

This shows the advantage which the employment of calcium chloride has over that of refrigerating machines. While, with the latter system, cooling to −5° leaves a minimum of 2.60

grams per kilogram of dry air, the use of the calcium chloride regenerated under the form $CaCl_2+1$ to 2 parts H_2O and maintained in sufficient excess, at 15° allows of a degree of moisture as low as .5 gram per kilogram of dry air being attained.

In the winter months, during which the refrigerating system is practically inoperative, particularly in Continental localities, chloride of calcium allows of exceedingly low figures being attained—that is, distinctly less than 1 gram per cubic meter of air.

The installation is too recent for the true value of the economies to be stated, but it is established that the manufacture of dry air need present no difficulty whatever.

The installation has cost a little less than one quarter of what would have been the cost of an installation for desiccation by means of refrigerating machines. One man for the day shift and one for the night shift are sufficient to handle the apparatus, which is of the most simple description. The expenses of working are thus greatly reduced.

The authors state, in conclusion, that similar apparatuses are under investigation, not only for other metallurgical works, but also with a view to their application in other industries, that require drying operations carried out at low temperatures, as, for example, for the desiccation of chemical or pharmaceutical products, india rubber, resins, gums, gelatin, albumen, glue, aniline colors, and various organic bodies. It may thus be seen that there is a vast field open for the commercial production of dry air.

PERSONALS

John J. Lincoln, General Manager of the Upland Coal and Coke Co., Elkhorn, W. Va., had the American Institute meeting, the class day at Lehigh, and a business meeting in Philadelphia so arranged he could attend them all on one trip.

J. C. Welch has resigned as smelter manager for the East Butte Copper Mining Co., at Butte, Mont.

Edwin Ludlow, former general manager of the Mexican Coal and Coke Co., is now manager of the New River Collieries Co., with headquarters at Eccles, W. Va.

George A. Shroter, consulting engineer of the Mines Company of America, has returned to New York from Mexico.

Norman Fraser has resigned from the Crow's Nest Pass Coal Co. to take the management of the German American Development Co. in the Brazean fields, Alberta.

E. M. Thornby, E. M., of Los Angeles, Cal., has returned after a three months trip of inspection for English gold mine seekers.

Hood McKay, for a number of years connected with the Susquehanna Coal Co. as superintendent of Summit Branch Mining Co., Lykens, Pa., and district superintendent of Lehigh Coal and Navigation Co., Lansford, Pa., has been elected president of the O. S. Richardson Coal Co., with headquarters at Chicago.

Luther W. Balmey, assistant professor of mining and metallurgy at Leland Stanford University, has been appointed to a similar position at Sheffield Scientific School of Yale University.

M. L. O'Neale has been appointed superintendent of the Seaboard Coal and Coke Co.'s mines at Coal City, Ala.

Geo. I. Adams, formerly with the United States Geological Survey, later in the Philippine Bureau of Mines, and more recently in Peru, has returned to the United States, where he will continue his professional career as geologist. His present headquarters are at Washington, D. C.

Reese Hammond, formerly with the Lehigh & Wilkes-Barre Coal Co., has accepted the position of mine foreman with the American Gypsum Co., of Akron, N. Y.

J. B. Tyrrell, mining engineer, of Toronto, Can., and one of Canada's leading explorers, is writing on a new edition of Samuel Hearne's Diary and David Thompson's Journal.

Harwood Frost has severed his connection with the *Engineering News* and is now located at 226 La Salle Street, Chicago, Ill.

G. S. Rice, of Pittsburg Testing Station; Erskine Ramsey, of Birmingham, Ala.; A. B. Jessup, of Wilkes-Barre, Pa.; H. M. Warren, of Scranton, Pa.; and John Bart, of Windber, Pa., sailed from New York June 16, to study European safeguards against mining accidents and investigate the mining methods abroad.

LEHIGH UNIVERSITY GRADUATES

The subjects of theses presented by candidates for degrees in mining, metallurgy, and electrometallurgy at June, 1911, commencement at Lehigh were as follows:

FOR THE DEGREE OF ENGINEER OF MINES

Horace Daniel Bleiler (with P. N. Lopez), Frackville, "The Cyanide Process."

William Ewart Fairhurst (with G. R. Wood), Paterson, N. J., "The Cyanide Process."

Pedro Nicolas Lopez (with H. D. Bleiler), New York, N. Y., "The Cyanide Process."

Carl Weaver Mitman, B. A. (Lehigh University), South Bethlehem, "The Pre-Cambrian Limestones in Pennsylvania."

Charles William Rauch, Bethlehem, "The Oxford Furnace, N. J., Iron Mines."

Louis Allgaier Rehfuss, Philadelphia, "Proposed Plans for the Development of a Mexican Mine."

James Humble Smith, Jr., Mount Carmel, "Development of Bituminous Coal Property in Kanawha County, W. Va."

Felix Frank Trotter, Jr., Albuquerque, N. Mex., "Flow of Underground Water at the Friedensville Zinc Deposit."

Leon Wittgenstein, Louisville, Ky., "Magnetic Ore Concentration."

George Reid Wood (with W. E. Fairhurst), Pottsville, "The Cyanide Process."

FOR THE DEGREE OF METALLURGICAL ENGINEER

Robert Fulton Crawford, Steubenville, Ohio, "A Study of the Thermal Efficiency of a Zinc Furnace."

Paul Robert Snyder (with C. C. Walters), Bethlehem, "The Influence of Various Paint Coatings on Radiation Losses From Furnaces."

Albert Poole Spooner, Harrisburg, "A Study of the Duplex Process of the Bethlehem Steel Company, South Bethlehem Pa."

Clarence Claitan Walters (with P. R. Snyder), Bethlehem, "The Influence of Various Paint Coatings on Radiation Losses From Furnaces."

FOR THE DEGREE OF ELECTROMETALLURGIST

Moses Appel, Baltimore, Md., "The Electrolysis of Copper Matte."

Joseph Ralph Dawson (with N. M. Downs), Washington, "The Physical Properties of Electroplated Alloys."

Nelson Miller Downs, E. M., Lehigh University (with J. R. Dawson), Steelton, "The Physical Properties of Electroplated Alloys."

Maurice Good, Havre de Grace, Md., "The Regeneration of Cyanide Solutions by the Use of Cyanamid."

Thomas Claude Kraemer, Pottsville, "A Proposed Method of Electrolytically Treating Speiss."

Claude Calvin Messinger, Allentown, "The Production of Ferro-Boron From Colemanite."

COST OF CYANIDE PROCESS

In 1894, the cost of treating Cripple Creek ores by the cyanide process was $15 a ton; in 1910 it was $1.50 a ton, with a prospect of a further reduction to $1.25 and $1 a ton. This has added very largely to the field for profitable ore reduction, not only in Cripple Creek but throughout Colorado.

DEVELOPING POVERTY GULCH CLAIMS

By Chas. W. Henderson

(Continued from June)

Plans for Development.—Plans are drawn for an inclined shaft to follow the intersection of the dikes in the foot-wall. All prospecting work has been stopped on the chutes *A* and *B*;* one hoist is to be used in sinking the inclined shaft, while one is to be used to take ore out of the prospect hole and to continue the exploration work.

Cost and Method of Sinking Shaft, Drifting, Constructing Underground Ore Bins, Etc.

It is assumed that by the time the inclined shaft has reached the first level drifts already excavated, the plans for the surface buildings will be finished and the cost determined, and that by the time the shaft has reached the second level the contract will be let for building the surface structures, and all mining equipment will be ordered. This gives 210 days for the erection of surface buildings and for machinery to arrive while drifts and raises are being driven.

The first consideration is the construction of a spur 5,010 feet long to the Colorado Springs & Cripple Creek District Railroad on a 4-per-cent. grade, with a switchback at the 3,400-foot station.

The cost per mile of this track is estimated as follows:

Grading, all earth excavation	$ 7,000.00
Ties, 2,112, at $1 (8½ feet long ×7×9, placed 2½ feet apart)	2,112.00
96 tons of 60 pound rails, at $58 delivered	5,568.00
9,244 spikes, 4,622 pounds, or 31 kegs of 150 pounds	150.00
Tracklaying	600.00
Joints, 352 per mile	300.00
Small wooden bridges	400.00
Per mile	$16,130.00
For 5,010 feet	$15,300.00

Preparations are next made to sink the shaft in the foot-wall about 50 feet below the ore at the intersection and at a distance of about 20 feet on either side from the ore extending along the two dikes. Conditions warrant sinking a shaft at an inclination of 22° 42′, as the work must necessarily be a series of steps of shafts and level, winze and level, etc., if carried on. This necessitates tramming to the foot of the winze, hoisting to the top of the winze, tramming to the shaft, and then out.

For development work, the 8-horsepower electric hoist and the prospecting head-frame on hand can be used. The size of a shaft to go down 1,000 feet on the incline and large enough to accommodate the handling of 200 tons of ore a day and to accommodate all future increase in tonnage, is 12 ft.×7.5 ft. in the clear. This sized shaft belongs in the middle class where it is about as cheap to put down a fair-sized shaft as to put down a small one, as a small shaft will allow for only a small number of workmen.

With the addition of a small compressor and air drills the equipment left from prospecting operations can be used, including the buckets and old shafts of the three prospects. The prospect head-frame, Fig. 1, is fitted with a simple automatic dumping device.

The shaft is timbered with the sets shown in Fig. 2, the track being laid full 33½-inch gauge for the skip.

When the shaft has been sunk a short distance, two skips are added to the equipment. These weigh 550 pounds, hold 10 cubic feet (930 pounds), and cost $60 each, with an addition of $10 for water valve.

With these skips it it proposed to hoist all the material broken in the drifts and to handle the small amount of water encountered. The 8-horsepower hoist handles the rock broken during sinking and drifting.

There are purchased also, 16 standard ore cars, capacity 14 cubic feet, fitted for 24-inch track gauge, No. 4 thickness of steel (¼ inch), weight 570 pounds, and price $45 each.

In using these cars before the ore pocket is built, the cars are loaded only enough to fill the skips.

* See Map, June Mines and Minerals, page 694.

Cost of Equipment

Six Little Giant drills, 2¼-inch diameter	$ 900.00
Six columns and arms up to 8 feet	240.00
Six sets drill steel, each 15.4 pounds, at $14.40	86.40
Buffalo exhaust fan, diameter outlet 24½ inches, price with bed and countershaft	700.00
460 feet of 24-inch pipe, at $54 per 100 feet	248.40
1,200 feet of 12-inch pipe, at $27 per 100 feet	324.00
Ten Leyner No. 5 stopers, $135	1,350.00
Drill steel, 10 sets, at $15	150.00
Sixteen ore cars, at $45	720.00
Compressor, motor and pipe, freight	2,689.14
Freight on 19,553 pounds, at $.55 per 100	107.54
Total	$7,515.48

In the development work two drills will be needed in shaft sinking and four drills in drifting. The drills are to be 2¼ inches in diameter and use air at 90 pounds pressure per square inch at drill. According to the catalog specifications, each drill will need 67.2 cubic feet of air per minute. The factor to determine a compressor capacity for four drills at 10,000 feet altitude is 4.49; hence, 301.5 cubic feet of air per minute will be required, but deducting 5 per cent. for leakage and allowing the compressor a volumetric efficiency of 80 per cent., the total air required is nearly 400 cubic feet per minute.

E. A. Rix allows 20 horsepower for every 100 cubic feet of cylinder displacement, to compress air to 90 or 95 pounds gauge pressure at sea level. Although 20 horsepower is higher than the value given by Peele for the theoretical horsepower required, and figuring efficiency the figures would then be below 20; but, since compressors are usually purchased for excess power to supply possible additional uses, and the use of 20 horsepower would only add a small percentage on the safe side, the power necessary for four drills is taken at 80 horsepower, and it is decided to purchase a two-stage air compressor 18″×11″ diameter, 12-inch stroke, 125 revolutions per minute, with a capacity of 440 cubic feet at 10,000 feet elevation.

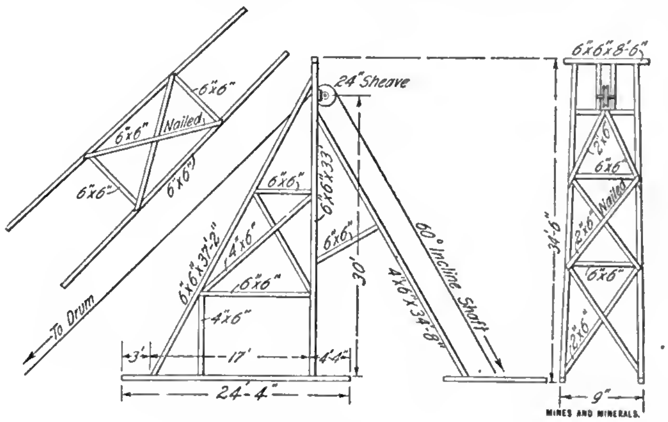


Fig. 1. Prospect Head Frame

This sized compressor gives a reserve of 21 per cent., and costs with freight from Denver to Cripple Creek $1,903. The motor for the compressor is 80 horsepower, 900 revolutions per minute, 440 volts, and costs delivered $710.84.

The following calculations give the power consumed during development by two piston drills: The catalog multiplier is 2.39 and as each drill will need 67.2 cubic feet of air, 67.2×2.39 = 159. Allowing for air loss in pipe line and efficiency of compressor, 210 cubic feet are necessary, or 42 horsepower per minute.

Each stope drill requires 25 cubic feet of air per minute and the factor for seven drills is 7.55; therefore, 25×7 = 189 cubic feet, to which 63 cubic feet is added to allow for loss and efficiency, and this is equivalent to 50.4 horsepower.

The power for two stope drills is 25×2.5 (multiplier) = 62.5 cubic feet of air, and if to this be added 20.7 cubic feet for pipe loss and efficiency, the power required is 16.68 horsepower. The diameter of the pipe needed for carrying air 800 feet is 3 inches and will cost at Cripple Creek $75.30. The total cost of compressor, motor, and pipe is $2,689.14.

Shaft Sinking.—The area of a 12′×7.5′ shaft is 90 square feet, and the arrangement and number of drills decided on is shown in Fig. 3. Each of the 18 holes will be 4 feet deep, and one man with a 2¼-inch diameter drill is capable of drilling 39 feet in 8 hours, or at the rate of 4⅞ feet per hour. If two drillers are put at work they can put in the 72 feet in 7.4 hours.

In this way an advance of 3 feet is made every round; blocking out 3×12×7.5=270 cubic feet of solid material, which is equivalent to $\frac{270}{12.4}=21.8$ tons broken each round, or

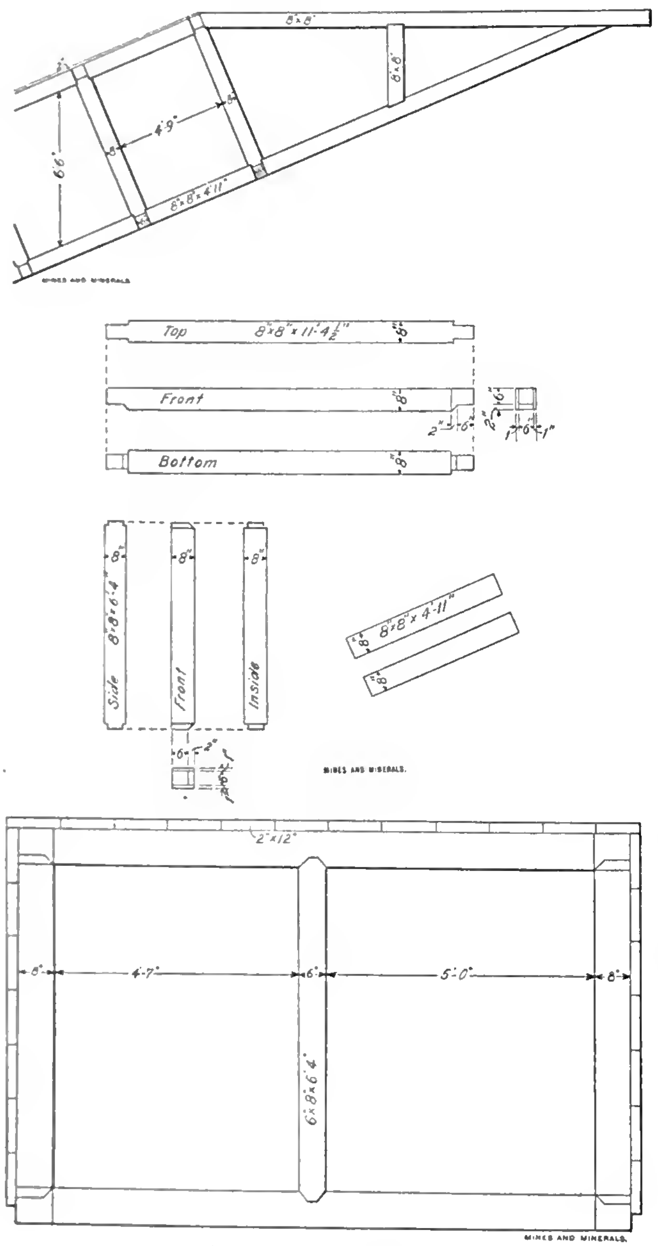


Fig. 2. Shaft Timbering

21.8×21.5 cubic feet=469 cubic feet,=17.4 cubic yards loose. To estimate the cost of handling the material: It will take two men $\frac{17.4}{2.4}=7\frac{1}{4}$ hours to muck out the rock broken in one round, and if ¾ hour be allowed for delays, changing skips, etc., one round can be drilled, fired, and removed every 16 hours. With this rate of progress the advance will be 90 feet per month.

To estimate the explosive required, take the depth of each hole at 4 feet, and assume that six sticks of 40-per-cent. dynamite, each weighing $\frac{6}{10}$ pound, or 3.6 pound per hole, are required; then 18 holes, per round, require 64.8 pounds of explosive.

The shaft is to be sunk 570 feet, a distance of 461 feet to reach the lowest workings, and then an allowance of 89 feet to provide room for underground ore bin to be made and 20 feet additional for a sump. The time required to sink the shaft will be

$$\frac{570}{3\text{ ft. per round}}=190\text{ rounds, or days.}$$

The detailed cost of sinking 570 feet of a 90-square foot inclined shaft is as follows:

Cost of Shaft Sinking

Two machine men, 190 shifts, at $4.50		$ 1,710.00
Two muckers (also top men) 190 shifts at $3		1,140.00
Two hoistmen, 190 shifts at $4.50		1,710.00
One blacksmith, 190 shifts at $4.50		855.00
One blacksmith helper, 190 shifts at $4		760.00
One foreman, 190 shifts at $4.50		855.00
One superintendent, at $175 per month, 6⅓ months		1,108.35
One timberman, 190 shifts, at $3.50		665.00
Powder, 12,312 pounds, at $.127*		1,563.62
Fuse, 190 rounds, 7-foot lengths, 23,940 feet, at $.0035*		83.79
Caps, 3,420, at $.007*		23.94
Depreciation on steel		14.40
Operation compressor plant (power), 336 horsepower hours per day:		
Installation charge	$ 20.50	
40,000 kilowatt hours, at $.013	520.00	
7,700 kilowatt hours, at $.005	38.50	579.00
Timber, 95,440 feet, at $20 per thousand		1,908.80
Electric power, for hoist, average 6.23 horsepower hours per hour, 49.84 horsepower hours per day, 9,470 kilowatt hours (190 days) at $.013		123.11
Coal for blacksmith, 28.75 tons, at $20.75		243.20
Candles, 950, at $.0145		13.78
Rails (30-pound), 570 feet, 5.08 tons, at $50		254.00
Cost for 570 feet		$14,207.55
Or cost per foot		$ 24.93

* These low costs of powder are those of the Portland Gold Mining Co. and include freight and unloading charges.

Cross-Cuts and Stations.—At vertical depths of 52 feet and 178 feet, two stations are to be cut the width of the shaft and 10 feet high in order to allow timbers, steel, etc., to be handled. The stations, Fig. 4, are made 14 feet long, 12 feet wide, and 10 feet high, narrowing at the cross-cuts to 7 feet wide, 7 feet high, and a length of 30.4 feet, at which point the mouth of the underground ore bin is reached. This demands a cross-cut at right angles to the cross-cut from the station, 13 feet wide, 7 feet high, and 17.25 feet long to connect with the drifts.

Underground Ore Bin.—A raise is to be started from the shaft to connect with the bin mouth and form an ore bin, Fig. 5. It is to have an average length of 20 feet, average width of 13 feet, and a height of 6 feet.

The two raises are to be made with Leyner No. 5 stopers. One drill is capable of making holes that when blasted will break 8 tons. As there are 20×13×6 =1,560 cubic feet, $\frac{1,560}{12.4}$=about 126 tons in an ore bin, and one man in each raise will break 8 tons, it will require $\frac{126}{8}$=about 16 shifts to drive one raise. This will produce 126×21.5=2,710 cubic feet, or 100 cubic yards broken material, and require $\frac{100}{1.1}=91$ hours, or 12 shifts, for one man to load into cars.

The bill of timber for the underground ore bin is as follows:

Number Pieces	Size	Board Feet
2	8″×12″×19′ 4″	320
2	8″×12″×15′ 8″	256
4	6″×12″× 5′ 9″	144
5	3″×12″×16′ 0″	240
11	3″×12″× 5′ 9″	198
5	2″×12″×11′ 0″	120
4	2″×12″× 5′ 0″	48
16	3″×12″× 9′ 0″	480
1	2″×12″×50′ 0″	100
7	3″×12″× 5′ 9″	126
2	8″× 8″× 7′ 0″	85⅓
2	8″×12″×12′ 0″	192
1	8″× 8″×10′ 10″	64
Total board feet		2,373⅓

The steel needed for an underground ore bin of the size specified is: Iron plate, 24 in. × 7 ft. × ¼ in. = 142.8 pounds; iron chute (total area) 4 ft. × 5 ft. × ¼ in. = 204 pounds; grizzlies, 5 ft. × 13 ft., space between bars, 3 inches, ⅝-inch rods, 65 square feet, 14 pounds per square foot, total, 910 pounds, at $1 per square foot.

The powder that will be consumed in raises for an underground ore bin is: 4 pounds powder per ton, or 504 pounds; fuse, 4 feet per ton, or 504 feet; caps, 1.4 per ton, or 177 caps.

The station, which is to be 14 feet long, will need 24 holes in the face, each 3 feet deep, in order to make an advance of 2 feet per round. Each hole will require four sticks of dynamite, or 2.4 pounds per hole, and 57.6 pounds per round.

One man can put in 39 feet of holes in 8 hours, or $4\frac{7}{8}$ feet per hour, and because there are 72 feet in each round it will take $\frac{72}{2\times 4\frac{7}{8}} = 7+$ hours, and with two men laboring per round, this station will take $\frac{14}{2} = 7$ rounds, or 7 shifts, and use up $7\times 57.6 = 403.2$ pounds of 40-per-cent. dynamite.

An estimate of the labor required is as follows: $10\times 10\times 14 = 1{,}400$ cubic feet of material broken, or $\frac{1{,}400}{12.4} = 113$ tons broken for the station. 113 tons × 21.5 = 2,430 cubic feet of loose rock, or 90 cubic yards, which will require $\frac{90}{1.1} = 82$ hours for one man to load in cars. Two men will load the material in 41 hours, or $5\frac{1}{8}$ shifts, or to tram the material will require $114\times 2\times 2 = 456$ minutes, about 8 hours, or one shift. The labor needed, therefore, will be seven shifts of two machine men, six shifts of two muckers.

The progress and powder on the proposed cross-cuts is estimated next; the dimensions of each cross-cut being 7 ft. × 7 ft. × 30.4 ft. Fifteen holes are needed in the face, each 3 feet 6 inches deep, to furnish an advance of 2.44 feet per round. Each hole will require 2.88 pounds dynamite, or

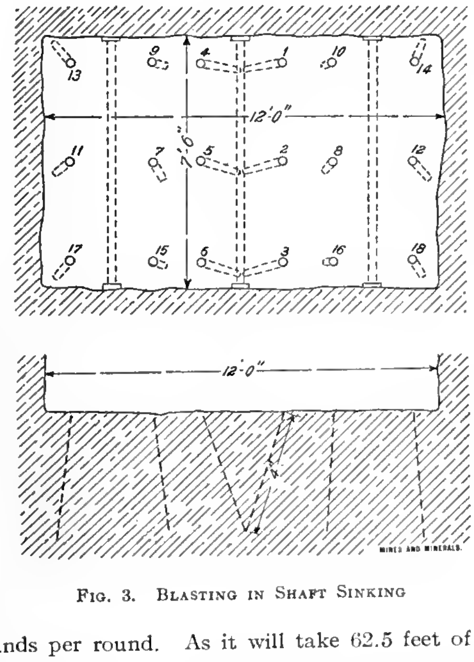


Fig. 3. Blasting in Shaft Sinking

43.2 pounds per round. As it will take 62.5 feet of holes for one round, the time occupied in drilling will be $62.5\times\frac{4}{39} = 6.4$, say 7 hours, and there will be $\frac{30.4}{2.44} = 12.45$ rounds.

In all it will take about 12 shifts to make this cross-cut, with 537.84 pounds of powder. The progress in the second cross-cut, whose dimensions are 13 ft. × 7 ft. × 17.25 ft., provided there are 24 holes in face, and 57.6 pounds powder per round, will be, when it takes 7 hours to put in one round, about $\frac{17.25}{2} = 8.63$ rounds, or 9 shifts. The nine rounds will consume 518.4 pounds of 40-per-cent. dynamite.

7 ft. × 7 ft. × 30.4 ft. = 1,489.60 cubic feet broken in first cross-cut. 13 ft. × 7 ft. × 7.25 ft. = 1,569.75 cubic feet broken in second cross-cut, or a total of 3,059.35 cubic feet broken in

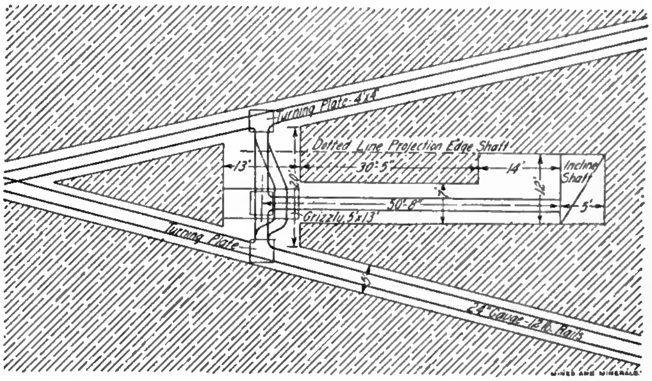


Fig. 4. Plan of Station

both cross-cuts, or $\frac{3{,}059.35}{12.4} = 247$ tons solid broken, which is equivalent to $247\times 21.5 = 5{,}310$ cubic feet, or 196.5 cubic yards of loose material, and it will take $\frac{196.5}{2.2} = 89.4$, say 90 hours, or about 12 shifts, for two men to load the broken material in cars, but there are 594 half-ton cars to be filled, and these, trammed at the rate of one every two minutes, takes up 1,188 minutes, or 20 hours, or the time of one man $2\frac{1}{2}$ shifts.

Cost of Cross-Cuts and Station

One superintendent, 14 days, at $175 per month	$ 81.66
One foreman, 14 days, at $4.50 per day	63.00
Two machine men, 28 shifts, at $4.50 per day	252.00
Two muckers, 28 shifts, at $3 per day	168.00
Powder, 1,459.24 (pounds), at $.127	185.32
Caps, 487, at $.007	3.41
Fuse, 1,461, at $.0035	5.11
Drills (depreciation)	.84
Candles, 616, at $.145	8.95
Rails, 85 feet, 12-pound rails, .327 gross ton, at $47	15.37
Thirty ties, 4 in. × 6 in. × 3 ft., at $.41 (freight, $3.30)	15.60
Turnplates, 4, at $18	72.00
One blacksmith, 14 days, at $4.50	63.00
One helper, 14 days, at $4	56.00
Coal for blacksmith, 5.6 tons, at $20.75	116.20
Freight, rails, etc. 733 lb. } 1,453 lb. at $65 per 100; Freight, 4 plates, 180 lb. each, 720 lb. }	9.44
Hoist (power for) 520 kilowatt hours, at $.013	6.76
Compressor power, 4,704 kilowatt hours, at $.005	23.52
Cost for about 68 feet	$1,146.18
Cost per foot about	$16.61
Cost of two stations and cross-cuts	$2,292.36

Cost of Underground Ore Bins

(Superintendent, foreman, and one blacksmith salaries included under Stations and Cross-Cuts.)

One machine man, 16 shifts, at $4.50	$ 72.00
One mucker, 12 shifts, at $3	36.00
Powder, 504 pounds, at $.127	64.00
Fuse, 504 feet, at $.0035	1.76
Caps, 177, at $.007	1.24
Compressor power, 133.36 horsepower hours per shift, total 1,590 kilowatt hours, at $.005	7.95
Hoisting power (included under stations)	
Candles, 112, at $.0145	1.62
Timber, 2,374 board feet, at $20 per thousand	47.48
Iron plate, 142.8 pounds } Iron chute, 204 pounds } at $.02¼	7.80
Grizzlies, 65 square feet, 14 (pounds) per square foot, $1 per square foot	65.00
Freight on iron, 1,256.8 (pounds), at $.65 per 100	8.17
	$313.02
Cost of two ore bins	$626.04

Drifts.—The work of drifting, if carried on by hand, will be hampered by bad air and the necessity of hoisting in two stages, but the advance will be about 800 feet on the 60-foot level. The stations have been cut at vertical depths of 52 feet and 178 feet, and cross-cuts run to connect with the hand-

drilled drifts. A compressor has been in use for the machine drills for shaft sinking for the stations and cross-cuts, and the drifts are now to be driven by machine drills.

The drifts are 7 ft. × 5 ft. in the clear, grade, ½ per cent.; rails, 12-pound; 24-inch gauge; ties, 4 in. × 6 ft., and placed on 3-foot centers.

The Portland Gold Mining Co. has decided that the small 2¼-inch hammer drills are decidedly better for cross-cutting and drifting.

The results obtained in working the property of the Portland company show that there is no ground in their property

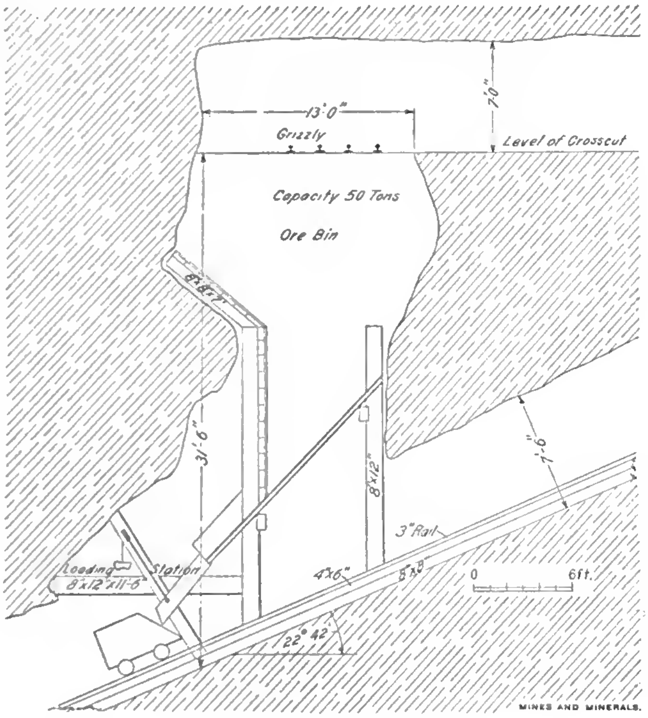


FIG. 5. UNDERGROUND ORE BIN

which cannot be handled with the small machine. During a period of 2 years there have been driven with them 4 miles and 308 feet of development headings, through a diversity of ground, including Pike's Peak granite (a coarsely porphyritic type of granite), highly indurated, andesitic, or phonolitic breccia; true massive andesite; trachitic phonolite; tufas; and along dikes of decomposed basalt and hard phonolite.

The method of driving drifts is shown in Fig. 6. As shown, nine holes are drilled in the face, each 3 feet 6 inches deep, and with these blasted an average advance of 2.44 feet per round is made. Each hole has an average charge of 4.8 sticks of 1¼"×8" dynamite, weighing $\frac{6}{10}$ pound per stick, or $9\times4.8\times.6=25.92$ pounds per advance of 2.44 feet.

One man can put in 39 feet of holes in 8 hours, or 4⅞ feet per hour; but as one man is required to drill only 31½ feet per round, it will take him $\frac{31.5}{\frac{39}{8}}=6\frac{1}{2}$ hours.

Aside from the usual work of loading and firing, the machine man helped the mucker to place flat steel plate 48×96×⅜ in. in place during 1¼ hours of his shift. The number of tons of material to be loaded and trammed to the shaft is found as follows: $\frac{7\times5\times2.44}{12.4}=\frac{85.4}{12.4}=6.89$ tons (solid), or 6.89×21.5 cubic feet = 148 cubic feet, or 5.5 cubic yards loose.

A man can shovel 1.4 yards from steel in 1 hour, and 5.5 cubic yards in say 4 hours.

The 600 feet of drifts are to be worked in four faces, each furnishing 6.89 tons of material per day, or 14 half-ton cars. When 56 cars are to be trammed per day at the rate of one every 4 minutes, it will take one man 3 hours and 44 minutes to tram all the cars.

Three muckers should be able to load and tram, lay the track, and load the skips in 8 hours.

Cost of Drifts.— 2.44 feet advance made in each heading per shift equals 9.76 feet in the four headings, which estimate places the time at $\frac{600}{9.76}=61.4$, say 62 shifts to run drifts.

One superintendent, at $175 per month, 31 days	$ 180.85
One day foreman, 62 shifts, at $4.50	279.00
One night foreman, 62 shifts, at $4.50	279.00
Four machine men, 62 shifts, at $4.50	1,116.00
Three muckers, 62 shifts, at $3	558.00
One hoistman, 62 shifts, at $4.50	279.00
One blacksmith, 62 shifts, at $4.50	279.00
One helper, 62 shifts, at $4	248.00
Powder, 248 rounds, 6,428.16 pounds, at $.127	816.38
Fuse, 2,232 holes, 3 feet, 6,696, at $.0035	23.44
Caps, 2,232, at $.007	15.64
Depreciation on steel	13.20
600 feet 12-pound rails and fastenings, 2.32 tons, at $47	109.04
200 ties, at $.41 each (plus freight, $21.12)	103.12
Cost of compressed air, 29,600 kilowatt hours, at $.005	148.00
Power for exhauster after firing (20 minutes a shift) 65 kilowatt hours, at $.005	.32
Candles, 936, at $.0145	13.57
Coal for blacksmith, 3.9 tons, at $20.75	80.93
Power for hoist, 2,300 kilowatt hours, at $.013	29.90
Freight on 4,640 (pounds) of rails and fastenings, at $.65 per 100	3.02
Cost for 600 feet	$4,575.41
Or cost per foot	$7.63

Including the two cutting-out subdrifts, there are three drifts worked at one time, making a total cost of $13,726.23.

The cost of driving the two subdrifts includes rails and ties, which are removed before stoping, and can be used elsewhere. There are, therefore, as an asset, rails and ties to the value of $212.16 for 1,200 feet of drifts.

Raises.—The five raises needed for manways and ventilation are put up from the second level (178.4 feet vertical depth); two are to be 139 feet long, and three are to be 155.5 feet. There are five raises to be put up from the first level (52 feet vertical depth); two of which are to be 39 feet long, and three are to be 48 feet long. These raises are to have a size of 4½ ft. × 6½ ft.

There are also twenty-four 6-foot raises to be driven for chutes, or 144 feet in all. The total length of the manways will be 966.5 feet, making a grand total, 1,110.5 feet. The quantity of material to be broken is $\frac{1,110.5\times4.5\times6.5}{12.4}=2,620$ tons, and because every stoper should break 8 tons per day the work

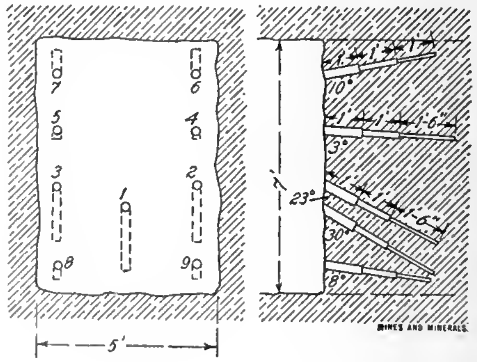


FIG. 6. BLASTING IN DRIFTS

should be accomplished in $\frac{2,620}{8}=327.5$ shifts. Seven men can work in these raises at one time without getting in each other's way, which gives an estimate of $\frac{327.5}{7}=$ about 47 shifts to a raise.

Cost of Raises

Powder, 10,480 pounds, at $.127	$1,320.96
Fuse, 10,720 feet, at $.0035	37.52
Caps, 3,700, at $.007	25.90
Seven machine men, 47 shifts, at $4.50	1,480.50
Two timbermen, 24 shifts, at $3.50	168.00
Timber	503.85
One superintendent, at $175 per month	140.00
Two foremen, 24 days, at $4.50	216.00
One hoistman, 47 shifts, at $4.50	211.00
One top man, 47 shifts, at $3	141.00
One mucker and skip tender, 47 shifts, at $3	141.00
One blacksmith, 24 shifts, at $4.50	108.00
One helper, 24 shifts, at $4	96.00
Power for hoist, 1,750 kilowatt hours, at $.013	22.75
Power for drills, 14,500 kilowatt hours, at $.005	72.50
Power for blower, 2,360 kilowatt hours, at $.005	11.80
Drills (depreciation)	12.50
Candles, 1,880, at $.0145	27.26
Cost for 1,110.5 feet	$4,747.04
Or cost per foot	$4.27

Lumber and Timber of Raises.—Stulls are placed in raises every 10 feet, with 10-foot ladders made of 2″ × 4″ lumber sides 14 inches apart, with nailed rungs 10 inches apart. There are needed 194 stulls 10 in. × 10 in., averaging 6 feet in length. As the length of the ladders will be 2,706 feet, there will be needed 1,624 strips 1 in. × 2 in., for rungs, each 18 inches long. Roughly, there will also be needed 388 pieces of 2″ × 6″ staging about 6½ feet long, and 24 chutes requiring 563 feet lumber.

Summary of Underground Costs (for Development and Time Expended)

	Days	
Additional equipment for development		$ 7,515.48
Shaft sinking	190	14,207.55
Cost two stations and cross-cuts to	14	2,292.36
Ore bins (underground), time included in above	14	626.04
Drifts and subdrifts	186	13,726.23
Raises	24	4,747.04
Total	414	$43,114.70

KATALLA, ALASKA, OIL FIELD

Written for Mines and Minerals, by W. T. Prosser

The oil fields of Katalla, Alaska, are attracting attention on the Pacific Coast, from the fact that the first large commercial shipment of this Katalla petroleum is promised shortly. Steel tanks to contain 37,000 barrels of oil have been shipped from Seattle to catch the flow of four producing wells. One tank from which vessels will load, will have a capacity of 30,000 barrels and will stand on Kanak Island, at the entrance to Controller Bay. Two other tanks, one of 2,000 and the other of 5,000 barrels, will be erected at the wells. A pipe line connecting the smaller tanks and the larger one, a distance of 8 miles, is almost in place, and will be ready before the reservoirs are completed.

Ten years ago, Dr. H. T. Burls, of London, England, made two trips to Alaska to collect data, and make observations and studies of the oil territory. Upon his report, which was supplemented by a very enthusiastic statement from Sir Boverton Redwood, his associate, work was begun drilling in the most favored locations. After oil was found, contending interests striving to gain possession of the most promising properties held up development for half a dozen years. It is only within the last year that new interests have gained possession, and are making ready to give the wells a chance to flow.

This new company found that the four old wells were choked with old iron and junk of all kinds, so that the oil could scarcely trickle out in drops. From one well two tons of debris were removed; then the flow carried away the top of the derrick, and settled down to a steady stream of about 720 barrels of petroleum a day.

Two of the other wells, in which the petroleum stood near the surface, were stimulated by the use of a pump, and after that flowed a very satisfactory stream. The four wells at a test made last December, in the presence of two officials of the Interior Department, yielded a total of 2,100 barrels a day.

Since that time they have been capped, awaiting the arrival and installation of equipment for taking care of the oil. The well with the greatest pressure creaks and groans at being held in restraint.

The oil belt is a strip of territory along the coast, averaging about four miles in width, and lying east of the Copper River's mouth. It is situated just to the southward of the Katalla coal measures, which so far as known are among the greatest in the world.

The Katalla oil is essentially a refining product, as it contains too high a percentage of valuable constituents to burn in its crude state. It is doubtful if there is another petroleum that can aggregate from 65 to 75 per cent. in gasoline, benzine, and kerosene. The Katalla oil averages about one-third gasoline and another third coal oil.

The following analysis of the oil is published by Prof. G. C. Martin, of the United States Geological Survey, in Bulletin 335, "The Geology and Mineral Resources of the Controller Bay Region, Alaska":

Distillation	Per Cent.
Below 150° C., naphtha	38.5
150 degrees to 285 degrees, illuminating petroleum	31.0
Above 285 degrees, lubricating petroleum	21.5
Residue, coke and loss	9.0

Katalla, Alaska

This same report shows that the Alaska oils have a percentage of gasoline 38.5, far higher than that of any other field on the continent. Pennsylvania is given as 16.5 per cent.; Ohio as 10; Colorado, 16; Mexico, 10; and Beaumont, Tex., as 2.5. The California oil is not included in this table, but analyses give it an extremely low proportion of gasoline—between the Texas and Mexican tests.

Gasoline-producing petroleum is not as plentiful as the demand, and almost the entire supply of the country comes from the Pennsylvania wells. Therefore the opening of the new Alaska wells will be appreciated by the consumers.

Doctor Burls and Sir Boverton Redwood expressed the opinion that it would not be necessary to go very deep in drilling for oil in the Katalla district, and their predictions were correct. The deepest producing well is 1,500 feet, and oil was struck in this well at a much higher level. Oil oozes out of the soil in so many places that the ground is saturated with it. Some of the seepages flow as much as two or three barrels a day. Part of the territory is low and marshy, penetrated by sloughs from the Gulf of Alaska, but equally as good indications and results have been obtained on the ground above.

Much operating will be done through the oil belt this year, and oil will win out against the Alaska coal in the race to get to the market.

RADIUM AND RADIOACTIVITY

By L. F. Miller, Professor of Physics, Colorado School of Mines

Different Rays and Their Effects—How to Recognize Radioactive Minerals

Radioactivity is a physical property exhibited only, so far as we know at present, by the elements uranium, thorium, radium, polonium, and actinium. They are called the radioactive elements. All excepting uranium, and thorium were unknown before the discovery of radioactivity. Like the discovery of the planet Uranus, radium, polonium, and actinium were first discovered through their effects instead of being observed directly.

Radioactivity is the property a substance has of sending forth a certain kind of invisible ray. There are a great many different kinds of rays projected by substances. Some emit a kind of ray which affects the nerves associated with the sense of touch, and are known as heat rays. Some send out rays which affect only the sense of sight; these are called light rays. Radioactive bodies send out a kind of rays which affect neither the sense of touch nor the sense of sight. They are invisible and can be detected only by certain physical effects which they produce. For instance, they may strike upon certain substances, as zinc sulphide or barium platinic cyanide and be converted into visible light rays. This is known as phosphorescence. They may be transformed into heat rays or produce certain chemical reactions, as when they fall upon a photographic plate, or finally they may pass into a non-conducting gas and make it conducting to electricity. This latter effect is called ionization. These rays also have the property of penetrating certain substances which ordinary light rays do not have. Some will pass through thick layers of iron and other metals with very little decrease in intensity.

Much was known about these rays, through the study of X rays, Crooke's tubes, and electric spark discharges, before the study of the radioactive elements. But they were never investigated as thoroughly as they have been since the discovery of the phenomena of radioactivity. Now it is found that these rays may be divided into three classes, according to the different physical properties which each exhibits. The three classes are designated by the three Greek letters, α alpha, β beta, and γ gamma. In general they all have penetrating power, but in varying degrees. They all have ionizing power, but the more penetration the rays have, the less effect they have as ionizers, that is, in making a gas conducting to electricity.

The alpha rays have very little penetration and are entirely absorbed by a sheet of aluminum one-tenth of a millimeter thick, or an ordinary thick sheet of paper, or even a few centimeters of air. They are the strongest ionizers, but they have very little photographic effect. If the rays are allowed to pass through a magnetic field they are bent from their path in such a way as to show that they are positively charged particles. The same effect is produced by an electric field.

The beta rays are more penetrating and require a five-millimeter thickness of aluminum to be completely absorbed. They have less ionizing power than the alpha rays, but the greatest photographic action. In an electric or magnetic field they are deflected much more readily than the alpha rays and in the opposite direction. This latter condition indicates that the beta rays are negatively charged particles.

The gamma rays are the most penetrating of all and will pass through 30 centimeters of iron or several centimeters of lead without much absorption. They have the least ionizing power and very little photographic action. They are not deflected in the least by either a magnetic or an electrical field. Therefore, they do not appear to carry an electric charge.

If r in Fig. 1 represents a small amount of radium at the bottom of a narrow tube, the three types of rays, alpha, beta, and gamma emitted by this radioactive substance when in a magnetic field may be illustrated as shown in this figure.

The beta rays are bent much more readily and opposite to the alpha rays, but the gamma rays remain undeflected.

Uranium, thorium, radium, and actinium emit all three classes of rays, but polonium gives off only the alpha rays. The relative proportions of rays given off are different for the different elements. For instance, the ionizing powers of thorium and uranium are about equal, so they must emit about the same proportion of alpha rays. But the photographic effect of thorium is much weaker than that of uranium. Therefore, the beta rays from the latter must be stronger than from the former. Poloninum shows practically no photographic effect or penetrating power at all, but its alpha rays produce strong ionization. The rays from actinium have about the same relative intensities as the rays from thorium, but they are all much stronger. Radium has the strongest rays of all, about 1,000,000 times more active than uranium.

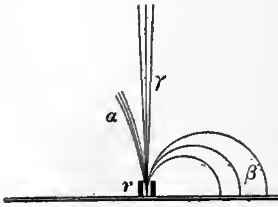


Fig. 1

It was found that by treating uranium by a certain chemical process a product could be dissolved off, which would give only the beta rays, while the product remaining behind showed now alpha rays but no beta rays. After a time this remaining product, which may be regarded as the original uranium, would begin to show beta rays again. Investigation revealed the fact that uranium is continually forming a product, by transformation, that may be removed chemically and gives only these beta rays, while the original uranium is the part that gives the alpha rays. After a time this original uranium forms more of this product by transformation and begins to show beta rays again. This transformation product, which can be separated from the original and which has been called uranium X, will finally exhaust all its beta rays and transform into another product of still different properties. Thorium and actinium have also shown the property of forming a similar transformation product which can be separated chemically as in the case of the uranium. A form of product similar to these has not been obtained from radium.

There is another form of emission given off by certain radioactive elements which is called an emanation. Investigation of all the previous products emitted by radioactive elements has shown that they are very minute particles, but these emanations which have been obtained show every indication of being a form of gas. Radium, thorium, and actinium give out these emanations, but, so far, uranium has not been found to give off any emanation. The strongest proof which we have that these emanations are of a gaseous nature is that it was not found possible to condense them on any object until they were cooled down to about $-150°$ C. by means of liquid air. This is one of the conditions for condensing a gas. These emanations will also diffuse through porous solids. These are not affected either by an electric or by a magnetic field, and although they seem thus not to consist of ions, yet they have the power of ionizing a gas and do not lose their power when they pass through cotton, wool, or when they bubble through solutions. They will also act upon a photographic plate.

It was also found possible to obtain transformation products from these emanations. That is, these emanations will transform into other emanations of still different properties. It is through the study of these various transformation products that we are brought upon the most startling discovery in the history of science. No credence has ever been put upon the alchemist's idea that one element can be transformed into another. But here in the phenomenon of radioactivity we seem to find such a process really taking place. For example, alpha particles obtained from uranium were collected, examined with a spectroscope and found to be helium, a substance which we have always regarded as an entirely different and separate element from uranium. From the

uranium X it was also found that, by transformation, radium was formed.

Uranium, originally, is not a composite substance as one might think from these results. In its primitive state before this transformation, it has properties which are entirely different and distinct from radium and helium. Their spectra are found to be entirely different when examined by a spectroscope, besides many other distinguishing properties. Polonium likewise originates from radium. The latest announcement is Boltwood's determination of another new element, ionium. Yet there are others still undetermined and some which exist for so short a time that they do not permit extensive investigation. Another evidence supporting this theory of a transformation process is that helium, radium, and polonium have never been found in nature except in association with uranium. Uranium and the other radioactive elements also exhibit another wonderful phenomenon in their continuous emission of energy in the form of heat, apparently without diminution and from the great difficulty is that radium is so rare that it is the most costly element on the market. What scientists are hoping to do is, by the study of this process of radioactivity in radium, to learn how a similar action may be brought about, regulated, and controlled in some cheaper form of substance. When this achievement has been accomplished we shall have at our disposal a greater store of energy than any form of supply known today. Radium has also won a very popular interest in medical practice. It was found to have a very powerful action upon animal tissues and was consequently hailed as a very potent factor in curing skin diseases and cancer. In light of all demand by the scientists and the medical profession, and because probably not more than half an ounce of radium has been produced in all the world, it is easy to understand why it should command such a high value.

Since radium has been found to be always a transformation product from uranium, it is natural to look for it only in those ores which contain uranium. The proportion of radium to the

Radioactive Minerals*

Name	Composition	Remarks
Uraninite	Oxides of uranium and lead	Occurs primary as a constituent of rocks and secondary in veins with metalliferous sulphides
Cleveite	Usually contains thorium and other rare earths and helium	
Broggerite		
Nivenite	Uranium 50—80%	
Pitchblende	Thorium 0—10%	
Gummite	$(Pb, Ca)\ U_3SiO_{12}\cdot 6H_2O$? Uranium 50—65%	An alteration product of uraninite. Formed by the action of percolating waters
Uranophane	$CaO\cdot 2UO_3\cdot 2SiO_2\cdot 6H_2O$	An alteration product of uraninite through gummite
Uranotil	Uranium 44—56%	
Carnotite	A vanadate of uranium and potassium Uranium 42—51%	Occurs as a secondary mineral impregnating a porous, sedimentary sandstone. Found in Colorado and Utah
Uranosphærite	$Bi_2O_3\cdot 2UO_3\cdot 3H_2O$ Uranium 41%	Alteration product of other uranium minerals
Torbernite	$CuO\cdot 2UO_3\cdot P_2O_5\cdot 8H_2O$	Alteration product of other uranium minerals
Cuprouranite	Uranium 44—51%	Alteration product of other uranium minerals
Autunite	$CaO\cdot 2UO_3\cdot P_2O_5\cdot 8H_2O$	Alteration product of other uranium minerals
Calciouranite	Uranium 45—51%	Alteration product of other uranium minerals
Uranocircite	$BaO\ 2UO_3\cdot P_2O_5\cdot 8H_2O$ Uranium 46%	Alteration product of other uranium minerals
Phosphuranylite	$3UO_3\cdot P_2O_5\cdot 6H_2O$ Uranium 58—64%	Alteration product of other uranium minerals
Zunerite	$CuO\cdot 2UO_3\cdot As_2O_5\cdot 8H_2O$ Uranium 46%	Alteration product of other uranium minerals
Uranospinite	$CaO\cdot 2UO_3\cdot As_2O_5\cdot 8H_2O$ Uranium 49%	Alteration product of other uranium minerals
Walpurgite	$5Bi_2O_3\cdot 3UO_3\cdot As_2O_5\cdot 12H_2O$ Uranium 16%	Alteration product of other uranium minerals
Thorogummite	$UO_3\cdot 3ThO_2\cdot 3SiO_2\cdot 6H_2O$? Uranium 41%	A variety of gummite
Thorite	$ThSiO_4$	A primary constituent of pegmatite dikes
Orangite	Uranium 1—10%	
Uranothorite	Thorium oxide 48—71%	
Thoriante	Oxide of thorium, uranium, the rare earths and lead. Contains a relatively large proportion of helium Uranium 9—10% Thorium oxide 73—77%	Occurs as a primary constituent of a pegmatite dike in Ceylon. Geological age probably Archean
Samarskite	Niobate and tantalate of rare earths Uranium 8—10%	Primary constituent of pegmatite dikes
Fergusonite	Metaniobate and tantalate of rare earths Uranium 1—6%	Primary constituent of pegmatite dikes
Euxenite	Niobate and titanate of rare earths Uranium 3—10%	Primary constituent of pegmatite dikes
Monazite	Phosphate of the rare earths, chiefly cerium Uranium .3—.4%	Primary constituent of pegmatite dikes

* Rutherford, Radioactivity, page 557.

no source except themselves. For example, it is found that one-half a grain of radium bromide gives out heat at the rate of two calories per hour, and in four years it yields 70,000 calories, and yet, in that time, no diminution in weight can be detected. It is difficult to estimate what will be the ultimate limit. All this energy arises from the disintegration of these minute particles which compose the atoms. Seventy thousand calories is enough heat to raise more than one pint of water from 0° to 100° C. If one-half a grain of coal is completely combusted, and this is the ultimate limit for this substance, it will give only 250 calories. Of course in the case of the coal it is an atomic reaction, while in the case of the radium it is a disintegration of the still smaller units of which an atom is now found to be composed. But it shows how immensely greater is the heat of combination of these small ions forming the atoms than the heat of combination of the atoms. Where one might conceive, then, how two or three grains of radium would furnish sufficient energy to run the Lusitania across the Atlantic, yet amount of uranium in an ore is so definite that Rutherford and Boltwood have determined the numerical relation. They found that there is .0000038 gram of radium to one gram of uranium. Thus one ton (2,000 pounds) would contain .0034 gram of radium for every percentage of uranium present. Or one ton of 60 per cent. uranium ore will contain two-tenths of a gram of radium, which is equivalent to .35 gram of pure radium bromide. Because of this extremely small portion of radium, it would require an ore with a considerable percentage of uranium to be worth treating for the extraction of the radium. This is why pitchblende has been used in preference to other minerals. As seen from the above list of various minerals containing uranium, pitchblende has the largest percentage. Of course some consideration must always be taken of the quantity of ore available. Thus carnotite, even though it has a smaller percentage of uranium, might be, in some cases, considered for extraction.

On the market, radium is found only as a salt—bromide,

chloride, or sulphate. What is generally spoken of as pure radium is merely the pure radium salt, usually the bromide. The best pitchblende is said to be obtained from Johangeorgenstadt, Saxony. If the radioactivity intensity of the sample from this locality is denoted by the number 8.3, the pitchblende from the Joachimsthal mine, Bohemia, has a value of about 7. However, the latter has the greater supply and so far has produced the greatest amount of radium. Pitchblende has also been found in Cornwall, but of a much lower grade, probably not much higher than 1.6. It has been made commercial, principally through the simpler and cheaper chemical methods of Sir William Ramsay for extracting the radium. Very good qualities of pitchblende, mostly in small quantities though, have also been obtained in the United States, in Connecticut, North Carolina, South Carolina, Texas, South Dakota and Colorado. In Colorado the pitchblende deposits have come from the Kirk, German, Belcher, Woods, and Alps mines, of Gilpin County. It is claimed that some of this pitchblende has run higher in its uranium content than any of the European products, and considerable has been shipped abroad. Several good carnotite deposits have also been found in Montrose, San Miguel, and Routt counties, Colo.

How to Recognize Pitchblende.—It occurs in fissure veins, in igneous rock like granite, and is valued for its uranium and radium contents. It is not found in sedimentary rocks like sandstone, although alteration products of it may be found therein. Such alteration products are colored yellow or greenish and, when in sufficient concentration and quantity, are valuable for their uranium contents. These deposits constitute the carnotite ore. The radium value in these is small, as a rule.

Pitchblende is found in vein matter in stringers varying from knifeblade thickness to several inches. It is "bunchy." The associated minerals in addition to vein matter are sulphides, such as zinc blende, galena, and iron pyrite. The latter usually predominates. Pitchblende is detected in these streaks by the following properties:

Color.—Pitch black, velvety black, brownish sometimes with a grayish or greenish cast. The streak on rough porcelain; i. e., the edge of a broken plate, is brownish black, olive green, or grayish.

Luster.—Dull, metallic, greasy, pitch like.

Fracture.—Conchoidal, that is, with smooth curved surfaces, to uneven. Brittle.

Hardness.—Less than quartz, about 5 or 6.

Specific Gravity.—Heavier than iron or steel, about 8 to 9.7.

Fig. 2

A hardness of about 5 or 6 is a little too hard to be scratched by a good knife blade and not quite hard enough to scratch glass.

Specific Gravity Test For Pitchblende.—A simple test for the specific gravity of the sample may be made if an analytical balance is at hand. Take a specimen about the size of an inch cube and weigh it. Then support over the pan of the balance a glass of water on a small bench, as shown in Fig. 2, so that the pan is still free to swing. The pan must not touch or rub against the bench or glass in any place. From the stirrup above the glass, support the sample with a fine thread so that it is totally immersed in the water. This immersion lightens the weight of the sample. Weigh the sample, subtract this weight from the first and divide the first weight by this remainder. This gives the specific gravity, which should be from 8 to 9.7.

Photographic Test For Pitchblende.—Since uranium and radium both emit pentrating rays which will affect a photographic plate, it is very easy to perform this test. Enclose tightly in a black heavy paper, so as to keep out the light, a photographic plate, noting first which is the film side of the plate. Lay the covered plate down with film side up and place upon it a key or some small flat metal object. Then lay the sample of ore upon the key. Place the whole in a drawer or away in the dark, because some daylight can leak slowly through

Fig. 3

the paper and affect the plate. After it has remained in this position for three days, develop the plate and find if an image of the metal object is obtained similar to that in Fig. 3. In that case the sample may be worthy of a laboratory test.

SOCIETY MEETINGS

The annual meeting of the Appalachian Engineering Association was held at Pottsville, Pa., June 2 and 3. A number of prominent engineers from the eastern middle and eastern southern states were present. Mr. Frank Hill, of Pottsville, Pa., who was on the entertainment committee, took charge of the visitors, some of whom were his friends and acquaintances when he was located at Roanoke, Va. At the morning session, papers were read by D. C. Weller, C. E., of Waynesboro, Pa., on "Hindrances to Municipal Improvements," and by E. B. Wilson, C. E., Scranton, Pa., on "Pillar and Room Mining," special reference being made to the proportioning of mine pillars. Dr. I. C. White, H. M. Payne, and E. A. Schubert took part in the discussion. Frank Hill, some time on the Pennsylvania Second Geological Survey, talked on the geology of the Schuylkill coal field and illustrated his talk by means of excellent maps that showed the folding of the coal strata. After lunch the visitors were shown through the Philadelphia & Reading Coal and Iron shops by Mr. Wood and his assistants and then taken by trolley to Tumbling Run to dinner.

At the business meeting in the evening the announcement was made that the officers elected for the following year were E. A. Schubert, Roanoke, Va., president; S. C. Weller, Waynesboro, Pa, vice-president; Henry M. Payne, Morgantown, W. Va., secretary; C. E. Krebs, Charleston, W. Va., treasurer. After the meeting there was a smoker at the Pottsville Club, where the visitors were entertained by Mr. Richards, General Manager of the Philadelphia & Reading Coal and Iron Co., Judge Coke, Reese Tasker, and other prominent men of Pottsville. Saturday an automobile trip was made to the Gilberton stripping, where Frank P. Weiser, division engineer, appeared; and next to the St. Nicholas colliery, where John H. Pollard, division superintendent met the party. From this mine a hurried trip was made to Ashland for lunch, and from that town to Pine Knot colliery, where Reese Tasker, General Superintendent Philadelphia & Reading Coal and Iron Co., met the party and accompanied them to Pottsville. The Appalachian Engineering Association consider that this meeting was the best they have had, and to those citizens of Pottsville who so lavishly entertained them they expressed their appreciation through Dr. I. C. White.

THE CRESSON MINE

Written for Mines and Minerals, by R. L. Herrick

Ranking fourth* in the list of the great producers of the Cripple Creek district stands the Cresson mine. Located on Raven Hill, about a mile north of the well-known Elkton mine, for lack of pretentious surface equipment, as shown in Fig. 1, the casual visitor might well be excused for classing the Cresson with the many neighboring properties of minor importance.

Method of Mining.
Hoisting Devices.
Boiler Pressure Regulator.
Electric Ore Cars.
Reports

Outside of the district the mine is little known, as it is owned by a close corporation of Chicago and Detroit men.

Not till 1906 did the mine pass the prospect stage and only since 1907, when its great ore bodies were found, has its production forced the recognition that it belongs in the first rank. The highest grade ore was mined on the upper levels, but the grade has decreased with depth, till now the ore shipped is considerably below the average for this district. During the year 1909 a total of 65,856 tons of ore was hoisted, of which 32,286 tons of sorted material was shipped.

The margin of profit in the gross production will probably compare with that at the three mines ranking above it in total production, and this in spite of a lower average grade of ore shipped. The average total cost of mining, figured on the basis of tons shipped, is said to be $5.02 per ton, which compares well with costs common in the district ranging from $7 to $12 per ton.† This cost of $5.02 per ton is obtained by adding the total expense chargeable to the mining operations, including management, and dividing by the total tons shipped.

Fig. 1. Surface Plant at Cresson Mine

Such a low cost in a district of rather uniformly high costs is to be explained first by the kind of ore bodies, and second by the economies effected in operation. The explanation of the latter is the main purpose of the following article.

The Cresson ore bodies are in granite invariably associated with and penetrated by a number of basic dikes. The ore may follow closely along the strike of the dike and be in contact with it, or in the well-defined sheeted zone closely following the dike. Again the ore may occur as an irregular body replacing the granite adjacent to the dike at a point where a series of fissures have evidently provided suitable physical conditions for metasomatic replacement.

The important ore bodies of the Cresson mine so far have been largely of the second type. One of the first ore bodies struck was the largest thus far exploited. It was encountered on the 600-foot level where its horizontal extent proved to be 60 ft. × 120 ft. Its vertical extent was from 80 feet above the 600-foot level down to the 800-foot level, a total distance of 280 feet. Another ore body on the 600-foot level had a shape like a long inverted radish with its main axis inclined about 70 degrees from the horizontal, its top resting just below the 600-foot level, and its tapering root extending to just above the 300-foot level. Its vertical extent was thus about 320 feet, and on each level its horizontal section was roughly oval. On the 600-foot level it was oval, 40 ft. × 70 ft.; on the 500-foot level, oval 45 ft. × 60 ft.; on the 400-foot level, roughly circular 60 feet in diameter; on the 300-foot level, oval 15 ft. × 40 ft.; and above this rounding out rapidly to a point. This particular ore body is well entitled to the name "chimney." Still another ore body which was being exploited at the time this article was written had a width varying from 12 to 30 feet on the 1,000-foot level and widening out to a fairly uniform width of about 30 feet in the stope above. Its length on the strike slightly exceeded 100 feet, and its vertical height had not been determined.

In each of these bodies the ore is calaverite, a telluride containing gold accompanied by minor quantities of pyrite. Owing to its degree of oxidation, however, the ore may be divided into three classes. The first class embraces ore mainly mined between the 200- and 400-foot levels, but in the case of one ore body it goes down to the 600-foot level. This ore is apparently a more or less complete replacement of the granite. It is soft and friable in spots, but mainly remarkable for its quantities of iron-stained clay which is probably the decomposition product of the granite feldspar. The second class of ore is somewhat similar to the first in that it is oxidized and iron stained, but carries only minor quantities of clay. This ore mainly goes down to the 800-foot level. The third-class ore, not necessarily of much lower grade than the other two classes, is mined mainly on the 900- and 1,000-foot levels. It is a hard, unaltered gray breccia of granite and basalt from the adjacent dike. Owing to the soft friable nature of the oxidized ore embraced in the first two classes mentioned and to the large size of the ore bodies, the cost of breaking ground is low. Indeed it has been nothing uncommon to break between 200 and 300 cars of ore in one round of shots. The method of mining followed is that of "filled stope" or "shrinkage stoping," and it was adopted to reduce the quantity of mine timber. This is accomplished by allowing a portion of the broken ore to accumulate in the stopes as a support for the walls and to afford a footing for the miners to work on. This method takes advantage of the fact that broken ore requires from 20 to 40 per cent. more space than does the solid ore; consequently, only enough of the ore broken each shift is drawn out to afford working room for the miners next the ore back.

In the Cresson mine about 100 cubic feet of solid rock occupies 125 cubic feet when broken, so that about 30 to 35 per cent. of the rock broken each shift is drawn off.

The method of exploiting an ore body is about as follows: If, as is usually the case, the ore body is in contact with a basalt dike, a drift is driven in ore along the wall constituted by the dike till the length of the deposit has been determined. One or two cross-cuts driven in ore to the other wall then give a good idea of the vagaries in width and also allow a more complete sampling of the ore. Upon the width and grade of ore largely depends the method adopted of supporting the weight of the stope to be mined above. If the ore is uniformly narrow, say 6 to 10 feet, and the grade rather low, a back of ore, whose thickness will equal that of the width of ore body, may be left to support the stope. If the width of ore body varies from about 10 to 30 feet or more, and assays an ounce or more of gold, both of which conditions usually exist together in the Cresson mine, the plan of leaving an ore back for support is quickly abandoned.

The ore on the floor level is then stoped out completely from wall to wall for a height of about 10 feet. The horizontal dimensions of the ore body now largely determine whether the stope shall be supported by stulls and lagging, or by square sets—the latter either open or filled. The size of most of the ore bodies permits the support of the stope on lagged stulls. If the width does not exceed the length of timber obtainable, say 16 feet,

* Professional Paper No. 54, United States Geological Survey. Probable order of ranking is: 1. The Portland; 2. The Golden Cycle; 3. The Vindicator; 4. The Cresson.

† The average cost at the Portland is given by Finlay in his "Cost of Mining" as $9.36 per ton shipped. This is probably a little high for present conditions.

the stulls are placed on a slant set in hitches cut on the foot-wall. If the width exceeds the length of timber, say 30 feet, two heavy timbers, from 14 to 16 inches in diameter, are spliced, that is cut half through and given a lap of about 2 feet. These spliced stulls are set horizontally, instead of inclined, and are supported from the level floor by posts 9 feet high, spaced from 6 to 8 feet apart. Such stulls have been found to provide ample strength and are both quicker placed and cheaper than square sets. In one case, however, a single run of filled square sets was used to support the excessive weight of an unusually large stope.

Having provided a floor for the broken ore, in which chutes are placed at about 15-foot intervals, the stope is carried up by the usual series of benches. At each end of the stope a manway is cribbed up and its top kept just above the level of the broken ore. The manway cribs are about 6 feet square and built up of 6-inch diameter round timber with the ends squared for about a foot in order to afford a spiking surface. The manway crib has a partition dividing it into a ladderway and chute. The latter is plank lined and ends in an ore pocket at the bottom. The ladderway contains 15-foot ladders, staggered, also the air pipes and a wooden box made of old slabs down which the dulled steel is dropped, thus protecting the ladders.

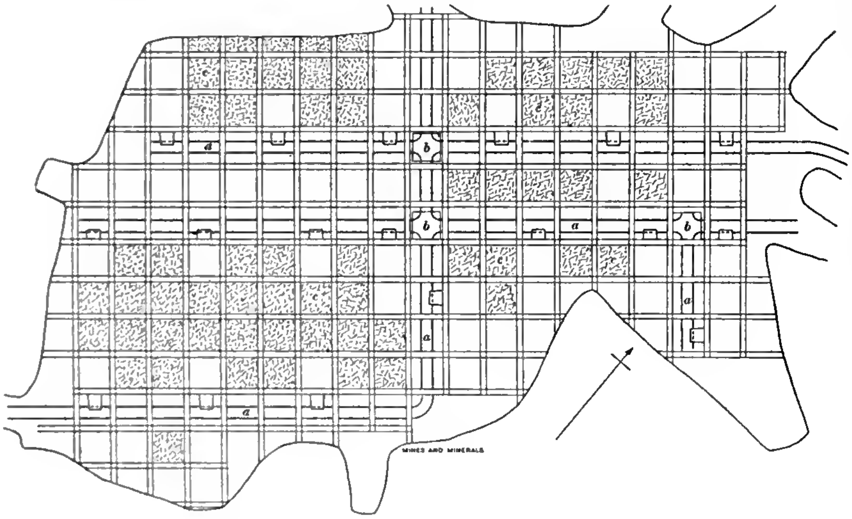


Fig. 2. Plan

After the stull lagging has been covered all over with a protecting layer of broken ore, a $4' \times 4'$ raise is started in the middle of the stope and driven to the level above. This raise is untimbered save for the few stulls necessary to support the ladders. It serves to ventilate the stope during its breaking upward and to bring down more air pipes. It is likewise used by the miners for descent into the stope when the climb up the manways from below has become more than 40 feet. With the exception of the drifts, which are usually driven by water Leyner drills, all stoping and raise driving is performed with the efficient little hammer drill of the Waugh type. By keeping the broken ore within working distance of the back, the series of benches are advanced wholly by the upper holes drilled. In a hard ground stope 100 feet long, six of these Waugh stope drills are usually worked to put in holes 4 feet deep.

The drill weighs but 62 pounds, and so can be rapidly moved about the stope by the drill runner. An average drill runner will drill from five to six 4-foot holes per day, although as high as 32 feet of hole per hour has been maintained in hard ground during periods of competition between drill manufacturers.

In the soft ground of the Cresson mine, as high as 10 holes per hour have been drilled In blasting the stope ore, 35 per cent. dynamite is used, about two sticks per hole being the common charge. In the soft ore of the Cresson mine two drill runners on one shift can commonly drill and shoot at least 200 tons of ore, an amount considerably in excess of the ordinary mining in the hard-ground stopes. An ore body is commonly exploited from two or more levels simultaneously. The uppermost stope is, of course, emptied of its accumulated ore first, a back varying in thickness from 8 to 30 feet being left between the top of the stope beneath and the floor level on which the ore is being drawn off. This thickness of overhead ore is varied to conform with the stope width and the strength of the ore. Eight feet suffices for hard ground of ordinary width, while 30 feet is none too much for the soft Cresson ground where the width equals or exceeds 30 feet. Having emptied the uppermost stope, the stulls may be left in or not, depending on the stope width.

In the large stopes of the Cresson mine, the timber is usually pulled out, the ore back filled with holes and shot down into the broken ore of the stope below, after which the next lower stope may be likewise emptied, and so on down. As long as the ore breaks fine enough to pass through the chutes, there is no trouble, but care has to be taken to avoid their choking with boulders.

After each shift shoots, a runner goes into each stope equipped with a light Hardscogg buzzer drill, searches out the boulders, and block-holes and shoots them into small pieces.

With this general description of a method which varies but little if any from the well-known practice, a few details concerning the exploiting of two stopes in an unusual manner may be of interest.

The first of these stopes has been previously stated to be about 60 ft. $\times$ 120 ft. in plan and 280 feet high. Having blocked it out on the 600-foot level, a single run or layer of square sets with posts 8 feet in the clear set on 5-foot centers was lagged over as usual. On account of the size of this ore body, four cribbed manways were carried up with this stope, each placed approximately in the middle of the four sides.

Fig. 2 shows in plan the arrangement of the 19 ore chutes (two more not shown were used) and the square sets. The car tracks *a* were arranged with turn sheets *b* placed at convenient intervals. After carrying up the stope some distance the weight of the broken ore became excessive, so that it was decided to fill enough of the sets to support the strain. In the illustration *c* represents those sets that were filled, the method being as follows: The groups of sets selected for filling were lagged inside after which the overhead lagging was shot down at enough points to allow the broken rock overhead to run in and completely fill the lagged space, which then acted as pillars to help support the strain. It will be noted that care was taken to leave a square set open either in front or behind each chute to allow ready access to it. This same scheme of work was followed on both the 700- and 800-foot levels below. After drawing the broken ore of the stope from the chutes and shoveling from the filled sets, the timbers were recovered by starting at one end and taking them back, retreating to the other end, taking up the tracks, etc. in the same order. It now remained to break down the back between the two stopes. Of course, this might have been drilled with holes from below and all shot down at once, but in the way actually followed the back was satisfactorily broken up with a considerable economy of labor and explosive over what the usual method would have involved. As fast as the top of the stope was brought up to within say 8 feet of the floor above, the back was immediately supported from the top of the broken ore by means of cribs built beneath the back and wedged against it. The cribs

were in some cases filled with broken ore. They were built similarly to the manway cribs and spaced on about 10-foot centers.

After having cribbed the entire back, its breaking was started as follows: A crib in the center of the back was selected as a starting point, and around this crib the drill holes were placed. The crib was then removed and the holes shot, breaking through a hole some 12 feet in diameter into the open stope above. After an interval of several days allowed for the rock thus loosened to work and drop, the back was again attacked adjacent to this center hole in a similar manner, drilling the holes in the protection of the crib and firing them after its removal. After the center hole had attained a diameter of about 30 feet, the bending of the whole back above resulted in giving it a slight saucer shape, bringing all the weight to bear on the cribs, and resulting in not only considerable incipient fracturing, but in rock strains which found relief in cracking when the blasting allowed further sagging. In this way the weight of the ore back was used to assist in its breaking down, thus reducing the amount of explosive. The primary object of the cribs, it is to be understood, was to protect the drill men while at work.

Having formed the central hole, the drill men were started at one end of the stope where a line of four cribs across the width of the stope supported the back. The back around the sides of each of these cribs was then drilled with holes, the cribs removed, and the back shot down the width of the stope. An interval of several days was then allowed for the stope to work, after which an adjacent strip of back was shot down in a similar manner. In this way the connection between the stopes was enlarged slowly, the cribs drawn line after line; in retreating toward the protected end of the stope, the men were always protected from working under a dangerously high roof, and the leaving of boulders too large to run through the chutes was avoided.

That the method is safe is sufficiently indicated by the fact that only one life was lost in the first 5 years' history of the mine.

With one exception, the ore deposits of the Cresson mine have been mined from two or more levels simultaneously. In the one exception, the ore body was stoped up about 320 feet from the 600-foot level in one great stope. This was the ore body whose shape was likened to that of an inverted radish, and whose western wall was formed by a basalt dike striking about N 30° E and dipping southeast about 70 degrees from the horizontal. Having been encountered on the 600-foot level, where the ore tapered down to a comparatively narrow streak, having its main axis along the basalt which formed a footwall, a drift was run along the basalt. Having left a back of ore above the drift about 10 feet thick pierced with chutes at 15-foot intervals, the stope was started in the customary manner. At the level of the top of the back, the ore widened out to 40 feet, but as the back was mainly country rock it was not taken out and the stope supported by timbers as in other cases. On approaching the 500-foot level, the softness of the ore and its excessive width would have necessitated the leaving of a 30-foot back below it for support. Instead of leaving this back and supporting the stope above on square sets, it was decided to carry the stope on up as before so that eventually it was carried some 10 to 15 feet above the 300 level. On the levels above the 600, however, the mine workings were extended to approach the ore body from the foot-wall side. Leaving the basalt as a barrier between the stope and the drift in the granite, chutes were driven through into the stope. From these chutes on the various levels the broken ore was drawn from time to time. Eventually, after the entire stope was completed, the ore was drawn simultaneously through the chutes on the various levels as well as from those on the 600. In this way the entire ore body was mined without the use of a stick of timber for supports, and the only timber used was that employed in the construction of manway cribs, and ore chutes.

Development Work.—The mine drifts are driven by contract. The No. 7 water Leyner drill, weighing 120 pounds, known as "the one man" drill, is used in this work. The contract usually stipulates the payment of $3 per running foot of advancement, on which basis the company furnishes the drill, air, and attends to repairs, while the drill runner furnishes his own powder and pays his trammer

Sinking is likewise done by contract, with conditions similar to those in drifting. The shaft timbers are placed by timbermen on company time. The lowest mine level is the 1,100-foot, from which sinking to the 1,200-foot is in progress. The shaft is vertical, having two compartments, each 4 ft. × 4 ft. in the clear. In the sinking work, the hoisting is done in the

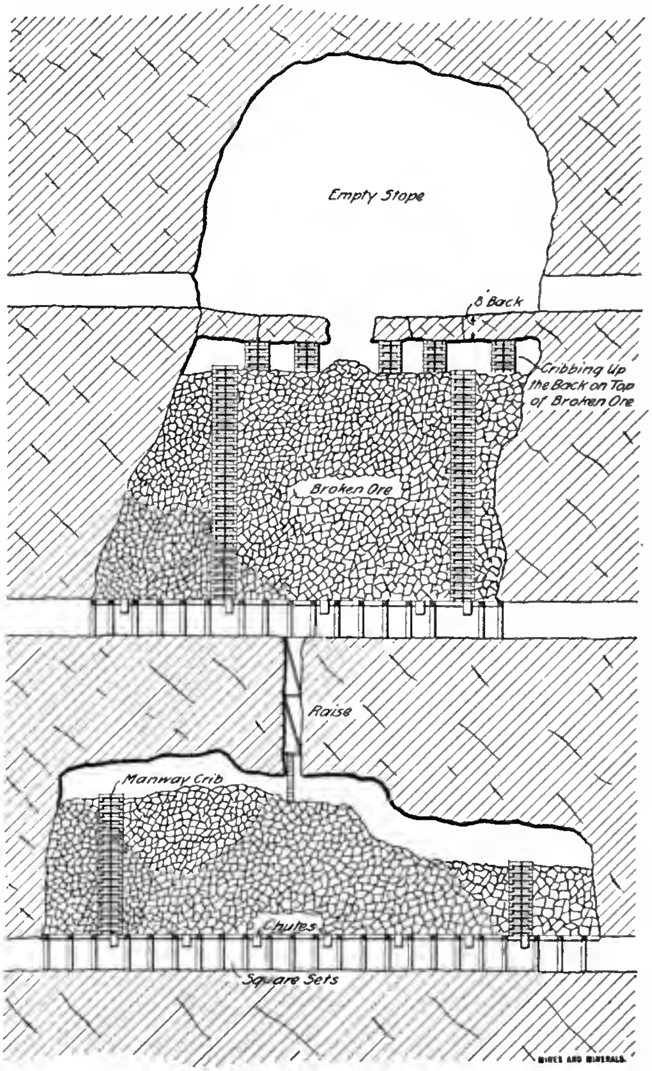


FIG. 3

manway compartment, the skipway being bulkheaded below the 1,000-foot for the protection of the men.

The sinking skip is run by a small compressed air engine located in the 1,000-foot station. On this level two swinging doors, one 4 feet above the other, protect the men below. By means of the well-known ball-and-chain device, these doors are opened as the skip comes up and closed behind it, before dumping, by means of two compressed-air pistons suitably connected to the doors by wire ropes. The air pipes connecting with these pistons run to within easy reach of the hoistman in the 1,000-foot station, who controls the piston movements by means of the usual valves. After dumping the skip into a car placed for the purpose, a movement of the valves causes the pistons to raise the shaft doors, after which the skip descends and the

pistons allow the doors to close after it. By means of these pistons, operated by the hoistman, the necessity of a top man at this point is eliminated, thus effecting a daily saving of his wages.

Drifting on the 600 level along a basalt dike, the emanation of carbon dioxide CO_2 gas from the rock interstices caused considerable trouble. Near the breast of the drift where the mine ventilation was naturally weak, an ingenious device was installed in order to prevent the CO_2 gas from entering this end portion of the drift, and measures to increase the atmospheric pressure at this point were taken as follows:

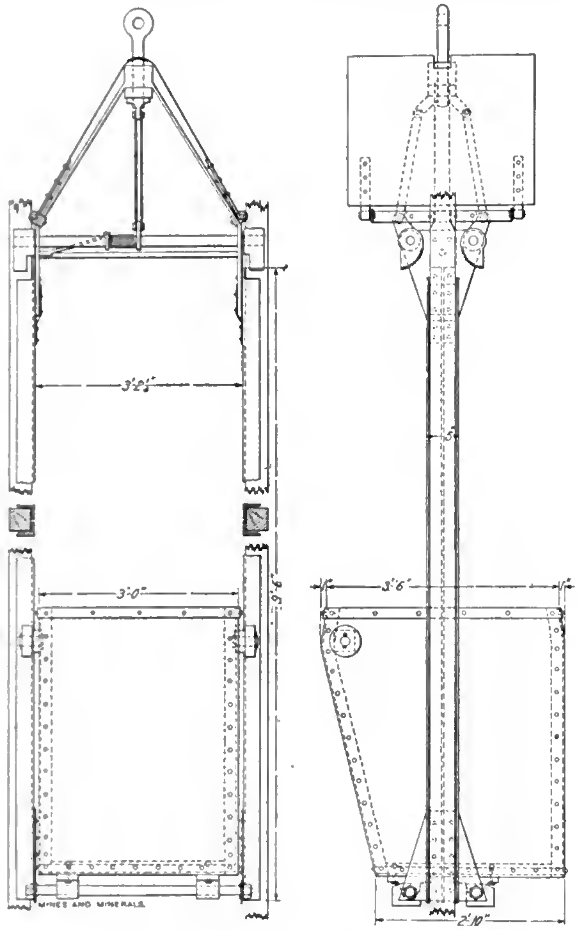


Fig. 4. Automatic Dumping Skip

A heavy air-tight door was placed across the drift, its frame consisting of 8″×8″ timbers between which and the rough walls the spaces were closed with cement. On one side, the door frame was recessed to admit of a short piece of 5-inch diameter pipe, so as to establish a connection between the air in front and behind the door. The 1-inch diameter compressed-air pipe was then led to this 5-inch orifice, inside of which a spider supported a ⅛-inch diameter nozzle connected with the air pipe. The pressure in the air main at this point was about 90 pounds. A valve in the main placed just outside the door allowed the operation of the device when desired.

When turned on, the compressed air rushing at a high velocity from the nozzle operated on the well-known injector principle to draw the mine air through the 5-inch pipe and increase the atmospheric pressure behind the door. This was found to effectively prevent the emanation of CO_2 gas into the drift.

A complete system of mine telephones is installed in the mine and is found to save considerable time previously consumed in traveling to and from the surface. A Stromberg-Carlson phone is placed at each mine station and the line connected on the surface to both the office and the machine shop. By this means the shift bosses can communicate with the office from underground, while the drill runners can telephone the machinist for any needed parts or repairs. The saving of time in eliminating many useless trips through the shaft has already effected a cash saving equivalent to a large part of the first cost.

Hoisting is done in the automatically dumping skip illustrated in Fig. 4. It has a capacity of 32 cubic feet and is made at the company machine shop. While there are no particularly new features about it, it may be worth while to call attention to its false bottom. Between the true and false bottom is placed a wooden plank 1½ inches thick, upon which rests the false bottom of ½-inch boiler plate. The wood absorbs much of the concussion on the metal, thus increasing its life, while the false bottom may be quickly replaced when worn out, thus greatly increasing the life of the skip body.

The surface hoist is a 14 in.×30 in. double drum—round rope—of the Ottumwa Iron Works type. It is equipped with an automatic stop to prevent overwinding. The skip is counterbalanced by the counterweight, shown in Fig. 5, which is operated in the manway compartment. Aside from the well-known advantages of operating a counterweight to effect balanced hoisting, this particular weight combines the advantages of requiring a small portion of the space in the manway, and safety in case of a break in the hoist cable. The skip is run at an average speed of 1,700 feet per minute and as this requires close to 200 horsepower, while the boiler furnishing the steam is only of 100-horsepower capacity, it may be of interest to digress long enough to explain this point.

This boiler is one particularly well adapted for steam generation when burning the lignite coals commonly available

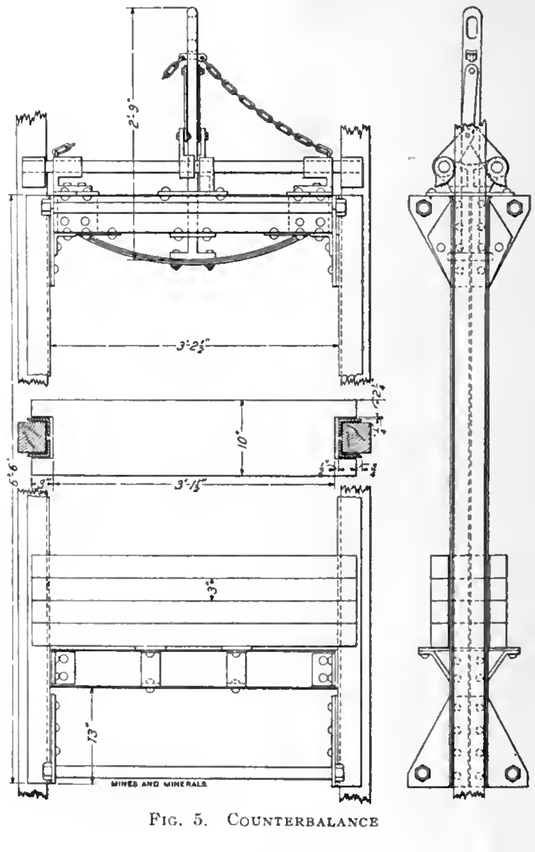


Fig. 5. Counterbalance

in the West at a cost of about half that of bituminous steaming coals. Briefly stated, its construction allows the use of a thick bed of coal. Steam introduced beneath the flat grate serves the double purpose of eliminating clinkers and combining with the volatile hydrocarbons driven off at a low temperature to form a water gas which immediately burns beneath the boiler

tubes. The introduction of preheated air at a point close to where the gases form and burn adds considerably to the efficiency of the boiler.

From what has been said concerning the requirements of the Cresson hoist, it will be understood that during hoisting the steam pressure under ordinary conditions varies widely, say from about 98 to 76 pounds, thus subjecting the boiler to considerable strain. In order to obtain a more uniform pressure the regulating device shown in Fig. 6 was made and installed. Its purpose is to increase the steaming capacity of the boiler during hoisting by starting a forced draft under the grate which lasts only while the steam pressure is below 98 pounds. Whenever this pressure is attained, the forced draft is automatically shut off by the device. The draft is induced by a steam jet in an 8-inch air pipe, the jet acting as an injector to force the air under the grate.

Referring to Fig. 6, steam pipe *a* leads from the valve *b* to the jet in the blower pipes. The valve is kept closed so long as the boiler pressure approximates 98 pounds, but the instant the pressure falls, the valve automatically opens to allow the boiler steam to induce the forced draft. Besides this connection with the boiler there is a pipe *c* which admits steam to the piston *d*, which is attached to the lever *e*, from whose end are suspended the weights *f*. The latter are equivalent to a pressure of 98 pounds per square inch. When the steam pressure exceeds 98 pounds the piston *d* keeps open the needle valve *g* and admits steam to the cylinder of piston *h* by the pipe connection shown. Here the steam pressure overcomes that of spring *i* and keeps the valve firmly against its seat. The instant the pressure decreases below 98 pounds, the weight *f* closes the needle valve, allowing spring *i* to open the valve *b*, thus admitting steam to start the forced draft.

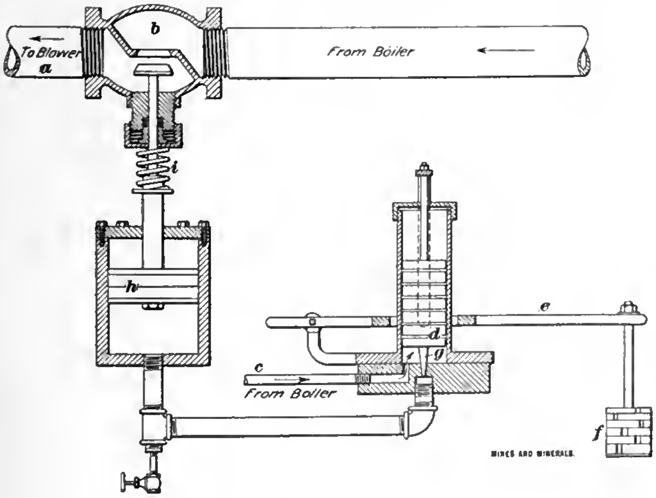


FIG. 6. BOILER PRESSURE REGULATOR

This simple regulator has been found very efficient in maintaining a fairly steady boiler pressure as shown by a Bristol recording gauge attached to the boiler.

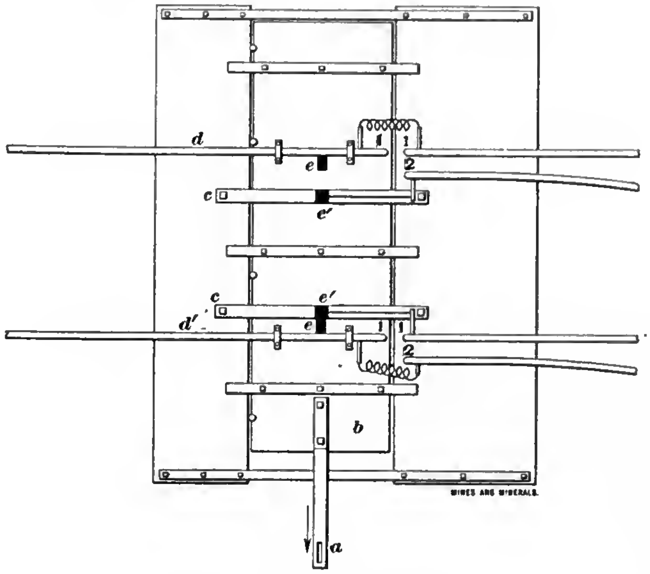


FIG. 8. TROLLEY-WIRE SWITCH

The ore when it arrives at the surface is automatically dumped from the skip into the trolley car, shown in Fig. 7. The car body is constructed of wood, lined with $\frac{3}{16}$-inch sheet steel. No advantage is claimed for it over ordinary cars except that it can be quickly and cheaply constructed by the mine carpenter.

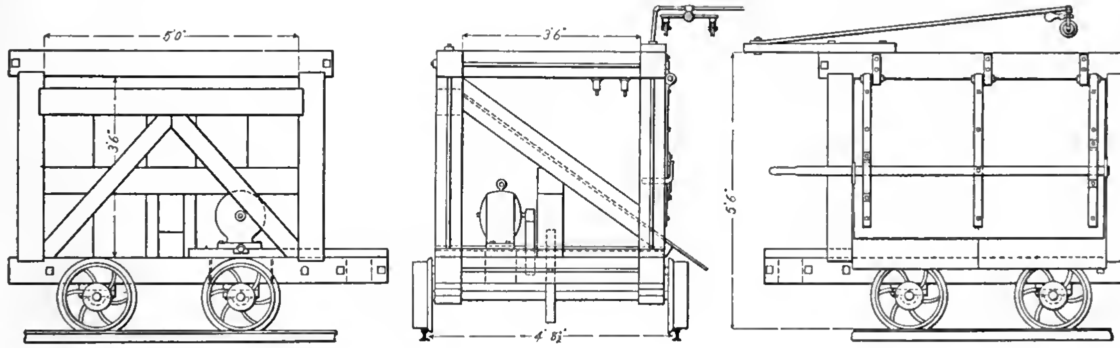


FIG. 7. TROLLEY ORE CAR

Manager Roelofs stated that one of these cars handled over 60,000 tons of rock in one year's time. The 4-horsepower motor on the car is operated by a car man who stands by the controller placed on the rear platform. Three-phase current is delivered to the trolley wires at 110 volts. There are two overhead wires while the rail serves as the third conductor.

In Fig. 1, showing the outlay of the surface plant, it will be noted that one track seen at the left leads from the head-frame to bins above the end of the sorting house. The sticky, clayey, first-class ores are dumped into these bins, thus being shunted around the sorting plant. The other ores are dumped into the bins seen in the central foreground, which feed the sorting belts. This separation of the ores requires a switch in the track and a corresponding arrangement for the trolley wires.

By means of an ingenious device, shown in Fig. 8, the trolley-wire connections are shifted properly and automatically by the same movement of the lever that sets the track rails.

This device is shown in plan, inverted from its true position. The board *b* is given a reciprocating motion by means of a vertical lever attached to the strap *a*, this lever being thrown by the movement of the track lever. As the board *b* slides

to and fro it is upheld in place by straps *c*. In the illustration the two trolley wires *d*, *d'* are shown in the proper position for sending the car to the sorting house bins. When the track lever is thrown for the other set of bins, the board *b* slides in direction shown by the arrow bringing the ends *1* of the wires opposite the ends *2*. This movement by means of the wiring connections shown and the making and breaking of contact by the springs *e* and brass strips *e'*, throws the current into the wires *2*, cutting it off from wires *1*.

No attempt is made to wash and closely sort the sticky clay ore. It is fed from the bins down a picking chute where the easily apparent waste such as basalt is picked out while the remainder falls into the tramway bins below.

The other two classes of ore are separately washed and sorted. The fine ores washed from the ore are settled in a series of tanks, steam-dried in pans, and shipped. The value of the fine material thus recovered pays the entire expense of the sorting house. The clear water flowing from the last settling tank is pumped back to the trommels, the quantity of water allowed to flow to waste being small.

The sorted ore falls into a series of bins for each class of ore. From these bins the ore is drawn into the aerial tramway buckets and sent over the line to the ore house a half mile below. The total cost of tramway transportation is but 15 cents per ton as against 65 cents for team haulage.

A very complete system of daily reports has been inaugurated at the Cresson mine and found efficacious both in increasing the efficiency of the working force and in reducing running expenses, repairs, etc. There is one form of report for the drill runners, its object being to fix the responsibility for every breakage and so ascertain whether it is due to carelessness and inefficiency or to faulty construction.

If the drill is in bad order on starting work the drillman waits till the shift boss comes into his place before doing a stroke, when he reports the damage. Otherwise he is blamed for it. In this way the runner on the previous shift is made responsible and is warned to report the damage next time and so enable its repair before the next shift goes on. This system has reduced the loss of working time by providing machinery in good order.

In like manner every trammer makes a daily report. While this at first led to considerable "mental aberration," to put it mildly, when it was learned how easily the report was checked at the end of the day, the trammers quickly got into the George Washington class.

The use of the ammunition order blank quickly has a tendency to cut down unnecessary consumption of explosives and supplies.

It is common experience that when other excuses give out the miner has a kick coming against the blacksmith department for its alleged failure to make repairs or sharpen steel.

By having the blacksmith's report it is an easy matter to determine who is the Ananias and furnish the "hike" to the proper party.

Finally the shift bosses are required to fill out a report which enables the superintendent to place his finger on that part of the mine where operations have dragged, as each working place is numbered. Thus, D 706 refers to the 6th drift on the 700 level. If this place has "tapered off" apparently without good reason, the shift boss in charge is requested to furnish a "what for."

For the guidance of those in charge of the oncoming shift, a book is kept which is always consulted before the final arrangements for the shift are made. In this book the foreman notes the important events of his shift before going off duty.

If a certain piece of roof needs watching or timbering, a stated ladder is broken, a shot missed fire in a given drift, etc., such facts are noted and warning given to look out and attend to them.

Such data are handed down to the shift bosses, and from them to the men assigned to the places mentioned. Those not anxious for sick benefits can then avoid unnecessary dangers.

The summary of the day's work is made out on another form. In case the superintendent or manager is away from the mine for a day or two these cards refresh his memory upon his return.

That the Cresson costs are considerably below the average for the district is due principally to the property having large bodies of easily and cheaply broken ore, and to keeping the working force at a minimum by means of proper mechanical appliances, which in the end reduce expenses.

By keeping close tabs on the performance of each individual on the company pay roll, his efficiency is encouraged and his expense kept at a minimum. Finally, by managing to get along with a modest though efficient surface plant, the cost of a more elaborate equipment is saved for dividends. The mechanical appliances described in this article are the results of the combined ingenuity of Mining Engineer Edward G. Morton and Master Mechanic George L. Hatfield, to whom the writer is indebted for his data and illustrations. For the remaining data the writer is indebted to the courtesy of Manager Richard Roelofs.

CONCRETE SHAFTS IN IRON ORE MINES

In a number of places in the Lake Superior iron ore district it is difficult to sink shafts owing to the amount of quicksand which must be penetrated to reach the ledge holding the ore body.

The Cliff shaft of the Cleveland-Cliffs Co., in Ishpeming, had much to contend with, but it did not offer so many obstacles to downward progress as some other locations.

The original shaft at the Negaunee mine was sunk where it met quicksand which was very wet and boiled up with considerable force as pressures were placed upon it. To overcome the various obstacles encountered in sinking required a long time and the expenditure of a large sum of money.

The original shaft at the Aragon mine in the Menominee Range was a difficult proposition to sink. It was, however, finally gotten down by perseverance and much money.

The big Chapin mine at Iron Mountain had troublesome ground, and at one point it was necessary to freeze the quicksand. This was done by using a foreign patent which was exceedingly expensive, as well as the operation.

The shaft at the Maas mine, Negaunee, was one of the most stubborn in opposing downward progress. It required six or seven years for the Cleveland-Cliffs Co. to get this shaft down 160 feet and in hoisting order. The condition of the shaft was unsatisfactory after all owing to its not being in line; therefore the mine was shut down and a contract made for a concrete shaft from the surface to the ledge. The concrete work will be forced down from the surface by the weight of the material used in its construction. The outer portion of the shaft will be round, while the interior will be 14 ft. 10 in. × 10 ft. 10 in., so that the line drawn through the cement work at either end of the shaft's dimensions will find at the shaft's bottom concrete having a thickness of 7 feet 11 inches. This thickness will be carried a considerable distance upwards from the bottom and then be gradually lessened until nearer the surface it will be tapered to 2 feet 6 inches.

The concrete work will be reinforced with steel, thus making a substantial shaft lining and the concrete will be carried well into the ledge to make a strong bottom joint. When finished, the concrete portion of the shaft will be 200 feet long from top to bottom and will be one of the largest concrete jobs in the Lake Superior country; the cost will be heavy owing to the nature of the work to be done and the conditions. The same company is concreting its Negaunee shaft, located a short distance from the Maas mine. The Cleveland-Cliffs Co. have now a concrete shaft at the North Lake mine, a short distance west of Ishpeming, and several in the Swanzey field, four are being put down on the Cuyuna veins, Minnesota.

EXAMINATION OF MINE OFFICIALS

Written for Mines and Minerals

An Examination System Inaugurated by a Mining Company Though Not Required by Law

In the state of Washington certificated mine officials are not required by law, but the Northwestern Improvement Co., with headquarters at Tacoma, Wash., and mines in Washington and Montana, has recently set in motion a system of examining their lesser mine officials. So far, this has been confined to the Washington field and to the examination offire bosses and can didates for that position, but the intention is to enlarge its scope sufficiently to include foremen and assistant foremen in the near future.

Thus far four examinations have been held, two at the Ravensdale mines and two at the Roslyn mines. The first examinations in each field were for incumbents of the positions at that time, the second were for the purpose of filling vacancies that occurred and providing an eligible list. In all 28 men have taken the examination and 21 of these have been awarded a certificate setting forth their qualifications to act as firebosses for the company.

The examinations have been entirely practical in their nature and no deep mathematical or scientific knowledge has been demanded of the candidates. Chief stress has been laid on practical experience and, next to that, a clear conception of the duties required in filling the position. The simple, yet practical nature of the questions asked is shown in the papers prepared for a recent examination. After satisfactorily passing the examination, an attractive certificate, as shown in Fig. 1, is given the successful candidate.

The following are typical of the questions asked:

Questions for Fire Boss Examination

Note.—Read all questions before answering any. Write your name, and the number of the mine in which you work, on the back of the last sheet.

Ques. 1. What is your name? Your age? How long have you worked in coal mines, and where? Were the mines gaseous or non-gaseous? Flat or pitching? In what capacity in each field?

Ques. 2. What are the duties of a fire boss?

Ques. 3. Name, and tell the characteristics of three gases commonly found in coal mines? What is the cause of each? How would you determine if any of them were present? Where would you look for each?

Ques. 4. What would be your procedure if, on your morning round, you found a room, up 60 feet on a 15-degree pitch, with no cross-cuts, full of gas back from the face for a distance of 30 feet?

Ques. 5. Explain how you would clean and test your safety lamp before taking it into the mine to test for gas?

Ques. 6. The dimensions of an airway are 8 feet wide and 5 feet high. At a velocity of 200 feet per minute, how much air would pass a given point in an hour?

Ques. 7. What are the principal precautions that may be taken to avoid explosions of gas and fires in mines?

Ques. 8. What regulations do the State Mine Laws make regarding the use of safety lamps in mines?

Ques. 9. Give the substance of the Company Rules regarding the use of powder in the mines.

Ques. 10. Explain the theory of natural ventilation, and the effect of atmospheric changes, and changes in temperature, on ventilation.

The following statement of the duties of a fire boss would be considered a satisfactory answer to Ques. 2:

"The duties of a fire boss in the mines of the Northwestern Improvement Co., in the state of Washington, are as follows:

"To make the examination of all working places before the time at which the mine begins to work in the morning as required by the state laws and the company rules. In detail, his duties on this examination are, in the order named, to examine and adjust his safety lamp and to make his entire examination with a safety lamp from the time of entering the mines; to ascertain that the ventilation apparatus of the mine is in working order and, approximately, that the usual current of air is in circulation; to thoroughly examine each working place, going in with the air, and beginning at the outside place on each split; to note all falls of rock and places where the roof is in dangerous condition through lack of timber or otherwise; to test for gas at the face of each room; to mark the date on the roof at the face of each room as a record of his examination; to test for gas in each face and on top of falls or in cavities where gas is liable to accumulate; to remove any accumulation of gas if feasible, but never by brushing except the quantity be very small; and in case the gas is not removed, he must fence off that particular opening and put up a danger board to warn any one from entering the place; to also put up a danger signal warning the miners against entering a place containing any other dangerous conditions where he deems such warning necessary; to inspect the return of a particularly gaseous mine or split for evidence of a sufficient percentage of gas to produce a cap on the flame of his safety lamp; to travel the manway and escapement shafts and note their condition for safe and rapid exit; and, finally, to come outside before the miners enter, and mark any dangerous places on a board at the mouth of the mine, and also to make a report of same, and of any gas found, in a book kept for that purpose at the office, and to the foreman. Furthermore, it is the duty of the fire boss to act as an assistant to the foreman, having particular charge of all things pertaining to ventilation, bratticing, timbering, etc., and to perform such other duties as the foreman may assign to him.


Northwestern Improvement Co.
Certificate of Competency in Mining
This is to Certify, That Mr.____ ____________
on____________191__, at ____________, Washington, successfully passed an examination for competency as ____________, as given under the rules of the Company, and is hereby pronounced qualified to act in that capacity in any of the mines of the Company in the State of Washington.
____________191 (Signed) ____________
General Manager

Fig. 1. Facsimile of Certificate

"In the absence, or at the request, of the foreman he must make the weekly air measurements and inspection of the accessible old workings of the mine as required by law."

The increase in interest and efficiency induced by these examinations has so far been very gratifying, and it is expected that like good results will be attained when the examination is extended to include foremen.

Mr. C. R. Claghorn, well known in Pennsylvania, is general manager of the company, and Mr. J. F. Menzies, of Roslyn, is general superintendent for the state of Washington. The work of giving the examination and grading the papers is attended to by the company mine inspector, Mr. J. B. Warriner.

COAL NEAR TAMPICO, MEX.

It is claimed that a good vein of coal has bee 50 miles from Tampico, Mex., but as the locality i transportation, it will probably be some time is developed.

Mines and Minerals

ESTABLISHED 1881

PUBLISHED MONTHLY AT SCRANTON, PA.

Western Editorial Office: 728 Equitable Building, Denver, Colo.

Chicago Office: 309-311 Michigan Ave.

Entered at the Post Office at Scranton, Pa., as second-class matter

Address all business communications to Mines and Minerals, Scranton, Pa

CABLE ADDRESS, "RETSOF, SCRANTON"

RUFUS J. FOSTER.......................Manager
EUGENE B. WILSON, Scranton, Pa.......................Editor
GEORGE F. DUCK, E. M., Denver, Colo.......................Western Editor
P. G. MOORE.......................Circulation Manager
LONDON AGENTS: KEGAN PAUL, TRENCH, TRUBNER & CO., Ltd., Dryden House, 43 Gerrard St., Soho, LONDON, W., ENGLAND.
AUSTRALASIAN AGENTS: GORDON & GOTCH, 123 Pitt Street, SIDNEY.
SOUTH AFRICAN AGENT: E. HERBERT BYSSHE, CAPE TOWN, Mansion House; JOHANNESBURG, Trust Building; DURBAN, Natal Bank Chambers; BLOEMFONTEIN, Florence Buildings.

SUBSCRIPTION RATES

Single Copies.......................25 cents
Yearly Subscriptions, including postage:
To United States, Mexico, Cuba, Porto Rico, Hawaii, and Philippines. $2.00
To Canada....................... 2.50
To all other countries....................... 3.00

Remittances should be made by P. O. Order, Express Money Order, Bank Check or Draft, or Registered Letter.

The date after your name on the wrapper is that of the month to which your subscription has been paid. If you do not want to continue your subscription after that time, kindly notify us.

Notice to discontinue or change of address should be sent to the Scranton office

When notifying us of a change of address always give your FORMER as well as your present address.

TABLE OF CONTENTS

Webb City Lead Smeltery, by L. L. Wittich....................... 709
Mining on the Gogebic Range, by Percival S. Williams....................... 712
Lake Superior Iron Ore Notes....................... 714
Specific Gravity Estimation of Pulp, by B. F. Hyder....................... 715
Stoping Drill Steels....................... 717
Lake Superior Copper Notes....................... 717
Copper Converter Flames, by Donald M. Levy....................... 718
Mammoth Cave of Kentucky, by James H. Gardner....................... 720
Dessication of Furnace Air....................... 723
Personals....................... 726
Lehigh University Graduates....................... 726
Developing Poverty Gulch Claims, by Chas. W. Henderson....................... 727
Katalla, Alaska, Oil Fields, by W. T. Prosser....................... 731
Radium and Radioactivity, by L. F. Miller....................... 732
Society Meetings....................... 734
The Cresson Mine, by R. L. Herrick....................... 735
Concrete Shafts in Iron Ore Mines....................... 740
Examination of Mine Officials....................... 741
Editorial—
Pennsylvania's Bituminous Mine Law—"Safety the First Consideration"—Federal Aid for Mining Colleges....................... 743
Book Review....................... 744
Obituary—
Nathaniel W. Lord—G. B. Bond—P. M. Boyle....................... 744
Correspondence—
Storing Coal—The Problem of Education—Mining in San Domingo—Coal Dust Explosions....................... 745
Trade Notices....................... 747
New Legislation Will Affect Mining Shares....................... 748
Mine Support Tests....................... 749
Mining Wrinkles—
Inspection Table at Cokedale Mine—Rope Throw at Radiant Mine—Rock Handling Arrangement at Globe Mine—Stopping Blocks and Derailing Switches....................... 753
American Institute of Mining Engineers....................... 754
Electricity in Mining, by Walter C. Wagner....................... 756
A New Breathing Apparatus....................... 759
Method of Installing Fans, by Thos. W. Fitch, Jr., and Jay R. McColl....................... 760
New Illinois Mining Law....................... 761
Alabama Inspectors' Warning Card....................... 763
The Testing of Miners' Oil, by C. E. Scott....................... 764
Firing Ammonium Nitrate Explosives....................... 767
Compensating Grades, by R. D. N. Hall....................... 768
Examination for Assayers in British Columbia....................... 769
Examination Questions—
Anthracite Mine Foreman's, April, 1911....................... 770
Catalogs Received....................... 772
New Inventions....................... 772
...ies—
Solid Nitroglycerine, 711—Vanadium in Cast Iron, 716—Prevention of Scale in Boilers, 719—Cost of Cyanide Process, 726—Coal in Mexico, 741—An Important Oil Land Decision, 751—C. S. Anniversary Banquet, 754—India's Mineral Resources, ?—French Coal Market, 766—Power to Ventilate a Mine....... 771

the
stated l.
such facts
to them.
Such data

PENNSYLVANIA'S BITUMINOUS MINE LAW

The new Pennsylvania Bituminous Mine Law, recently made operative by Governor Tener's signature, is a decided improvement over the first bill drafted. There are a number of matters that will be criticized when sufficient time has elapsed for both miners and operators to digest the law.

As a preliminary criticism, the verboseness of the lawyer and the vernacular of the mine inspector are too prominent. The result of this profuse use of words is that a number of important passages in the law are made ambiguous. There was a time when the creation of mine inspectors was opposed with all the influence operators could command, because they believed their rights would be infringed. At present, however, they favor the increase in number and the increased power given to inspectors. This change in policy shows that the operators are using their utmost endeavors to conserve the lives of their men and property irrespective of any mine law and that they welcome suggestions that will benefit their business. The increased power given inspectors removes a great responsibility from the operators' shoulders and places it on the mine inspectors, and it also refutes the statement so often made by sensational writers that mine officials care more for money than they do for the lives of their men.

The writer, when a mine official, was often compelled to do things that, had the property been his own, he would not have done. This was because he was an agent for others, and had he not done as he did he would have been an unjust steward. If the critic will examine into the affairs of any corporation he will find that there are conditions that can be improved when the lion and lamb lie down together, and not before. Looking at the new law from the miners' side, we find that he is now compelled to obey the rules laid down for his guidance by the state and by the company. In addition to that, he must protect himself; that is, not depend on the management entirely for safety; and if this phase of the rules can be carried out it will do more for the safety of the miner than any previous law has been able to accomplish.

As we interpret the law, the mine boss stands the brunt of the responsibility about the coal mine. He must be alert to keep the mine in safe condition, and not take chances of any kind that in any possible way might result in loss of life or injury. In case he has any doubts in regard to the matter his only safe way will be to call on the inspector for advice and thus shift the responsibility.

If a railroader does or leaves undone anything that might cause an accident he is disciplined, or if through his neglect an accident occurs, he is discharged. The same drastic measures should be applied to mine workers, and while mine foremen are not always responsible for accidents in their mines, there have been many that could have been avoided had the mine foreman asserted his prerogative.

Mine inspectors, when working under the old law, claimed that they were hampered in many ways by mine officials who would not carry out their suggestions regarding safety. The new law will not permit them to furnish an excuse of this kind, for, as we understand it, the law was drafted by the inspectors to overcome such obstacles, and the operators, miners, and public now expect them to make good, and decrease the number of accidents.

The English laws are preeminently clear in expression and just to miners, operators, and the public. This is because they are drafted by experienced mining engineers and submitted to miners and operators for approval. They are not intended to suit alone those directly interested, but include the general public as well. Lawyers, whose sole business is to interpret laws, are not permitted to have a hand in their drafting, and consequently the laws are not befogged with useless words, as is the new Pennsylvania mine law. Among the new features we find: That there must be apparatus to provide means of signaling from top to the bottom, as well as from the bottom to the top; that there must be an efficient safety device to prevent overwinding; the speed of the cage when lowering or hoisting persons must not exceed 900 feet per minute; boilers must be inspected every 6 months; no boilers used for generating steam are to be placed in a mine without the consent of the inspector; the quantity of air for a non-gaseous mine is not to be less than 150 cubic feet per man per minute; in a gaseous mine the quantity of air is not to be less than 200 cubic feet per man per minute. No provisions are directly made for mine animals, oxidization, and emanations of gases, the determination of the quantity of air for such purposes being left to the inspector. There is to be no permanent door in the main entry unless allowed by the inspector. In 6 months after the act is signed not more than 70 persons shall work in one split unless the inspector says so. Cut-throughs in room pillars are to be made "at such distances apart as in the judgment of the inspector may be deemed requisite, but not more than 105 feet or less than 48 feet each for the purpose of ventilation." Every ventilating fan must be provided with a recording instrument, which is to register the number of revolutions or the effective ventilating pressure of the fan. "No main ventilating fan shall be placed inside the mine; and no auxiliary fan, unless driven by electricity." This stops the use of hand-driven fans in headings, rock tunnels, etc., if the law be construed literally. "No product of petroleum or alcohol, or any compound that in the opinion of the inspector will contaminate the air" is to be used as motive power. If the owner of surface land sewers into a mine he commits a misdemeanor and is liable to a fine of $1,000 and to undergo an imprisonment not exceeding 1 year.

The entire article on electricity is new. If any one owns or leases coal lands, and the laws cannot be complied with otherwise, he can make openings on another's land for drainage, ventilation, or ingress and egress.

Inside stables are to be excavated in rock, and if in coal the walls must be lined with concrete.

Not more than one barrel of lubricating oil is to be in a mine at one time; no greater quantity than 5 gallons of explosive oil is allowed at one time; oils used in open lamps "shall have a burning point not lower than 300 degrees, and must not produce over eleven one-hundredths of one per centum of their weight of soot when burned in a miner's lamp with a flame 1½ inches high. All illuminants to be used in open lamps shall be branded with "percentage of soot." There are to be 25 inspection districts with the inspectors' salaries fixed at $3,000 per annum. Their duties are prescribed in the law.

"SAFETY THE FIRST CONSIDERATION"

A WORKING system to promote carefulness on the part of the employes has been devised by P. J. Tormey, of the H. C. Frick Coke Co. At the bottom of the shaft when the men are about to step off the cage an electric sign flashes the words "Safety the First Consideration." At danger points within the mine other signs flash the words "Safety First." This is to impress sensibly on the minds of the men the fact of precaution and to admonish them to observe the rules established as a condition of employment.

In the recent Price-Pancoast disaster it was evident that too much confidence on the part of the men was placed on the management taking care of them. When told that the work was over, instead of hurrying out of the mine they took their time, and being unacquainted with the outlets of the mine were unable to reach a place of safety. The D. L. & W. Coal Co. and the D. & H. Coal Co. are now putting signs in the mines with arrows showing the way out in different parts of the mines. This is one more step toward preserving the lives of the miners, and we believe it should be universally adopted; further, the miners should be made thoroughly acquainted with all the live workings and how to get out in case one avenue of escape is stopped. If this practice is followed out, undoubtedly a large number of lives will be preserved.

FEDERAL AID FOR MINING COLLEGES

CONGRESSMAN M. D. FOSTER, Chairman of the Committee on Mines and Mining, introduced a bill at Washington which should appeal to mining men. The purport of bill H. R. 6304 is to extend the same aid to state mining schools as is now given state agricultural colleges.

The days when cotton, corn, and wheat were the sole medium of barter between the United States and other countries has passed, for the products of the mines have steadily increased in importance until their aggregate is more than corn or wheat. Agricultural products

find a market only where manufacturing and mining communities exist; mineral products find a market everywhere, even among farmers. The products from the farm are consumed and form but temporary wealth; the products from the mines, with a few exceptions, form permanent wealth that makes for civilization.

Whenever the value of this country's mineral products is appreciated by the general public, there will no doubt be created a Department of Mines, but until that time arrives there is no reason why in the distribution of benefits the miner should not receive his as well as the farmer.

BOOK REVIEW

Magnetic Properties of Heusler Alloys, by Edward B. Stephenson, has just been issued as Bulletin No. 47 of the Engineering Experiment Station of the University of Illinois. This bulletin is a contribution of data on the subject of magnetic alloys, and describes fully the methods of magnetic testing, thermal analysis and photo-micrography used in the work. The results show that an alloy of ferro-magnetic properties comparable with those of cast iron can be made of the non-magnetic components copper, manganese, and aluminum, and that the magnetic properties depend largely on the heat treatment. Copies of this bulletin may be obtained gratis upon application to W. F. M. Goss, Director of the Engineering Experiment Station, University of Illinois, Urbana, Ill.

The Bureau of Mines has begun the publication of a series of short papers to be known as "Technical Papers," intended to convey to engineers and others quickly the results of certain of the investigations. The first of these, known as Technical Paper No. 1, has just been issued under the title, "The Sampling of Coal in the Mine," and the author is Joseph A. Holmes, the Director of the Bureau. Those interested may obtain copies of this technical paper by applying to the Director of the Bureau of Mines, Washington, D. C.

The United States Geological Survey Press Bulletins are now issued in a more convenient form than hitherto. They contain abstracts of official reports made by the Geological Survey, and if there was an index of their contents on the first page of each number it would add to their convenience.

Hurd's Iron Ore Manual, by Rukard Hurd, C. E., St. Paul, Minn., Secretary Minnesota Tax Commission. Price $7.50. This is a general reference, guide, and hand book of the Lake Superior iron ore districts. It contains 162 pages of reading matter, tables and statistics of special value to those engaged in iron-ore mining in Michigan and Minnesota, and to those who are purchasers of iron ore from these fields. The iron ore districts of New York, New Jersey, Alabama, Tennessee, and Colorado, on account of their limited extent and production, and ownership by consuming interests, without a basic system of valuation of ores, are not considered. In the absence of published explanation the "basic system" has seemed very intricate, confusing, and mysterious to mining men generally, and all along the line, from mine to furnace, time and labor are consumed and wasted in miscalculating values and misapplying premiums and penalties. Even ore experts do not agree in their interpretations of the system. The book gives simple mathematical calculations to remove the confusion, besides much other data.

Coal Mining in Arkansas, Part 1. Issued by the Arkansas Geological Survey. This book is written by A. A. Steel, who has been in charge of the coal-mine investigations in Arkansas. It is divided into eight chapters and is a comprehensive review of coal-mining conditions in that state. The scope of the book is as follows: General Conditions Relating to the Coal and Coal Mines; Details of Mining; The Miners; Work and Wages of the Miners; Relations Between the Miners and the Operators; Mining Laws of Arkansas; The Mine-Run Law; General Condition of the Mining Industry; Glossary of Coal Mining Terms. The book is well written, illustrated, and printed.

Temperature-Entropy Diagram, by Charles W. Berry, 393 pages, 125 illustrations, price $2.50. Published by John Wiley & Sons. This is the third edition of Professor Berry's valuable book on the Temperature-Entropy Diagram. In the third edition the text of the second edition is extended so as to include much new matter. It treats of the following: Discussion of Reversible Processes and the Effect of Irreversibility; The Temperature-Entropy Diagram for Perfect Gases, Saturated Steam, and Superheated Vapors, and the Flow of Fluids; Mollier's Total Energy Entropy Diagram; Thermodynamics of Mixtures of Vapors, of Gases and Vapors, and of Vapors; The Temperature-Entropy Diagram Applied to Hot-Air Engines, Gas-Engine Cycles, Non-conducting Steam Engine, the Multiple-Fluid and Waste-Heat Engine, and the Actual Steam-Engine Cycle; Steam Engine Cylinder Efficiency; Liquification of Vapors and Gases and the Application of the Diagram to Air Compressors and Air Motors; Discussion of Refrigerating Processes and Kelvin's Warming Engine; and the Entropy Analysis in the Boiler Room. The work also includes a table of the properties of saturated steam from 40° F., and a table of Hyperbolic Logarithms. The work treats very clearly and fully the application of the Temperature-Entropy Diagram to these various conditions, and engineers interested in the design of various classes of machines mentioned would no doubt find this book of interest and service.

OBITUARY

NATHANIEL W. LORD

Nathaniel Wright Lord, born in Cincinnati, Ohio, December 26, 1854, died in Columbus, Ohio, May 23, 1911. Mr. Lord was of the class of 1876 Columbia School of Mines, and after some practical mining experience in Central America was made in 1879 Professor of Mining and Metallurgy in the Ohio State University. He became chemist of the Ohio Geological Survey in 1883, and for the past 8 years has been chief chemist or consulting expert of the Technologic Branch of the United States Geological Survey, now the Bureau of Mines. Professor Lord has done much for the mining industry of Ohio, and his death will be regretted by all who knew him intimately.

G. B. BOND

G. B. Bond, of Ballarat, N. S. W., a young man of much promise in the mining industry of New South Wales, died of peritonitis in the latter part of April. Mr. C. W. Mayo, M. E., and mine manager, in announcing his death, says: "At the time of his death, Mr. Bond was engaged in erecting a mill in New South Wales. He was associated with me for a number of years and occupied trustworthy positions as engineer at the Duke of Wellington Mine, Victoria, as mill foreman for the Colonel North Silver-Lead and Railway Co., in Tasmania, and as cyanide foreman at the British Lion Mine at Blackwood."

P. M. BOYLE

With regret we announce the death of P. M. Boyle, Mine Inspector of the Eighth Anthracite District of Pennsylvania. Mr. Boyle was well and favorably known in Northeastern Pennsylvania. He obtained his education while assistant foreman for Coxe Bros. & Co., in Drifton, Pa., working day time and attending the Coxe School at night. He was a zealous advocate of temperance and those who knew him will testify to his untiring endeavor to uplift and improve the moral conditions of his fellow man.

CORRESPONDENCE

Storing Coal

Editor Mines and Minerals:

SIR:—We would like to yard our nut and slack coal but cannot do so without it taking fire from spontaneous combustion. Our analyses show that we have 2.14 per cent. sulphur in our coal, and this mostly all goes into the nut and slack. We also have clay beds and a great deal of this gets into the nut and slack. Can you tell us how we can yard this coal so that it will not fire? Of course we know we could wash it but we do not want to do so.

P. W. V.

The Problem of Education

Editor Mines and Minerals:

SIR:—Having read with much interest the article in May MINES AND MINERALS by Mr. John Verner on "Coal Dust Explosions," I must say that I agree with his conclusion that intelligent cooperation of employes, which can only be brought about by education, is the greatest factor in preventing mine accidents. But there is another side to this question. Some months ago I arranged a series of weekly meetings for my employes, providing a hall for them to meet in, and arranging for lighting and other incidentals without expense to them. Lectures by competent mining men were provided, and the men were encouraged to tell their experiences and discuss all questions in their own way. Practical mining, as well as theory, was to be systematically studied and illustrated; the object being to thoroughly educate every employe so that he could act intelligently in every possible contingency, conserving his own safety and the welfare of his fellows. It was made plain in the beginning that the employers had no ax to grind, the sole motive being the education of the employes. I may say here that the employes are all English speaking, about one-third being negroes. The first meeting was largely attended and the discussion enthusiastic. At the second meeting there was a noticeable falling off in attendance of the more ignorant men. At the third meeting no negroes at all attended. At the fourth meeting only a few of the more intelligent of the white miners were present, the result being that the meetings were discontinued by common consent because of lack of support. The very men whom it was intended to reach, and who most needed instruction, refused to be instructed, notwithstanding every reasonable effort was made to induce them to come out. The problem is: How are you going to educate men who refuse to be instructed? I am open for light on this point. On account of the meetings being instituted by an employer, some sinister motive might have been ascribed had it not been that nearly all the men had been in my employ for many years, knew me well, and intimately; were on the friendliest terms with me, and gave me the honor of professing the utmost confidence in my justice and open fair dealing.

It strikes me that the first need is to instill into the average miner the desire for education; and I confess that that is a problem for which I have not the faintest idea of a solution.

WM. KELLUM

Superintendent Southern Coal and Coke Co.

Mining in San Domingo

Editor Mines and Minerals:

SIR:—I noticed in your May issue a letter from Mr. Haussler on Santo Domingo mining. I fear Mr. Haussler is not well informed on the present condition of the island in the West Indies.

There is now no mining law of 1904. The present mining law is dated April 10, 1910.

Quoting from Article 35, Chapter IV: "All proprietors of mines shall pay into the public treasury 2 per cent. of the gross products of the mines."

Article 38, Chapter IV: "The proprietors of the soil (the owners of the surface rights) shall have a right to 2 per cent. of the net product of the mines that are exploited on their property."

Article 43, Chapter V: "The present law annuls all other laws that may be to the contrary."

Like all such government documents, this new mining law is very voluminous. I have in my office a translation of the document; and it will give me pleasure to show it to any one interested.

I think well of Santo Domingo, R. D., as a mining country. I made a thorough examination of the San Francisco Hill, in the Cristobal district, Province Santo Domingo. I found very promising prospects, which since have turned out good mines.

For the last 15 months there has been active development work on the prospects I reported on. The present owners or lessees of this portion of the main Cristobal concession have been employing a number of hands under practical American miners. Several hundred feet of tunneling have been added at Bocara, many cross-cuts at San Francisco Hill, and several cross-cuts and shafts at Majagual.

The concession of the San Cristobal Mining Co. comprises 400 square miles. I have examined not more than 4 square miles, which has been leased, and is now under development. This is a typical mining country. There is copper with some gold and silver in all the hills here. The low lands and the river banks, especially near the River Jaina, offer rich placer propositions from Mt. Banileja to the Caribbean Sea; thus many hundreds of square miles could be profitably worked for placer gold. The gold is coarse, but very pure.

Parties wishing to obtain concessions of placer properties, usually engage native labor for washing, and to prevent mistakes or stealing, the natives are paid 60 cents for every pennyweight, plus 50 cents for a day's labor. Thus reliable results can be obtained as to the average gold values per cubic yard. The natives make use of the home-made "batea," a wooden pan about 11 inches in diameter and 3 inches in depth. The native women are adepts with these bateas.

There are mountains like Mt. Lomatina, in the Province of Azua, which are over 10,000 feet high. The mineral zone is very extended and it seems as if the whole island's saddle was mineralized. Most assuredly, Santo Domingo has wonderful possibilities as a mining country, and since the United States is and will be there for the next 50 years to come (looking after the revenues under the Monroe doctrine) and revolutions are a thing of the past, finances are safe, and investment in the island will be very profitable.

I brought back samples of copper, gold, silver, iron, chromite, nickel, and other metal ores, and I know there is petroleum on the island. Coal I have found, but owing to my not being able to thoroughly examine the deposit, I can say no more. An assay of copper ore is here attached which gave the following results:

No.	Copper Per Cent.	Gold Ounces	Silver Ounces	Iron Per Cent.	Lime Per Cent.	Sulphur Per Cent.
1	2.78	.09	.75	14.00	.08	1.60
2	24.40	.02	1.20	16.05	.04	2.00
3	1.89	1.08	.65	12.56	trace	trace
4	16.54	.75	.50	17.40	.09	1.35
5	21.28	.02	trace	12.50	.03	2.28
6	2 .60	.64	.25	9.60	.02	1.36
7	17.48	.07	.85	15.20	.03	1.15
8	1.89	.72	.70	12.40	.01	.89
9	3.60	.62	.48	18.19	.02	.05
10	18.47	.07	.28	11.85	.03	1.28
11	14.60	.03	.37	15.60	.02	1.00
12	17.39	.04	.28	12.36	.01	1.30

The climate is good, tropical of course, but no matter how warm during the day, I could not do without a blanket at night.

I would advise avoiding the use of alcohol and meat in this climate. There is a great variety of fruit at the mines, the very best I ever tasted; and one can afford to do without meat and other delicacies.

Labor is good and plentiful; native labor, 50 cents gold per day. Ernest Fahrig, E. M.

Bailey Building, Philadelphia, Pa.

Coal-Dust Explosions

Editor Mines and Minerals:

Sir:—The Mines and Minerals of May, 1911, contains an article by Mr. John Verner on "Coal-Dust Explosions," which, in my opinion, includes some errors of both fact and judgment and may create in the minds of readers wrong impressions on a very important subject. If the mistakes were those of judgment only it might be unnecessary to call attention to them as such mistakes are common and all are likely to make them, but there are certain errors of fact on which these opinions are based which I think should not be allowed to go without some correction.

The article begins with a quotation from a report published by a British commission on coal dust in 1894, which states that "coal dust alone without the presence of any gas at all may cause a dangerous explosion if ignited by a blown-out shot or violent inflammation. To produce such a result, however, the conditions must be exceptional and are only likely to be produced on rare occasions." It will be noticed first that this report was published 17 years ago, before any widespread attention had been devoted to the subject of dust explosions and that the conclusions there stated must of necessity have been founded upon a comparatively small number of observations. The opinion here stated is undoubtedly in the main correct, as is shown by the fact that dust explosions occur as rarely as they do. But the writer of the article referred to bases on this report and on certain other data the opinion that the principal factor to be considered in an investigation of dust explosions is not the coal dust, but apparently the air with which it burns.

After a discussion of the air-currents produced by blasting, the author advances the statement that a dust explosion can occur only in case the air in which the dust is suspended is subjected to more or less compression, using the following language: "It is the presence and influence of this dynamic force (*sic*) that produces high explosive effects and in its absence the mere contact of flame and coal dust has proved comparatively harmless. The results of experiments and laboratory tests clearly and convincingly show that explosive effects and the propagation of an explosion can only be produced by the forcible injection of air and dust into the flame. Peckman and Peck could only produce explosive effects by blowing the dust into a flame by means of a bellows, etc." The work of other experimenters is cited as showing that dust must be suspended by compressed air, and the paragraph closes with a reference to an experiment by Holtzwart and Meyer, in which an explosion was obtained when the dust was puffed between the terminals of an electric circuit, but not when the dust was shaken in the tube.

The data given here are not sufficient for a conclusion that the dust must be suspended by compressed air in order to be explosive. In the first place, as far as I can learn, the reason for using compressed air in all cases except the last was the simple fact that it offered the most convenient means of getting the dust into suspension. That it is not necessary has been abundantly demonstrated.

A large number of experiments along this line at the University of Kansas do not suggest any such conclusion. In the earlier experiments compressed air was used simply because it was convenient, but in later experiments, comprising by far the larger number the dust was placed in a box and thrown into the air by an agitator. In most cases ignition was obtained by moving aside the cover of the box and holding a naked flame at the opening. The air was under no pressure whatever, the only object sought being to suspend the dust in the air in the form of a cloud. It was found that the result of the experiment depended more upon the quantity and condition of the dust than upon any other factors. Some dusts were inexplosive in any quantities, others were explosive in small quantities, others in large quantities, others only in the presence of gas. The following examples will illustrate:

No. 1. Drill dust from W. C. & C. Co., No. 16. Explosive limit 2.9 grams.

No. 2. Dust from haulageway W. C. & C. Co., No. 15. Inexplosive alone; 3 per cent. natural gas required to make it explosive.

No. 33. Dust from Monongah mine No. 8. Explosive limits 2.1 grams and 2.8 grams, depending on the method of selecting the dust.

These experiments are selected from a considerable number for the purpose of showing that the explosive limit depends upon the quality and quantity of the dust, other things being the same. The statement quoted by the author that "Unless there is an exceptionally large amount of dust in the air, experience shows that ignition does not take place from a naked flame," is undoubtedly correct. The reason, however, is that unless the dust exists in large quantities the mixture with the air is not explosive, but the quantity required depends upon the character of the dust. If the dust contains large quantities of combustible volatile matter, relatively small quantities suspended in the air will be explosive, but if it contains little combustible volatile matter our experiments tend to indicate that no quantity will be explosive. I wish to emphasize the statement that the conditions necessary for an explosion are first, an explosive dust; second, the suspension of a sufficient quantity of it in the air. The author quotes experiments of the Chesterfield and Derbyshire Institute of Engineers in which out of 134 tests ignition was obtained in 36 cases, and no violent explosion was obtained even with 6 per cent. of gas. A horse pistol was used to ignite the mixture. The conclusion is drawn that failure to produce more explosions was due to the fact that the firing of the pistol did not produce sufficient compression of the air. I do not believe that this conclusion is justified. The data are not sufficient. In the first place we have no knowledge whatever of the quantity of dust suspended, and second, we know nothing of its quality. Either it must have been very small in quantity or almost inexplosive, to show absence of violent explosion in the presence of 6 per cent. of gas, because this much gas alone is explosive under favorable conditions. I assume here that gas means methane. This statement that 6 per cent. of gas alone can be exploded is based upon a large number of experiments and our experiments have also demonstrated that gas and coal dust can replace each other in explosive mixtures. The fact that so few explosions were obtained in these experiments is very good proof that the dust was nearly inexplosive, or was present in small quantities, or else that the flame from the horse pistol was insufficient to ignite even an explosive gas mixture.

Farther on in the article the author advances the idea that increase in the density of the air due to decrease in temperature is likely to be a determining factor in the occurrence of explosions. I think it probable that increase in density of air may make a dust mixture somewhat more explosive, but that the effect due to a natural change in temperature would be very slight indeed. But the author's reason for this increased explosiveness is at least not well stated. He refers it to an increase in oxygen and a little later says: "The quantity of air rather than the quality of dust is really the measure of the magnitude of an explosion." I wish to take issue with this conclusion, and I can best give my reasons for doing so by making some statements concerning dust explosions in general.

An explosion of dust is not a detonation but a burning so rapid that a violent expansion of air and gases occurs. This

expansion is due partly to the formation of gases and vapors during the burning of the coal, and partly to the expansion of gases and vapors because of the increase of temperature. Coal dust can burn with explosive rapidity only when it is suspended in the air. Otherwise the supply of oxygen will be too small.

The propagation of the combustion through the mixture depends upon the quality and quantity of the dust. In other words, some dusts will burn only in the immediate neighborhood of the igniting flame, others will burn throughout the mixture, and this burning constitutes a dust explosion. In order that a flame may be propagated through any mixture of a combustible substance with air, it is necessary that the combustion at the point of ignition furnish sufficient heat to raise the surrounding particles to the ignition temperature. If the dust furnishes a large amount of heat, the combustion will be propagated. If it does not, burning will take place only in the immediate neighborhood of the ignition agent. The quantity of heat furnished depends upon the quality and quantity of the dust. The dust must also be readily ignitible, and it will be so if it contains large quantities of combustible volatile matter. This volatile matter will be distilled by heat and will be intimately mixed with the air. The difficulty of igniting anthracite dust is due to the fact that it contains very little volatile combustible matter. Therefore, it does not readily explode. I do not know whether it is possible to explode it or not. Experiments at the university lead to the conclusion that it is not possible, but I do not wish to say positively that its explosion is impossible.

The error in the author's conclusion that explosiveness depends largely upon the quantity of air present, lies in the fact that in most cases there is too little dust present to be explosive. In other words, the air is greatly in excess of the required amount. The dust present does not furnish sufficient heat for sustained combustion; therefore, there is no explosion. It is only when considerable quantities of dust are suspended in the air and when this dust is of readily combustible character that an explosion occurs.

One of the gravest sources of danger lies in the fact that a deficiency of volatile combustible matter in the coal may be made up by the presence of gas in the mine air. For example, in the case of one sample of dust, whose limit in our apparatus was 2.3 grams, it was found that .7 gram of dust would explode in an atmosphere containing 3 per cent. gas, .8 gram in an atmosphere containing 2 per cent. of gas and 1.2 grams in an atmosphere containing 1 per cent. of gas. The presence of so little as 1 per cent. of gas cut in two the quantity required for explosion, and 2 per cent. of gas practically divided the quantity by three. The ordinary means of detecting firedamp will hardly show 2 per cent. of gas and probably most fire bosses will not detect less than 3 per cent. It is readily seen then that dust may be a very great source of danger even when it is supposed to be harmless. I wish to emphasize in conclusion my beliefs, founded upon a large number of experiments, that the conditions necessary for a coal-dust explosion are that the dust shall be of such quality as to be explosive, and that it shall be suspended in the air. Also that gas and dust are completely interchangeable in explosive mixtures. In other words, there may be a pure gas explosion or a pure dust explosion, but I believe that in most cases the explosion is due to both dust and gas, because I greatly doubt whether any coal mine is entirely free from gas. The experiments referred to, and a large number of others, will be given in detail in the forthcoming Volume X of the University Geological Survey of Kansas.

I wish in conclusion to very heartily commend what is said by the author concerning the training and education of all persons connected with the coal-mining industry. I believe that the only way to prevent coal mine explosions and other accidents in mines is to thoroughly educate all persons concerned with the industry to a real appreciation of all of its conditions. Until this is accomplished it will not be possible to prevent explosions. It is for this reason that I have been so bold as to call attention to what in my opinion are errors in the article referred to.

C. M. YOUNG
Asso. Professor Mining Engineering, University of Kansas

TRADE NOTICES

A New Solder in Paste Form.—Recently a new kind of solder has been placed on the market. It is in the form of a paste in a collapsible tube, like the familiar tube of tooth paste, and all that is necessary for its effective use is to scrape off the surface of the part a little with a knife, squeeze some of the soldering paste on and apply a match, candle, or torch. When the paste becomes hot it fuses and solders in the same manner as the old-style soldering stick. The name of the new device is Solderall, and it is marketed by the H. W. Johns-Manville Co. through their branch houses in various cities throughout the country. Its convenience, cleanliness, economy, and many other advantages have naturally made a wide appeal to householders as well as to plumbers, tinsmiths, electricians, hardware and supply stores and others.

Goulds Mfg. Co. Plant.—The new barge canal that is being built in New York State passes through Seneca Falls and requires the destruction of part of the old plant of the Goulds Mfg. Co., of that place. To take the place of the condemned buildings and to provide facilities for rapidly increasing business, the company is about to build four new buildings—a storage house for rough castings, two machine shops 300 ft. × 100 ft., and a four-story warehouse 240 ft. × 60 ft. The buildings are to be ready by November 1.

Ohio Representative of the Goodman Mfg. Co.—The Goodman Mfg. Co., of Chicago, have arranged with Mr. Parker Cott to represent their line of electric coal-mining machines and locomotives in Ohio, with headquarters at Athens, Ohio. Mr. Cott has been for the past 5 years division mining engineer for the Sunday Creek Co., and his familiarity with the Ohio coal field and his training and experience in engineering problems in coal mining will enable him to be of assistance to the operators in this field.

Office Ventilation.—The American Blower Co., of Detroit, Mich., is marketing the "Ventura" electric ventilating set. This consists of a 12½-inch motor-driven ventilating fan designed to be fitted into a window or transom. By a simple reversing arrangement the fan can be made either to exhaust air from a room or to blow it in and discharge it in any direction desired. The apparatus is furnished complete ready for connection to a plug or electric lamp socket and can be put in place by any carpenter.

Notice of Infringement.—The Williams Patent Crusher and Pulverizer Co., 545 Old Colony Bldg., Chicago, has issued a notice to the public stating that the United States Circuit Court of Appeals, at Philadelphia, rendered its final decision in a suit brought by them against the Pennsylvania Crusher Co. for infringement of patent No. 843,729, for improvements in dumping cages for crushers and pulverizers. This decision of the Court of Appeals handed down in the March term, 1911, concludes "the record will be remanded with instructions to the Circuit Court to enter a decree reversing the former one adjudging claims 1 and 2 of the patent in suit to be valid and infringed and awarding to the complainant an injunction with the usual accounting and costs of suit." Under the law, a user of an infringing machine is liable for his acts of infringements. The maker and seller of the infringing machine in question has been found to have infringed patent No. 843,729 and the Court of Appeals, in addition, has found the infringed claims of said patent to be valid. The company states it is their intention to protect their rights as secured by the above patent and numer-

ous other patents which have been granted them on improvements made in crushing and pulverizing machinery and they warn the public against buying crushing and pulverizing machines which infringe any of their 87 patents.

Centrifugal Pumps for Mine Drainage.—The De Laval centrifugal pumps contain many excellent features, both as regards efficiency and economy, that we can only briefly refer to in this department. Centrifugal pumps have, by actual use in mine drainage, as well as for other pumping purposes, demonstrated their superior worth, and the De Laval pumps are the most widely known of the centrifugal type. Mine officials who have drainage problems to meet, no matter what their nature, especially if the water is strongly acidulated or gritty, will be well repaid by an examination of the handsome and well-arranged catalog B of the De Laval Steam Turbine Co. It is in a measure a textbook on centrifugal pumps and contains, besides the illustrations descriptive of De Laval pumps, much information of value to any man having any interest in pumping propositions. It will be sent free on request to mine officials by the De Laval Steam Turbine Co., of Trenton, N. J.

Denver & Rio Grande.—Announcement is made by the Denver & Rio Grande Railroad of improved train service between Denver and Salt Lake City and San Francisco. A new fast train—making four trains daily in each direction—has been added, and the running time reduced 2 hours. The Western Pacific has added a second daily train over its line between Salt Lake City and San Francisco, with through standard and tourist sleeping cars on both trains between San Francisco and Chicago in connection with the Denver & Rio Grande. The opening of the Western Pacific (Pacific Coast extension of the Denver & Rio Grande) has in no way changed the relations of the Rio Grande with the Southern Pacific and the San Pedro lines. Through cars over the Denver & Rio Grande and the Harriman lines will continue to be operated as heretofore, the new service via the Western Pacific merely supplementing that now in effect through the Ogden gateway.

New Home of Electric Service Supplies Co.—Electric Service Supplies Co. has recently moved into its new office and factory building, at 17th and Cambria Sts., North Philadelphia. The old factory, the outgrowth of the old Garton–Daniels Co., at Keokuk, Iowa, as well as the Philadelphia office and warehouse, formerly at 1020-24 Filbert Street, have been discontinued and all are housed in this new building. The site permits room for expansion, is but five blocks from both passenger and freight stations of the Pennsylvania and Reading railroads, and has a railroad siding on the property. This building, which is the first of a group to be erected on the site, is a six-story, monolithic, reinforced-concrete structure, fireproof in construction and equipped with automatic sprinkler system capable of extinguishing any blaze that might occur within the building. It contains machinery for the manufacture of all the varied devices of the company, as well as room for carrying a large stock of finished products, and is equipped with all the most modern facilities for light, heat, comfort, and convenience. The general offices are finely finished and conveniently arranged, and will enable a large business to be done economically.

The Gardner Crusher, for crushing or reducing coal, ore, rock, or any material, embodies in its construction many features of interest to mine or mill owners and managers. Its advantages, as shown in a concise and well-illustrated catalog, are briefly as follows: It can be fed with large pieces of coal, ore, rock, or whatever material is to be crushed. It has very large capacity. It requires comparatively small motive power. It is easy and cheap to install. Wear and tear are almost eliminated. It occupies but little space. It can be regulated to crush to any degree of fineness. It is solidly constructed, and is moderate in price. Free crushing tests are made at the demonstrating plant of the Gardner Crusher Co., whose offices, factory, and demonstrating plant are located at 550 West 34th Street, New York City. Catalogs describing the crusher in detail will be sent any mine or mill official on request.

Advertising Agency.—Wm. M. Chamberlain has opened a general trade and technical journal advertising agency, with office at 71 Griswold Street, Detroit, Mich. Wm. B. Milligan has been engaged as chief of copy department, and the agency is prepared to conduct a general business-promotion department for its clients.

Western Electric Co.—The business of Western Electric Co for May was 12 per cent. in excess of the same month a year ago and for the 5 months to June 1 sales have exceeded the same period of 1910 by 8 per cent. At the end of May the company was employing 26,000 men. When business was at the top in 1907 the company had a total of 29,000 employes, although the volume of production for that year was between 4 per cent. and 5 per cent. less than this. Increased efficiency of operation explains the difference.

Removal.—Ogden Assay Co. announce the removal of their office to more convenient and central quarters, 1711 Tremont Street, where they have one of the best fitted laboratories in the West for assaying and general analytical work. The chemical laboratory is under the direction of Mr. H. R. Brandenburg, D. Ch., who has had a wide experience, both in Germany and the United States, as an industrial chemist, and the company are prepared to work out all problems in their line for the benefit of their patrons.

NEW LEGISLATION WILL AFFECT MINING SHARES

The bill which Governor Foss, of Massachusetts, has signed relative to the listing and advertising of shares of stock of mining corporations, to take effect in 30 days, is as follows:

Sec. 1.—An officer of a mining corporation who makes a false statement, knowing the same to be false, in an application to any stock exchange to list the shares of such corporation, shall be punished by a fine not exceeding $500, or by imprisonment for not more than two years.

Sec. 2.—No officer, agent, clerk, or servant of a mining corporation, nor any person dealing in the shares of such corporation, shall cause to be published any advertisement of the shares of such corporation in which any statement is made of the value of the property of the corporation, or of its present or prospective earnings, or of a prospective increase in the price of the shares, unless the president and a majority of the directors of such corporation within 60 days prior to the date of the publication of such advertisement have filed with the commissioner of corporations, in such form as he shall prescribe, a statement under oath of the financial condition of the corporation, a full description of its property, and a statement of the earnings, if any, from the operation of the same for the fiscal year next preceding the date of the filing of the said statement.

Sec. 3.—Whoever, having caused the publication of such advertisement, or being a promoter, officer, clerk, or servant of a mining corporation, or a broker or agent for such corporation or such promoter or for the person or corporation causing the publication of such an advertisement, sells or offers for sale any shares of stock in such corporation knowing that any statement in such advertisement is false or is inconsistent with a statement filed under the provisions of section 2 of this act, shall be punished by a fine of not more than $500, or by imprisonment for not more than two years.

Sec. 4.—Whoever violates the provisions of section 2 of this Act shall be punished by a fine of not more than $200, or by imprisonment for not more than one year, and an officer or director, of a mining corporation who signs any false statement filed with the commissioner of corporations under the provisions of said section, knowing such statement to be false, shall be punished by a fine of not more than $1,000, or by imprisonment for not more than two years.

MINE SUPPORT TESTS

In order to arrive at some definite conclusion relative to the strength of the various materials used in anthracite mines for the purpose of supporting roofs, Messrs. Conner and Griffith, of the Scranton Mine Cave Commission, had Prof. Frank C. McKibben make a series of tests for them at Lehigh University. The results of these tests are of considerable importance to engineers and are therefore copied from their report. The words in parentheses are not taken from the report. The results of the engineers' findings were printed in the May issue of MINES AND MINERALS. The report, however, is so long it is impossible to make use of it all at one time, besides there are necessarily considerable data unimportant to engineers outside of Scranton.

Tests of Crushing Strengths of Supports Made of Different Materials Used in Mines

Test 1.—This test consisted in crushing a pillar made of mine rock, four sides of which were laid vertically and then filled with stone of various sizes packed in by hand so as to make the pillar of loose stones without mortar of any kind. This pillar was 5 feet long, 2 feet 4 inches wide, 18 inches high. It was laid directly on the bedplate of the testing machine and under three steel beams, which were later used in applying the load over the entire top surface of the pillar. The stone used was slate, bony coal and fireclay. The load on this pillar was applied in increments until a maximum pressure of 489,150 pounds was reached. When the maximum load was reached, which approximated 42,000 pounds per square foot, the maximum compression amounted to 5.3 inches. (This kind of roof support is used in coal mines with satisfactory results, in fact dry-pack walls are almost universally adopted in longwall mining to keep open the haulage roads and airways. In Fig. 1 is shown a dry-masonry pillar in an anthracite mine which has

FIG. 1. DRY-MASONRY ROOF SUPPORT

evidently been adopted as a temporary support. In cases where it is desirable to have a support not so likely to collapse when pressure is thrown on it, masonry pillars, such as that shown in Fig. 2, are substituted. Pillars having areas so small as those shown in Figs. 1 and 2 are of little use for more than local roof supports, and are not intended to support the entire cover up to the surface.)

Test 2.—This test consisted in crushing a timber crib, shown in Fig. 3, made of four layers of round timbers which were about 5 inches in diameter. Each of two of the layers consisted of two of these round timbers 5 feet 4 inches in length, and each of the other two layers consisted of three round timbers 2 feet 8 inches in length. The spaces between these

FIG. 2. MASONRY PILLAR

timbers were filled with slate, bony coal, and fireclay, and the crib was then filled wth small stones shoveled in, the whole resulting in a timber crib 5 feet 4 inches in length, 2 feet 2 inches wide, and 23½ inches high. The load was applied in increments. The maximum load reached 900,000 pounds and the settlement was 7.1 inches. (This kind of mine support is used in both coal and metal mines for temporary work, but is not suitable for permanent support. The timbering shown in Fig. 4 will probably last 3 or 4 years under favorable conditions, but eventually it will be attacked by dry or wet rot unless antiseptically treated previous to its being taken into the mine. Fig. 5 shows another timber crib that has been attacked by wet rot, the fungi showing above and back of the men.)

Test 3.—This test consisted of crushing a circular pillar 28 inches in diameter and 14½ inches high, made of slate arranged so that the outer surface was fairly smooth, the spaces between the stones and the interior of the pillar being filled with small stones. The load was applied in increments, the maximum load being 361,000 pounds with a corresponding settlement of 4½ inches.

After tests 1, 2, and 3, the stones were found to be very badly crushed, many having been reduced almost to powder, especially those immediately under the load.

Test 4.—This consisted in loading a pile of broken stones and observing the settlement caused by the loading. The stone used was crushed sandstone which would pass through a ring 1¾ inches in diameter and 40 per cent. voids, but under the bearing plate where the load was applied the voids were filled with small broken stones so as to get a secure bearing. The pile of stone was 25 inches wide, 9½ inches high, 2 feet 10 inches long on the top and 4 feet 5 inches long on the bottom. The load was applied on a cast-iron bearing plate 20 inches square, which rested on the top of the pile of stones. The maximum load reached was 581,000 pounds and the maximum settlement was 4.36 inches. After the tests many of the stones were found reduced practically to a powder, those stones under the bearing

plate being greatly disintegrated and the plate was pressed downward into the stones. At the ends the pile moved outward as the load was applied, but on the sides the pile was confined by timbers which prevented lateral movement.

Test 5.—This test consisted of crushing a pile of broken sandstone into varying sizes up to pieces as large as a man's

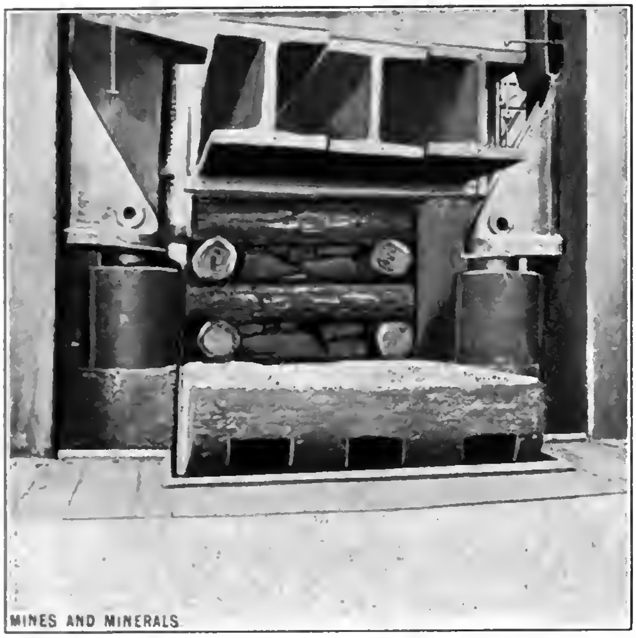


Fig. 3. Crushing a Timber Crib

head. Small stones were placed under the bearing plate. The stones were confined on the sides but were free on the ends and were not laid in any order. The pile was 25 inches wide, 11½ inches high, the length was 3 feet 8 inches on top and 5 feet at the bottom. The load was applied by a cast-iron bearing plate on the top of the pile, the plate being 20 inches square, the maximum load was 417,000 pounds and the maximum settlement was 4.6 inches.

Test 6.—This test consisted of applying a load to a pile of river sand by means of a 20″×20″ bearing plate. The pile was 8 inches deep, 2 feet 6 inches long on top, and 4 feet 2 inches long on the bottom, having a width of 25 inches, and was confined on the sides but not on the ends. The maximum load reached 600,000 pounds, and the maximum settlement was 5 inches.

Test 7.—This test consisted in crushing a pile of broken sandstone having 40 per cent. voids and sizes that would pass through a ring 1¾ inches in diameter, mixed with river sand in proportion of 10 volumes of broken stone and 4 volumes of sand. The pile was 10½ inches in diameter, 25 inches wide, 2 feet 5 inches long on top and 4 feet 6 inches long on bottom, and was confined on the sides but not on the ends. The load was applied on the top of the pile through a 20″×20″ bearing plate. The maximum load was 800,000 pounds and the maximum settlement 4.7 inches.

Test 8.—This test consisted of filling a cast-iron cylinder by flushing in coal culm with water until the cylinder was filled with the culm. The piston was then placed on top of the culm and the whole allowed to stand on a boiler for two days, after which the cylinder was then placed in the testing machine and the pressure applied to the piston, which in turn communicated the pressure to the culm within the cylinder. The culm was confined within the cylinder, the dimensions of which were as follows: Inside diameter 6.4375 inches, inside depth of cylinder 10.4375 inches, depth of culm in the cylinder 10 inches. The pressure was applied to the piston and the culm was compressed until the settlement reached 2.7 inches under a load of 200,000 pounds. This load corresponds to a pressure of 6,150 pounds per square inch, or 443 short tons per square foot.

Test 9.—This test consisted of applying a pressure to the piston of the cylinder in the same manner as in test No. 8, except that broken dry sandstone instead of coal culm was used. This broken sandstone has 40 per cent. voids and the pieces would all pass through a ring 1.75 inches in diameter. The cylinder was filled to the top, giving a depth of 10.4375 inches to the stone. The maximum load applied was 300,000 pounds, which caused a settlement of 3½ inches, at which time the pressure was 9,200 pounds per square inch. As a result of the test the stone was completely crushed and compressed into the iron cylinder so that it had to be cut from the cylinder with a chisel.

Test 10.—This test consisted of applying pressure to the piston of the cylinder in the same manner as in test No. 9, except that the broken sandstone, which had 40 per cent. voids and pieces which would all pass through a ring 1.75 inches in diameter, had voids filled with river sand. The cylinder was filled to the top, giving a depth of 10.4375 inches to the mixture of stone and sand. The maximum load applied was 300,000 pounds which corresponds to a settlement of 2.4 inches, at which time the pressure was 9,200 pounds per square inch. As a result of this test the stone was completely crushed and compacted in the iron cylinder.

Test 11.—This test consisted of applying pressure to the piston of the cylinder in the same manner as in test No. 9, except that cinders formed by burning anthracite under boilers were used in the cylinder. The cylinder was filled to the top with cinders which had 64 per cent. voids. The maximum load applied was 300,000 pounds, corresponding to a settlement of 5.3 inches, at which time the pressure was 9,200 pounds per square inch.

Test 12.—This test was exactly similar to test No. 8, except that the cylinder filled with culm was allowed to stand for

Fig. 4. Timber Crib in Mine

8 days over a boiler with the piston removed. The culm was 9 inches deep in the cylinder and the pressure was applied to the piston until the settlement reached 3 inches under a load of 300,000 pounds. This load corresponds to a pressure of 9,200 pounds per square inch. Although the culm had been drying for 8 days there was considerable water in it. The water was squeezed out during the test.

(In the table, Summary of Tests, the maximum settlement of culm is given as 33 per cent.; however, this percentage does not take into account that previous to the test there was a settlement of 10.4375 − 9 = 1.4375 inches before pressure was applied and afterwards 3 inches, or a maximum shrinkage of 42.5 per cent. According to former experiments there is a

FIG. 5. FUNGI ON TIMBER CRIB

settlement of 31 per cent. in flushed culm after drainage is complete; if to this 33 per cent. is added pressure shrinkage, culm flushing as a mine support does not look inviting. The end of a pipe line for culm flushing is shown in Fig. 6, with water and culm escaping. Culm occupies about 46 per cent. more space than coal in the solid, and for flushing requires from 1½ to 6 pounds of water per pound of culm according to the distance and inclination of the pipe. So far culm has been used as a temporary roof support where pillars are to be robbed, and not as a surface support.)

SUMMARY OF TESTS

No.	Description of Test	Maximum Load: Total Pounds	Maximum Load: Pounds Per Square Foot	Maximum Settlement: Inches	Maximum Settlement: Per Cent.
1	Rectangular pillar of mine rock..........	489,150	42,000	5.26	29
2	Timber crib filled with mine rock..........	900,000	63,300	7.08	30
3	Circular pillar of mine rock..............	361,600	85,000	4.51	31
4	Pile of broken sandstone, small sizes....	581,000	209,000*	4.36	46
5	Pile of broken sandstone, large and small sizes..............	417,000	150,000*	4.61	41
6	Pile of river sand......	600,000	216,000*	5.00	63
7	Pile of small broken sandstone and sand..	800,000	228,000*	4.69	45
8	Wet culm in cylinder..	200,000	886,000	2.73	30
9	Broken sandstone in cylinder............	300,000	1,330,000	3.66	35
10	Broken sandstone and sand in cylinder.....	300,000	1,330,000	2.42	23
11	Cinders in cylinder.....	300,000	1,330,000	5.33	51
12	Wet culm in cylinder..	300,000	1,330,000	3.00	33
13	River sand in cylinder.	300,000	1,330,000	3.35	32
14	Pillar of mine gob.....	600,000		2.43	27

* Pressure under 20″×20″ bearing plate.

Test 13.—This test is exactly similar to Test No. 8, except that the cast-iron cylinder was filled with Delaware River sand, the sand being placed in the cylinder and settled by shaking until the sand was flushed to the top of the cylinder. A load was then applied to the piston until a maximum pressure of 300,000 pounds, with a corresponding settlement of 3.1 inches, was reached. The sand was dry.

Test 14.—In this test pure sand was flushed into a cylinder. The sand was allowed to dry for a period of 48 hours. The top of the sand was 1.125 inches below the top of the cylinder. The maximum pressure was 300,000 pounds and the settlement 1.93 inches.

Test 15.—This test consisted of a pile of blue measure sandstone; the size of the pieces 3 to 6 inches square, 20.25 inches wide, and 3 feet 3 inches long on top and 9 feet deep, voids on top being filled with a small quantity of broken Potsdam sandstone for bearing. On the sides the 3-foot 3-inch dimension was confined by 6″×8″ timbers, which confined the material on the two sides and the base plate of the machine, and the I beams confined the material on the top and bottom, where on the ends the material was free to move outward, but in the test there was very little movement at the ends. The loads were applied to the top of the pile in increments and the maximum load reached 600,000 pounds, and there was a subsidence of 2.43 inches equal to 27 per cent.

AN IMPORTANT OIL LAND DECISION

First Assistant Secretary of the Interior Frank Pierce, in the case of the application of the Bakersfield Fuel and Oil Co., for a patent to 160 acres of oil land has rendered a decision of much interest to oil operators and investors in shares of oil companies. In June, 1899, eight persons attempted to locate 160 acres as a single oil placer claim. Before any discovery of oil was made, in August of the same year, these eight conveyed their alleged claim to the Bakersfield Fuel and Oil Co.,

FIG. 6. FLUSHING CULM

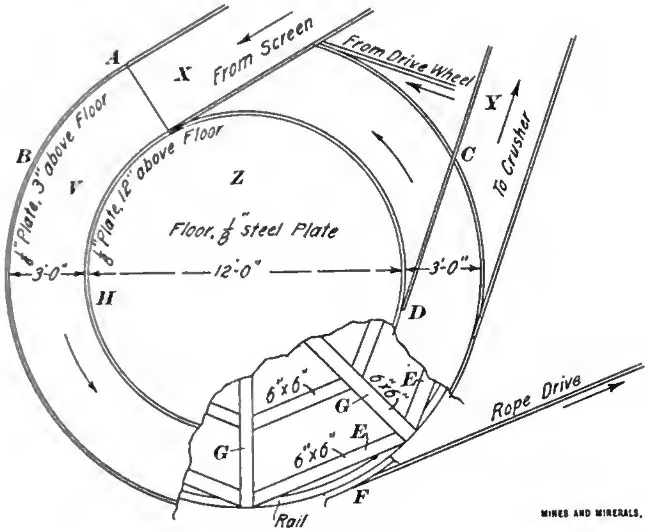
which sank a well and discovered oil in paying quantities in September, 1900. The application of the company for a patent has been denied on the grounds that the corporation has no greater rights than an individual, and is consequently entitled to but 20 acres of land, the amount conveyed in patents of placer lands after discovery of mineral and to single persons.

Mining Wrinkles

Inspection Table at the Cokedale Mine

Written for Mines and Minerals

The entire output, some 1,300 tons daily, of the Cokedale mine of the Carbon Coal and Coke Co., near Trinidad, Colo., is converted into coke. After the tipple was built it was found



Plan of Inspection Table

necessary to provide for a careful inspection of the coal as coming from the mine, that too much rock might not be sent out, thus overloading the washers and affecting the ash content of the coke. By reason of the screens, chutes, crushers, etc., being already installed, and by reason of the limited space available, the inspection table at this plant presents many points of interest.

As built, it consists essentially of a revolving table Z, 18 feet in diameter upon which the coal from the mine is fed through the chute X, and from which it passes through the chute Y, to the crusher. From the crusher it is elevated to the bins, from which it is fed into the washer. The revolving table is built similarly to an ordinary wagon wheel of $6'' \times 6''$ timbers, the spokes $G\,G$ braced by other $6'' \times 6''$ timbers $E\,E$, etc. Upon this framework is spiked 1-inch plank, and upon this planking are bolted $\frac{1}{8}$-inch steel plates. The framework is thus 7 inches thick.

The space between the braces $E\,E$, etc., and the outer circumference of the wheel is filled solidly by two pieces of $3'' \times 9''$ oak (original dimensions), the wide sides bolted together and cut with the proper radius, in this instance, 9 feet. Two grooves are cut around the circumference in this oak filling to engage the rope drive by which the table is rotated at the rate of $2\frac{1}{2}$ revolutions per minute.

To the under side of the table, and a few inches within the circumference, is bolted a rail with its base upward. The rail is continuous, of any convenient weight in use around the mines, and is bent in the form of a circle, the radius used at Cokedale being about 8 feet 9 inches, or 3 inches less than that of the table. This rail bears upon eight 16-inch pit-car wheels set radially and equidistant apart in the platform floor, serving as a bearing for the outer rim of the table.

The outer rim $B\,C$ is composed of a $\frac{3}{8}$-inch steel plate, bolted to the circumference, and set with its top 3 inches above the floor of the table. The inner rim $D\,H$ is also made of $\frac{3}{8}$-inch steel plate, but is 12 inches high. The table thus consists of an annular ring 3 feet wide with an outer diameter of 18 feet and an inner one of 12 feet.

The side of the chute X is extended with a height of 12 inches from A to B to prevent any coal dropping on the platform and is, of course, not attached to the table. The crusher chute Y is also extended with a height of 12 inches from C to D so that whatever coal may be backed up against it will not roll over on the center of the table Z. The chute X, into which the coal is dumped at about the rate of two mine cars a minute, is provided with an automatic feeding device, driven by a separate leather belt from the rope drive, so that the coal is spread uniformly over the annular space V. As this space moves with a tangential speed of but $2\frac{1}{3}$ feet per second and is but 3 feet wide, the inspector, who stands at about the point F, can not only plainly see, but easily remove any impurities.

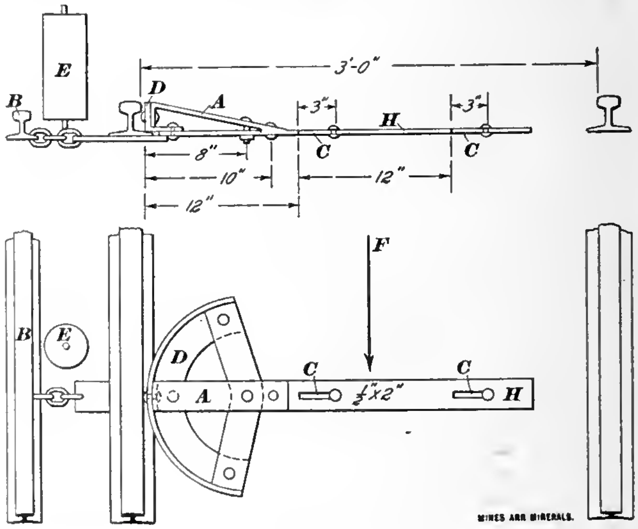
The outer circumference at Cokedale is $56\frac{1}{2}$ feet and the space from the foot of the feed-chute X to the mouth of the crusher chute Y, available for inspection, is about 40 feet. Should it be desired to use this table for picking by hand, from five to eight operators can easily stand around it. In this latter case, it would seem advisable not to floor over the inner 12-foot space Z, but to leave it open to a rock chute below into which the impurities could be thrown. To avoid injury to the timbers, if used as a picking table with open center, it might be advisable to cover the spokes $G\,G$, and the braces $E\,E$, etc., with thin sheets of iron.

We are indebted to Mr. Jacob Spahr, outside superintendent at Cokedale, for the above description.

Rope Throw at Radiant Mine

Written for Mines and Minerals

In dropping a trip down grade around a curve it not infrequently happens that the rope binds under the head of the inner rail. In event of the trip being long and the weight upon the rope consequently great, it is often necessary for an attendant with a bar to force the rope out from under the rail so that



Rope Throw

it may enter the sheaves. To do away with this extra labor an effective and cheaply constructed device is in use at the Radiant mine of the Victor-American Fuel Co., near Florence, Colo., and for a description of it we are indebted to J. Q. McNatt, mining engineer, of the latter city.

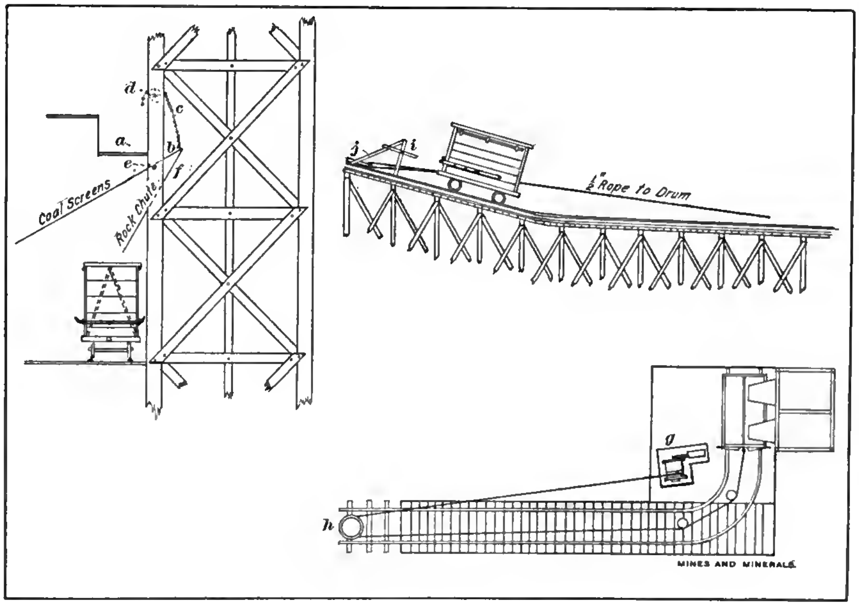
The device shown in the drawing is placed at the point where it is found that the rope binds under the rail. It consists of a plate of bar iron H, $\frac{1}{2}$ in. $\times$ 2 in. in cross-section and of a

length dependent largely upon the gauge of the track. To this bar at the inner rail of the curve is bolted and welded the segment of a circle D, made of angle iron, or a piece of a broken mine-car wheel may be used. At Radiant, this segment has a radius of 9 inches. On top of this is welded a plate A of bar iron $\frac{1}{2}$ in. $\times$ 2 in. in cross-section. Two slots, C and C, 3 inches long, are cut in the main bar. The bolts through these slots may either be driven into the cross-tie upon which the bar is placed, or, in event of the bar sliding upon a plate, as is the usual practice, they may be driven through this bearing plate. The inner end of the main bar H is attached to a spring rail B, the fulcrum of which may be so placed as to give any desired degree of tension.

After the empty trip has passed in the direction of the arrow F, the segment D, being forced out by each car, is brought back into place by the spring rail B, and after the trip has passed around the curve the rope begins to leave the center line of the track and to bear toward the inside of the curve. In so doing it is raised by the plate A above the top of the rail, and is thus, automatically, brought to bear upon the vertical roller, or spindle E. Beyond E, in the direction of the arrow F, are placed the ordinary horizontal-grooved sheave wheels, into which the rope is led by inclined guide blocks.

Rock Handling Arrangement at Globe Mine

During the development work at the Globe mine of the National Fuel Co., near Walsenburg, Colo., a very large proportion of rock to coal has to be handled. As ordinarily arranged, the cage, which is of the self-dumping type, would be stopped at the ground level, or a little above, and the cars of rock pushed to the dump by hand. To save this labor, Joseph Watson, the general superintendent, planned the head-frame so that the weigh boss handles all the rock as well as the coal.



ROCK DUMP AT GLOBE MINE

When the weigh boss, who stands on the platform a notes a car of rock coming up the shaft he raises the bottom plate b, by means of chain c, passing over the wheel d of the coal chute, which plate turns on an axle at e. The load of rock thus passes into the chute f, of which there are two, one for each compartment of the shaft, and thence directly into the rock car.

The rock car is of the side-dump pattern, with a Λ-shaped hopper bottom. This car has a length of two compartments of the shaft, to save moving, and a consequent capacity of two mine cars, although it may be built of any size that the strength of the trestle will permit.

When the car is loaded, the weigh boss leaves his station at the dumping level a, and admits steam to the small engine g. This engine, by means of a $\frac{1}{2}$-inch wire rope, passing around the sheave h, hauls the car to the end of the dump where the fixed bars i raise the latches j, thus opening the side doors and causing the load of rock to be dumped automatically. Releasing the brake on drum permits the car to return by gravity to its loading position at the shaft.

In addition to saving the labor of at least two men this arrangement is of value in that the cars underground may be gathered by the drivers in their regular rounds and do not have to be shifted out from among the loads. There is thus absolutely no interference with the mine haulage.

Stopping Blocks and Derailing Switches

Written for Mines and Minerals

The majority of the coal mines in Las Animas and Huerfano counties, Colo., are opened on the eastern outcrop of the basin, the seam having a general dip to the west of from 5 to 15 degrees. By reason of this dip and to prevent the possibility of a runaway trip wrecking the slope, numerous ingenious stopping blocks and derailing switches are in use.

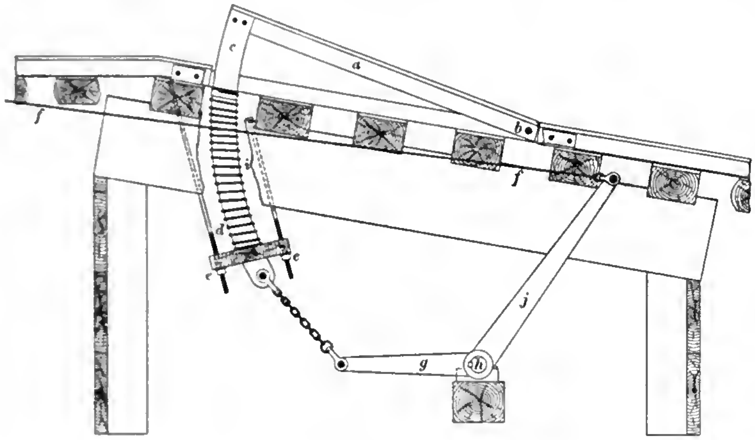
One of the best of these is manufactured under the McClure patents by the Hawkins & Barnet Machine Co., of Trinidad, Colo., and is installed at the Green Cañon mine of the National Fuel Co., near Aguilar. A side elevation is shown in Fig. 1, and the drawing is practically self-explanatory. It consists of a section of a rail a, 4 feet in length pivoted at b. The loaded trip depresses the rail, causing the sword c to descend, thus compressing the spring d. As the wheels of each car of the trip leave the rail, the compression of the spring naturally throws the rail back into position to engage the axle or bumper of the nearest car in event of the trip starting back into the mine for any reason. The tension of the spring is regulated by means of the two nuts at e, it usually requiring a weight of from 100 to 150 pounds to depress the rail a. The drawing shows one of these blocks placed near the knuckle for use on the loaded track.

When used on the empty track, which would be at a lower elevation than in the case shown in the illustration, the blocks are placed sufficiently far from the dump (toward the slope) to accommodate an entire trip. The first car of the trip is run down to the rail a, which acts as a stopping block. When the trip is ready and the rope attached, the dumpman pulls the rod f, which is attached to the rocker arm g, pivoted at h, thus depressing the rail and leaving the trip free to descend the slope. If for any reason the trip parts or the rope becomes detached, releasing the rod f immediately throws up the rail a, which engages the nearest car and stops the trip before a runaway is possible. In some instances the rod (frequently a heavy wire) is affixed to a lever at a switch stand which permits of the rail being locked down, but it is better practice to compel the dumper to hold the lever at the end of the rod until the trip has passed over, so that it may be instantly released in event of an accident. The lever operating the rod f may be set any desired distance from the block; this distance being determined chiefly by the length of the trip it is desired to handle and is not infrequently as much as 200 or more feet.

It is to be remembered that the illustration shows but one-half of the apparatus. There are two rails a, one on each rail of the track, attached to the pivot b, and consequently two

swords c, and two rocker arms g, attached to the axle h. It is apparent if there was but one rail a, say on the right-hand rail of the slope, a trip jumping to the left would not be stopped by the block. With a double rail it is clearly impossible for a trip to run away. Aside from the cheapness of these blocks, they may be installed for less than $100, and the saving in labor of operation, their positive action is to be commended. They are always in place to catch the first or last or any intermediate car of a runaway trip and cannot be put out of action except deliberately by pulling the lever attached to the rod f.

At the Piedmont mine of the Rocky Mountain Fuel Co., near Trinidad, a runaway switch, similar to those used on railroad sidings, is employed to derail the cars in event of the trip getting away. Like the above it is positive in action, the switch always being held open by a spring rail of the familiar type. When the trip is ready to descend the slope the derailing switch is closed by a lever to which is attached a rod or wire. This rod or wire may be made of any desired length, 200 feet or more, so that the switch may be closed by an operator at any point on the tipple. While this device is simple and effective



Stopping Block

and may be made by any mine blacksmith, it is not as efficient as the one first described. A runaway trip is, of course, always ditched and not stopped on the rails as with the McClure block, and further, this device cannot be used as a stopping block against which the empty trip may be made up.

At the Robinson mine of the Colorado Fuel and Iron Co., at Walsenburg, a Mitchell dump is used on the empty track as a stopping block against which the trip is made up. When the trip is ready to descend the slope the horns are opened by a lever attached to a rod or wire of any desired length so that it may be operated from any point on the tipple. Releasing the lever after the trip has passed over closes the horns so that a new trip may be made up against them. In the case in question the apparatus can only be used on the empty track and as a stopping block, as the loaded cars naturally cannot pass through the horns. While not intended as a derailing or stopping device in event of the trip getting away as it is about to descend the slope, nevertheless releasing the lever will throw the horns in, with the natural effect of holding the trip.

THE I. C. S. ANNIVERSARY BANQUET

On the 16th of October, 1911, will occur the 20th anniversary of the enrolment of the first student in the International Correspondence Schools. In order to celebrate the event President Foster has issued orders to his officials all over the world to hold a banquet on that evening in towns where there are headquarters. There will be about 350 banquets in America alone and upwards of 30,000 persons are expected to attend.

AMERICAN INSTITUTE OF MINING ENGINEERS

The one hundredth meeting of the American Institute of Mining Engineers, which was organized in Wilkes-Barre in May, 1871, was held at Glen Summit, near Wilkes-Barre, June 6 to 10.

The One Hundredth Meeting, Held at Glen Summit, Pa., June 6 to 10, 1911

A large number of members were present at the meeting, as well as many guests.

On Tuesday morning, the Mineral Spring Breaker of the Lehigh Valley Coal Co., located at Parsons, Pa., about 2½ miles from Wilkes-Barre, was visited. This breaker was completed May 1, 1911, thus replacing a breaker burned in March, 1910. The frame of the breaker is of steel with concrete foundations, and has a capacity of 1,500 tons of prepared coal per day. One million one hundred sixty-nine thousand pounds of steel were used in the construction, and besides being an absolutely modern breaker, it is noted for special loading systems, consisting of traveling belt and a box-car loader. The pockets being central inside the breaker make a radical departure in breaker design. From the Mineral Spring breaker the members and guests were taken by special train to the Hazard Mfg. Co. Here they inspected the method of making wire ropes and copper wire. The visitors saw in the process of manufacture a cable 5,200 feet long and 2½ inches in diameter of cast steel for No. 2 Ashley plane. This, with the exception of the 2¾-inch diameter rope for No. 1 Ashley plane was said to be the longest wire rope manufactured in the United States. The Ashley plane, which is nearly a mile long, hauls the loaded cars from Ashley, just outside of Wilkes-Barre, to the top of the mountain, and is a method followed by the New Jersey Central Railroad of saving a 12-mile haul of coal and freight from the valley to the top of the mountain. Automobiles were provided by the local owners of the Wyoming Valley Country Club, to convey the members and guests to lunch given by the Country Club.

After lunch the visitors were taken to the Vulcan Iron Works, where they saw mining machinery in the process of construction, also a large drum being made for the Ashley plane.

From the Vulcan Iron Works the visitors were taken to the Wyoming Historical and Geological Society, where they were welcomed by Irvin A. Stearns, one of the five original members of the Institute present, and two papers were read, one by S. D. Warriner on the "Anthracite Conciliation Board," the other by John Birkinbine, on progress made since the Institute was founded. From Wilkes-Barre the return was made to Glen Summit by special train, and in the evening a reception was held.

On Thursday the members took a special train over the New Jersey Central Railroad to Mauch Chunk, and then over the Lehigh Coal & Navigation Co.'s road to Lansford. On the way to Lansford the Hauto storage yard of the Lehigh Coal & Navigation Co. was visited and the method of unloading cars and loading them examined. The standard Dodge type of anthracite storage is used at this place. It consists of four 30,000- and two 60,000-ton piles with two more 60,000-ton piles in process of erection, all the machinery electrically driven.

From the storage pile the visitors went by train to Coaldale, where the new No. 9 breaker of the Lehigh Coal & Navigation Co. is situated. The coal bed at this place is between 30 and 50 feet thick, and as there is some rock, and the pitch is steep, everything broken must come out of the rooms. This requires that the plant be made in two sections. First, the head-house, which is made by the Jeffrey Mfg. Co., under contract, and is entirely fireproof; 250 feet distant is the breaker proper, or second section, where the coal is prepared. After examining this plant the members were taken back to Lansford

and furnished an excellent lunch at the club house of the company. After lunch a number of witty speeches were made, one by W. A. Lathrop, president of the company. He was followed by John Birkinbine and Mr. Dodge. The latter was presented with a loving cup made on the spot by Mr. Storrs, of Scranton.

The Lansford briquetting plant, which is practically the pioneer plant in the briquetting of anthracite for domestic use was visited. Besides a briquetting machine, the plant is notable for the pneumatic cleaning methods in use for reducing the amount of ash in the culm before briquetting. The visitors were taken from the briquetting plant to Summit Hill, where a mine fire started in 1859 in a pump house on the Mammoth bed 60 feet thick. Various efforts were made through half a century to extinguish this fire and cut it off from the main workings of the colliery. The final method, which was successfully adopted, cut off the fire by a clay and concrete wall 725 feet long and 12 feet thick, extending from the surface to the bottom of the coal bed. The cost of this closing off was approximately half a million dollars, in addition to over a million previously spent in unsuccessful attempts. The heat from the fire of the old workings could readily be felt and at times smoke could be seen issuing from cracks in the surface.

From the burning mine the visitors walked to the Summit Hill terminal of the Switchback Railroad, which they took for the return trip to Mauch Chunk. This road was built in the very early days of coal operation to convey coal from Summit Hill to slack water navigation on the Lehigh River at Mauch Chunk. It is about 9 miles long and the open cars from Summit Hill to Mauch Chunk run by gravity through the green woods, which added materially to the enjoyment of the trip. The road is now used only for passengers and tourist traffic.

On reaching Mauch Chunk the members took special train over the New Jersey Central back to Glen Summit, where an evening session was held.

Friday morning the members were taken by special train from Glen Summit to South Bethlehem. Here they were welcomed by President Drinker, of Lehigh University, and conveyed by automobiles to the top of the mountain back of the university, where the beautiful Saucon valley spread out before them. They were then conveyed back to the Fritz Laboratory where the 800,000-ton testing machine was examined. This is probably the largest screw testing machine in the world. From the Fritz laboratory they inspected the Coxe metallurgical laboratory and then were led to the college commons where an excellent lunch was served. After lunch they visited the Bethlehem Steel Works, first the Saucon plant, and afterwards the Lehigh plant. At the first-named plant the open-hearth furnaces were first visited, where 60,000 tons of ingots are made per month; next the Grey mill, where 20-inch Bethlehem girder beams weighing 112 pounds per foot are rolled. The capacity of this mill is 1,500 tons per 24 hours. After this the mill yard and fabricating shops were visited, and then the rail mill, where 100-pound rails were being rolled. The capacity of this mill is 1,500 tons in 24 hours. The armor plate department was next investigated, and the 14,000-ton forging press was seen at work forging armor for the Argentina battleships, Morena and Rivadavia, the 7,000-ton bending press rectifying armor for the Argentina battleships was also seen at work. The No. 2 machine shop was next visited. The dimensions of this shop are 1,522 feet long by 185 feet wide, and in it 4-, 6-, 12-, and 14-inch guns were being made. Next the machine shops were visited, and in No. 4 shop 6-inch guns, gun limbers, and caissons for the United States Army were being manufactured. In the No. 6 machine shop large gas engines for the Trenton Iron Co., Trenton, N. J., and the Consolidated Coal Co., of Fairmont, W. Va., were seen in the process of manufacture; also gas-driven blowing engines for the Bethlehem Steel Co., and pumping engines for the cities of Pittsburg and Detroit.

The return was made to Summit Hill on special train over the Lehigh Valley Railroad.

Among those present were the following: Mr. and Mrs. Ayres, Hazleton; Mrs. Ayres, Bound Brook; Hugh Archbald, Scranton; James Archbald, Jr., Pottsville; G. S. Adams, Washington, D. C.; W. Balz, New York; J. T. Beard, Scranton; E. H. Benjamin and Miss Benjamin, Oakland, Cal.; J. Birkinbine, Mr. and Mrs. J. L. W. Birkinbine, Philadelphia; W. H. Blauvelt, Syracuse; H. Boyd, Hokendauqua; J. C. Bridgman, Wilkes-Barre; J. L. and Alexander Bryden, Scranton; D. Bunting, Wilkes-Barre; Mr. Carpenter, New York; Mr. and Mrs. F. Chase, Wilkes-Barre; Mr. and Mrs. E. T. Conner, Scranton; Mr. and Mrs. T. Coryell, Lambertville, N. J.; Eckley B. Coxe, Jr., Drifton, Pa.; W. R. Crane, State College, Pa.; A. B. Crichton, Johnstown, Pa.; J. S. Cunningham, Johnstown, Pa.; Mr. and Mrs. N. H. Darton, Washington, D. C.; H. G. Davis, Kingston; A. F. Derr, Wilkes-Barre; Mr. and Mrs. E. V. D'Invilliers, Philadelphia; J. M. Dodge, Philadelphia; W. F. Dodge, Wilkes-Barre; Mr. and Mrs. H. S. Drinker, South Bethlehem; E. W. Dwight, Philadelphia, Pa.; A. S. Dwight, New York; Mr. and Mrs. L. O. Emmerich, Hazleton, Pa.; C. Enzian, Wilkes-Barre; Mr. and Mrs. T. N. Eynon, and Miss Eynon, Philadelphia, Pa.; E. B. Edgar, Wilkes-Barre; H. T. Firmstone, Germantown; F. S. Foote, Jr., Urbana, Ill.; Mr. and Mrs. R J. Foster, Scranton; B. E. Fernow, Toronto, Can.; F. A. Gleason, Scranton; William Griffith, Pittston; H. R. Gough, Scranton; Mr. Greslam, New York; Mr. Handy, New York; G. T. Haldeman, Wilkes-Barre; Mr. and Mrs. H. R. Hall, Catasauqua; S. H. Hamilton, New York; N. V. Hansell, New York; H. D. Hibbard, Plainfield N. J.; J. M. Hodge, Big Stone Gap, Va.; Mr. and Mrs. L. Holbrook, New York; J. A. Holmes, Washington, D. C.; O. P. Hood, Houghton, Mich.; Mr. and Mrs. F. W. Iredell, New York; A. B. Jessup, Wilkes-Barre, Pa.; R. W. Johnson, Wilkes-Barre, Pa.; W. Kelley, Vulcan, W. Va.; P. S. King, Philadelphia; H. C. Kirchoff, New York; A. C. LaMonte, Scranton; Mr. and Mrs. W. A. Lathrop, Dorranceton; A. F. Law, Scranton; J. S. Lane, Brooklyn; A. B. Ledoux, New York; W. O. Lentz, Mauch Chunk; Mr. and Mrs. L. F. Lentz, Jr., Mauch Chunk; J. J. Lincoln, Elkhorn, W. Va.; Mr. and Mrs. C. P. Linville, State College, Pa.; J. E. Little, Steelton, Pa.; W. H. Loomis, Jeddo; Mr. and Mrs. E. Ludlow, Eccles, W. Va.; Mr. and Mrs. D. A. Lyle, St. David's, Pa.; J. Lilly, Lambertville, N. J.; J. Lloyd, Wilkes-Barre; Mr. and Mrs. F. J. McMahon, Wilkes-Barre; Miss Merriman, N. Y.; Mr. and Mrs. Mansfield Merriman, New York; Mr. and Mrs. R. V. Norris, Wilkes-Barre; G. L. Olson, Ironwood, Mich.; G. Ormrod, Allentown; J. D. Ormrod, Emaus; W. D. Owens, Pittston; Mr. and Mrs. G. S. Page, Pittsburg; I. P. Pardee, Hazleton; E. W. Parker, Washington, D. C.; K. A. Pauly, Schenectady, N. Y.; Mr. and Mrs. C. P. Perin, New York; Mr. and Mrs. S. M. Pittman and Miss Pittman, Providence, R. I.; Mr. Pettibone, Dorranceton; Mr. Lfordte, Rutherford, N. J.; C. F. Rand, New York; R. W. Raymond, New York; J. W. Richards, South Bethlehem; G. S. Rice, Pittsburg; W. J. Richards and Miss Richards, Pottsville; W. B. Richards, Lansford; D. B. Rushmore, Schenectady; F. A. Saward, New York; Richard Sharpe, Wilkes-Barre; Mr. and Mrs. J. M. Sherrerd, Easton; Mr. and Mrs. A. H. Sherrerd, Scranton; E. H. Shipman, South Bethlehem; Mr. and Mrs. B. Snyder, Jr., Lansford; E. G. Spilsbury and Miss Spilsbury, New Rochelle, N. Y.; F. M. Stark, Wilkes-Barre; I. A. Stearns, Wilkes-Barre; P. Sterling, Wilkes-Barre; G. E. Stevenson, Scranton; D. C. Stearett, Wilkes-Barre; Mr. and Mrs. A. H. Storrs, Scranton; Mr. and Mrs. Oblin Smith, Bridgeton, N. J.; J. H. Smith, Bridgeton, N. J.; H. H. Stoek, Urbana, Ill.; T. W. Stiles, Philadelphia; S. A. Taylor, Pittsburg; Mr. and Mrs. K. Taylor, High Bridge, N. J.; Thomas Thomas, Wilkes-Barre; S. V. Trench, Lansford; B. W. Vallat, Ironwood, Mich.; E. B. Wilson and Miss Wilson, Scranton; Mr. Whitaker, Allentown; Mr. and Mrs. S. D. Warriner, Wilkes-Barre; H. S. Webb, Scranton; Mr. and Mrs. W. G. Whilding, Lansford; E. B. Wagner, Wilkes-Barre; Mr. and Mrs. F. E. Zerby, Wilkes-Barre; D. D. Davis, Kingston.

ELECTRICITY IN MINING

*Written for Mines and Minerals, by Walter C. Wagner**

The first practically successful electric mine haulage locomotive was installed in the Erie colliery, 14 miles from Scranton, Pa., in 1889. It had a single motor, operated by a controller at either end. The great cost of plant equipment at that time prevented a saving in the cost per ton mile, but effected an appreciable economy by increasing the output per day.

Views of a Practical Mine Electrician As to the Requirements of Different Machines

Three types of locomotive drives are now in general use: the standard friction drive, the rack-rail drive, and a combination of these. The first can be used on good track up to 7-per-cent. grade. The rack-rail is used for greater pitches than this. The combination finds its place where tracks of both ordinary and steep gradients are met with. The rack-rail is not popular because of the care necessary to maintain a good road bed and a clean one. Rock falls and heaves make track repairs expensive, while the motors in the locomotives, not having their load limited by slipping of the drivers, sometimes undergo severe mechanical stresses and excessive heating and sparking. Either inside or outside wheels are used, depending upon the size and arrangement of motors with respect to gauge of track and width of entry.

Another classification divides locomotives into the standard, the gathering, and the crab type units. The standard, having a draw-bar at either end, travels along entries having both the trolley wire and the bonded rail return. The gathering locomotive is of a standard type having an auxiliary motor-driven drum carrying a reel of insulated wire. This wire is attached by a hook to the trolley, and permits running into rooms or entry extensions having no trolley wire. The track, however, must be of iron with one rail bonded. In damp entries the insulation of the wire on the drum becomes charged and defective and dangerous to handle. The crab locomotive has an auxiliary motor-driven drum carrying a small haulage rope. This rope is paid out by the motorman when the car runner carries it into a room or slant and attaches it to a mine car. The locomotive then draws the car toward it, and at the same time approaches another room in the entry. By the time the car has reached the locomotive, the switcher is again ready to carry the rope to the car in the second room. Slants may be handled very economically by means of a dead rope and a standard locomotive.

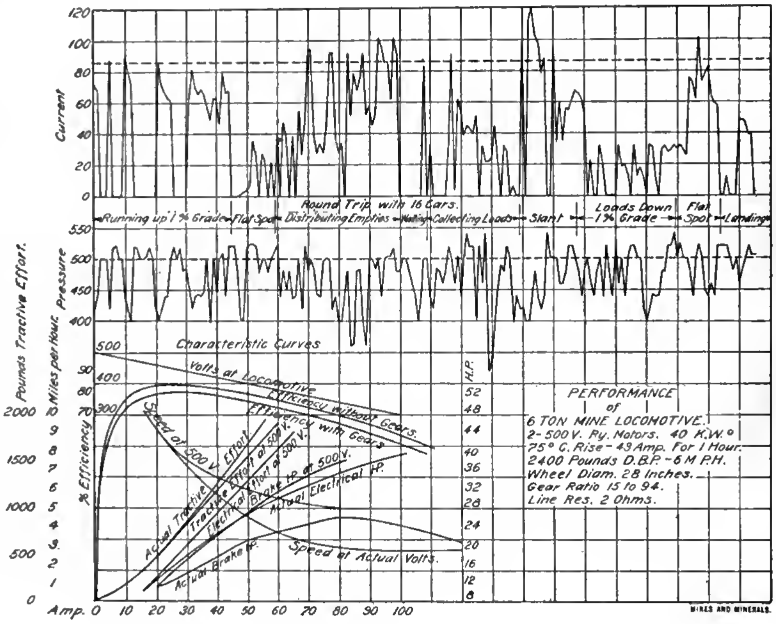


Fig. 1

One to three motors are hung in locomotive frames. They are mounted on the driving axles to permit of vertical movement and make differences of gear meshing impossible. Springs support the other end and lessen the vibration. They may be tandem, outside or inside hung. The tandem is practically standard, the motors hanging on the same side of their respective axles. This permits of a compact and satisfactory arrangement of motors, brake mechanism, and resistances. The outside hung motors are on opposite sides of their respective axles, giving the locomotive a very short wheel base for tracks having sharp curves, but making it unsuitable for high speeds. On larger units the motorman's position is in the center of the locomotive. The inside hung motors have their noses together and have a long wheel base. This type is most satisfactory for high speed service. It is well suited for double-end control, but makes inspection and repairs sometimes difficult; the available space being utilized by the motors, gears, and resistances.

The alternating-current single-phase motor has not as yet entered this field, because of the greater size and cost per horsepower output. This would in many cases, however, be compensated by the increased economy in power transmission.

The gear ratio is somewhat higher than that in the standard outside service, running from an average of 4.6 to 6.3 or more. This, of course, implies a smaller motor, a very important item, a strong starting torque, and a higher running speed. In mine entries the grade is about 1 per cent. in favor of the loads, thus permitting high-speed coasting and a consequently shorter time per trip. This coasting, however, is very hard on the motors and gears. Some practical method of throwing the gears out of mesh would also increase the braking effort of the locomotive. The resistances are generally of the flat iron ribbon type separated by mica or asbestos. The cost of resistance maintenance is excessive, as their current carrying capacity is not yet what it should be. The cast grid type is very efficient on heavy runs where there is a very good track but a great amount of switching.

The size, weight, and speed of locomotives are determined by local conditions. In ordinary mines having small beds of coal and light track and long entries, a locomotive heavier than 6 tons or faster than 6 miles per hour is not advisable. The size of motors and gear ratio, compared with the weight, should be such as to allow a locomotive to put her nose against a rib and grind away for half an hour without a burn-out.†

Series, parallel, and series-parallel controllers are used at one or both ends, and sometimes one controller operates two locomotives. One type of controller is arranged to connect the motors together when the controller handle is on the off position. Any slight difference in speed will cause one motor to act as a generator and tend to drive the other in an opposite direction, thus forming an electric brake. This attachment is not advisable, as it means an almost continuous service, and a consequently greater rise in temperature in the motors. For extreme emergencies the reverse handle may be thrown over and both sets of drivers reversed.

The various forms of good controllers make use of a magnetic blow-out by which the arcing is reduced to a minimum, and use pure copper segments and fingers, or those tipped with copper at the arcing points. Immersing a complete controller in transformer oil was tried. The wear and arcing was greatly reduced but it proved to be very dirty and made inspection

* N. W. Improvement Co., Tacoma, Wash.

† This should be a manufacturers' test, not an operator's.

difficult. Reversal of direction is generally accomplished by reversal of current through the armature, the field being on the ground side. With the severe service met with in mines, an electrical protective device on the locomotive would seem necessary. In practice, however, a circuit breaker or device in a screened fireproof chamber on the feed-line gives more satisfactory results.

It is customary for manufacturers to use but two 250-volt incandescent lamps, in series, for head lights. As the destruction of one lamp means the destruction of both, the use of an additional series lamp of half candlepower and half voltage has been tried. This results in a longer life to the lamps, a good illumination, and the burning out of only one of them at a time. Although the motors are enclosed, most of the troubles result from dirt and moisture. Armature bearings giving good service will, because of dirt, sometimes wear in a few days enough to allow the armature to rub on the pole pieces, tearing off band wires, and perhaps destroying brush holders and field coils as well. Severe sparking due to high mica on commutators has been cured by cutting down the mica about $\frac{1}{8}$ of an inch with broken hack saw blades and water. Mine moisture is splendidly adapted for corrosive action and finds its way into any winding or connection not protected. Coils that have been partly impregnated are more dangerous than coils without treatment. The moisture in the latter often is evaporated, but the former seems to hold it and break down more easily. This makes a regular and careful inspection and testing necessary. Frequent switching, bucking cars along with locked bumpers, running trips over bad tracks and road beds, and using a locomotive beyond the limit of its capacity, mean a service many times more severe than is met with outside. Manufacturers are now realizing this, and equip the same weight locomotives with motors of greater horsepower. Many other points could be remedied if the manufacturers made an effort to keep in closer touch with mine-operating conditions.

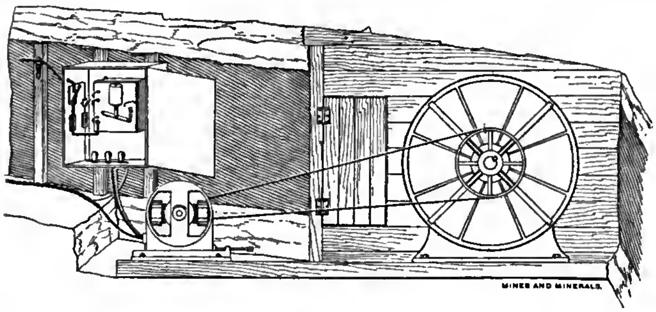


FIG. 2. BOOSTER FAN

On very heavy grades, consideration must be taken of the decreased tractive effort of the locomotive, as well as that necessary to drive itself. In some cases where the locomotive drawbar is somewhat higher than that of the car, a tilting action of the locomotive frame lightens the load on a motor farther from the coupling and increases the load on the other motor correspondingly, causing frequent burn-outs. Then too, when series control is used alone, one motor may cause its drivers to slip; the counter electromotive force of this motor preventing a sufficient current flow to enable the other motor to move under a heavy load.

A typical round-trip performance of a mine locomotive is indicated by curves shown in Fig. 1. This is a 6-ton locomotive with two tandem motors, rated at 6 miles per hour, with an electrical input of 40 kilowatts on a drawbar pull of 2,400 pounds, or 400 pounds per ton.

The starting drawbar pull, or maximum tractive effort, is approximately 150 per cent. of that of 3,600 pounds, with a current consumption of 500 volts of 115 amperes. The ragged current and voltage curves are noted to occur during the whole of the round trip. The current serves also as a direct measure of the torque, which can be computed from the actual characteristic curves.

As the entry often runs on the strike of the coal, one side of the track is ballasted and the other rests on bed rock. This means a poor, uneven track, sometimes dirty from loose coal falling off cars or from rock falls. Then, too, wet spots and small increases of grades are often met with, requiring frequent sanding. Although the couplings are long, the grade keeps them tight and requires a greater accelerative torque than when running on the level. As this entry is one of four in a mine about 2 miles from the power station, and each of them has a locomotive, besides using four electric booster fans and an electric hoist, no further explanation is necessary of the apparently inconsistent variation of voltage. It is typical of mine service. The generators at the power house are not overcompounded, and there is therefore no compensation for line loss. This results in this particular case in quite a drop of pressure when the electric locomotive is pulling heavily; consequently the characteristics obtained at 500 volts can not very well be used for correct calculation at 100 amperes consumption. The changes in such characteristics due to two ohms drop in the feeder line and return are sketched in roughly. The decrease in electrical and brake horsepower is very apparent, resulting in great loss of speed at heavy loads, but increasing the tractive effort at this lower speed. It is interesting to note the enormous difference of performance between an ordinary day trip such as is shown here, and one with very cold weather outside. The empties coming from the outside run with great difficulty because of frozen boxings, and often require more than a 50-per-cent. increase in torque; by the time they are loaded, however, the mine temperature has thawed them sufficiently to make the outgoing trip a fairly normal one.

Where the inertia of the trip of cars at starting, or increased resistance due to a dirty track, causes the locomotive to pull heavily and run slowly, the effect of a long feed line causing an appreciable voltage drop at the locomotive is of interest. The drop of course is proportional to the current consumption.

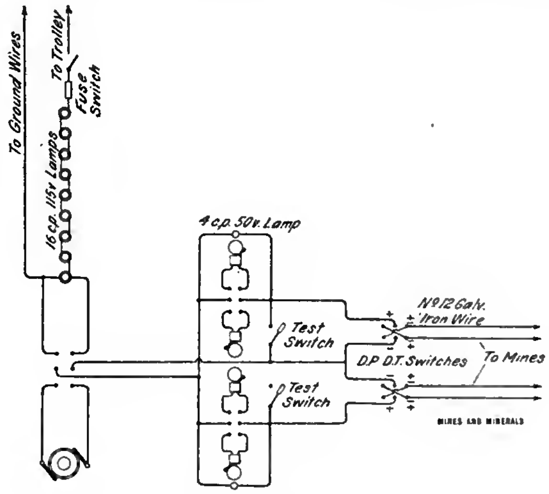


FIG. 3. SIGNALING ARRANGEMENT

The motor torque is proportional to the current only, and is independent of the impressed electromotive force. With a heavy pull, the motors' speed need not be so great; as an increase in current flow means less speed for the necessary current electromotive force. As the applied electromotive force decreases (due to line loss) and the current electromotive force increases with the greater current flow, the torque, and therefore the current, does not increase as in the case of constant applied voltage. There is therefore less danger of overloading the motors. As the tractive effort also is higher at lower speeds, such line resistance allows of larger trips to be started by a certain weight locomotive without an excessive output at

the power house. As the trip resistance is decreased after starting, the speed rises with the decreased current and increased applied electromotive force making up for time lost on the heavy pull when starting.

Another advantage of a long feed-line is decreased harmfulness of short circuits. A short circuit on this 2-mile line will blow the circuit breaker without causing a violent flashing of the generator brushes or excessive arcing at the contacts of the circuit breaker, since the current cannot exceed 300 amperes.

This excessive electromotive force variation is exceedingly hard on the shunt motors driving ventilating fans, and is the cause of a great portion of the trouble with them.

These motors are usually belted to the generating units, although in some cases they are direct driven. A belt drive permits of easy inspection and cleaning. The motor is less dirty and can be replaced by another motor with different speed by changing the size of the pulley. Blasting, causing a sudden stoppage of the fan, simply means the slipping of a belt having guides and not a burn-out. The motors may be of high speed and have a correspondingly less weight and volume to a given horsepower.

In places where a burn-out or stoppage of one motor would be dangerous, a relay is installed with its pulley just above or below the working motor, thus allowing the use of the same belt by either machine, with a minimum of time for changing over.

For 5 and 10 horsepower motors it has been found advisable to place a series winding over the shunt fields, heavily over-compounding them. This does away with the starting box and its attendant troubles. The present type of automatic starting devices are not sufficiently protected to withstand the moist mine atmosphere and the attendant electrolysis.

Mine fans are placed at the mine openings and also along air-courses inside. The former, often termed ventilators, usually handle the entire air flow of a mine. The latter, called boosters, are placed in the air-courses to assist and control the flow of air through the workings.

Electrically driven fans are of either the centrifugal or disk type. The centrifugal, as the name implies, is built somewhat like a waterwheel, having paddles or blades parallel to the axle of the shaft and forcing out the air at right angles to it. This type is in general use at mine openings. It is not efficient in booster service, as it drives the air against the rib, requiring it to make two turns of 90 degrees, and creates a whirl before resuming its natural flow along the air-course.

The disk fan, having radial blades set at an angle to the axis of the shaft, is better adapted for general inside mine ventilation, as the air is driven parallel to the rib. It is built in smaller sizes and used in smaller workings or in booster service. A typical booster fan installation, one of a series along the air-courses of an extensive mine, is shown in Fig. 2.

The fan, with its surrounding air-tight stopping, is placed directly across the air-course. The air is being driven toward the observer, as the door is built to swing in this direction. With the fan in operation the freely swinging door is closed by the excess pressure on this side of the stopping. If for any reason the fan should cease to operate, the flow of air, due to the other fans, would open this door. This open area, together with the spaces between the fan blades, guarantees a continuous flow of air through the air-course.

The motor is placed as far from the fan as possible to give a good drive without a tight belt. Sometimes a small room is cut out of the rib, or an old slant is utilized for this purpose. The feed-line is shown coming in on props set against the rib. The ground return is often an old steel haulage rope running along the entry to a main copper return outside. The automatic starter is shown in an asbestos-lined wood box having a sheet-iron door. The incandescent lamps keep the starter, fuses, and switches in a dry insulated condition. A change in wiring has been made since the time of this sketch, bringing the high or trolley wire through the main switch and fuse directly to the line terminal of the motor. This brings all the live parts of the starter to a ground potential when running, making it safer for the attendants.

There is a maximum demand in ventilation as well as in pumping. As the mine air is at nearly a uniform temperature the whole year, the outside temperature variation either aids or retards the flow of air. If the intake is below the level of the discharge the warmer weather will resist the flow; if colder, the weather assists the fans, and in some smaller mines makes them unnecessary. In some cases an economy is effected by a reversal of the fans with a decided change of temperature.

Signaling service in mining work must be absolutely reliable in the face of the most adverse conditions. Primary batteries are used where no other electric power is available. They require frequent inspection and renewals, but give good results where the bells are not in continuous service. On long inclines where no stops are made between terminals magnetos or telephones are used. This is not good practice, as the trip rider has no means of signaling in case of trouble. With electric power at hand, the arrangement shown in Fig. 3 has proven to be very satisfactory. A 1.5 horsepower 50-volt generator is driven from the shaft of a ventilating fan. A bank of 16-candlepower 115-volt lamps in series is used as a relay, the low voltage power being taken from the ground lamp of the series. It has been found advisable to replace this with two lamps in multiple to insure against the breaking of a filament. Each slope has a separate signal board in front of the engineer, upon which are mounted the set of two bells, two switches, and a 50-volt signal lamp. The single-pole switch is used to test the bells. The double-pole, double-throw switch allows either bell to be used. The lamp flashes up whenever a signal is given insuring absolute reliability. It is also a good indicator of the condition of the line in the case of short circuits and grounds on the mine wires. The double-throw double-pole switches leading to the iron signal wire are used in case one wire grounds in the mine and rings the bell continuously. Throwing over the switch connects the clear wire to the other side and keeps the signals working until the mine stops for the day and allows repairs to be made. In some installations a battery is placed as a relay for the lamp bank power supply. The direct-current vibrating bell has one weak point, the contact. In ordinary mining service the contacts are allowed to be eaten away and then trouble due to sticking or poor contact results. The continued vibration, together with a flow of current in one direction in the coils, often produces a polarized core with an appreciable residual magnetism, causing the armature to stick to the core. Breakage of the spring holding the vibrating contact is very common in hard service. A cure for these coils would be the use of a small alternating-current generator in the place of the direct-current one and the use of alternating-current ringers. The emergency relay would be from the direct-current supply with direct-current bells.

It is necessary in handling a large field to keep in intimate touch with all phases of the work, its performance with respect to thoroughness and efficiency, and the cost of operation. The group of reports and cards needs but little comment. The Motor Report is very useful; the date of trouble and the date of repairs, together with the time required, give a complete knowledge of daily performance. The Daily Report Electrical Department gives a summary of costs divided according to the classes of work. In this way one can observe the proportionate amount of attention given by each electrician in his mine to each division of work.

The Daily Station Report shows graphically the run of the generating units for the 24 hours. Current and voltage readings, together with their maximum values, give an accurate distribution for power charges against the different mines.

The Electric Light Department sheet gives an idea of the system used to handle the light revenue account. When the

work is completed the material list is filled out together with the amount of time. This is then approved by the electrical engineer and turned in to the accountant's office for collection.

A card-index system is kept of the different units. This enables duplicate repair parts to be obtained without going into the mines, and also facilitates more economical arrangements of the units. Tests of load voltage, etc., are placed on the back of the card.

Cost data are kept in a similar manner, each job being noted as completed.

A NEW BREATHING APPARATUS

Written for Mines and Minerals

Since the introduction of recovery apparatus in the United States there has been a decided feeling among mining men that something less cumbersome, less complicated, and more dependable was needed. From the outset it was evident that to do effective work the operator must not be incumbered by weight or be compelled to breathe rarefied air. Further, that he must not be troubled or made anxious because his life depended upon a system of complicated reducing valves, pressure gauges, high-pressure hose, or other intricate parts liable to become deranged.

Seeing the weaknesses in the first recovery apparatus, one was invented, which, however, did not comply with all the requirements, but being confident that he had the right principle for the foundation of a satisfactory rescue apparatus, the inventor persevered until he produced the apparatus here described. It is in two styles, the larger called the Dreadnaught as shown in Figs. 3 and 4 and a lighter "Emergency" apparatus shown in Figs. 1 and 2. The first demonstration convinced the writer that the "Emergency" form was a success and the second demonstration, before the district superintendents and the mining engineers of the Delaware, Lackawanna & Western Railroad Co., further convinced him that both were successes.

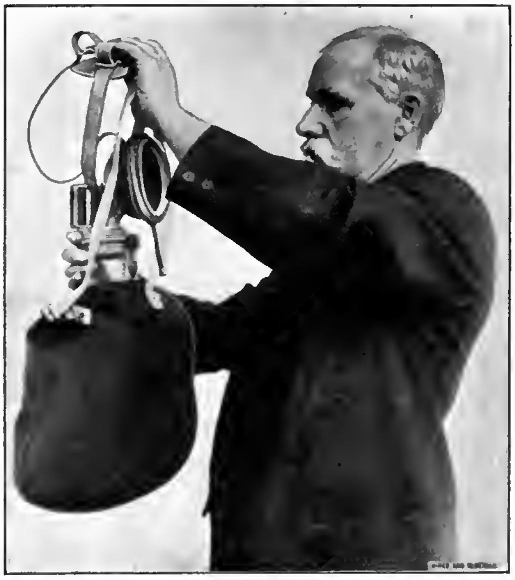


Fig. 1

Those present saw an operator equipped with a Dreadnaught enter a room filled with ammonia fumes, and then one of their own number enter with the Emergency apparatus and remain 18 minutes. It is not probable that a man unprotected could have remained 2 minutes in such an atmosphere, yet the Dreadnaught operator remained three-quarters of an hour and lifted iron billets about the room without inconvenience. There is no question but that he could have stayed there over an hour longer, but 15 minutes in such an atmosphere was more than was necessary to convince the observers of the value of the apparatus.

The Emergency apparatus weighs 4 pounds when charged and is carried on the head and neck in such a way as to cause no discomfort, and at the same time allows the use of both hands. It carries a charge of oxodon, a chemical oxygen compound manufactured by the electrolytic process, which releases oxygen gas and removes the carbon dioxide from the exhalation. One charge in this little apparatus is sufficient to sustain a man walking fast 1 hour, and owing to the light weight of the apparatus several can be carried at one time into the mine in case of an explosion or fire and dealt out to any men who have been walled off by gas or smoke. As the face of the operator is not entirely covered he is in no danger of becoming overheated, as is possible with the wearer of a helmet. One user of the helmet apparatus stated that after 20 minutes the operators lose their efficiency when working, for which reason, in fire fighting, shifts are reduced to 20 minutes each. The Emergency would permit men to retain their efficiency for 1-hour shifts, if necessary. The oxodon, which is hermetically sealed in cans, will remain active a long time, provided no moisture reaches it, thus making it practical to store several Emergency apparatuses at strategic places in mines where they could be readily reached by miners in case of disaster.

Fig. 2

The Dreadnaught is a much larger apparatus and weighs 20 pounds when charged with oxodon in quantities to last 3 hours. This apparatus has features which are worthy of consideration. In the first place the weight is distributed on the back and chest, making it self-contained, as the machinist says. The oxodon is carried in a tight bag in front while the water necessary for the generation of oxygen is carried in a box on the back. When the charge is placed in the bag and the water turned on, the chemical immediately generates oxygen. A mask which covers the nose and mouth is arranged to fit any face without adjustment. At the same time it is held in place by straps so that it cannot readily come off. This mask is connected with two rubber tubes leading into the generating bag, and since the liberation of oxygen depends on the moisture admitted to the chemical, increased respiration insures increased supply of oxygen to meet the demand of the lungs.

The advantages of the Dreadnaught are as follows:

The weight is 20 pounds, less than one-half the weight of the apparatus used by the Bureau of Mines, and is about equally distributed front and back, while in other types the weight is almost all suspended on the back. There are no high-pressure cylinders, reducing valves, pressure gauges, high-pressure hose, or other intricate parts, thereby reducing the chances of the apparatus getting out of order while in service. There is no expensive oxygen pump to transport and keep in order, neither are oxygen tanks required to be carried with the apparatus. The Dreadnaught is easily portable. One can carry an apparatus and enough cartridges for 12 hours' work in a case, and so save valuable time, expressage, and delays at critical times. The capacity of one charge is 3 hours, and this

is about 1 hour in excess of the best other apparatus. The cost of operation is low—about 85 cents per hour. Owing to the simplicity of the apparatus less training is needed for successful operation and the operator can always determine the condition of his charge without the aid of a gauge. In fact, with a watch he is safer alone than are two men with pressure gauges on their backs.

Fig. 3

Another feature which adds materially to the usefulness of this apparatus is that by adding a 6½-pound storage battery to the back piece a strong electric light may be had. This light is attached to the front of the device, but is readily detached and by means of flexible wires can be used as a hand light as shown in Fig. 4. Although the light increases the weight of the Dreadnaught to 26½ pounds, it is a most desirable acquisition, because the operator would carry an electric light probably weighing as much as 5½ pounds in his hands, while in this case he has both hands free, and a strong light always thrown ahead of him.

Another good feature in this new apparatus is that the heat due to having the neck and face covered by the helmet is greatly lessened by the mask used, and this will be appreciated. The cost of the apparatus is comparatively low, and duplicate parts are exceedingly cheap compared with other apparatus, besides there are few parts that get out of order.

The makers are the Servus Rescue Equipment Co., of Newark, N. J.

METHOD OF INSTALLING FANS

Written for Mines and Minerals, by Thos. W. Fitch, Jr., and Jay R. McColl, M. E.

The most general practice in the use of large fans is to install concrete or brick foundations up to the center line of the fan. The shaft carrying the fan wheel is usually supported on bridges or cast-iron standards at the inlet circle, and steel housing is provided above the center line of the fan; also steel stack. The drifts at the side of the fan with roofs over them are usually built of brick or concrete. In some cases, fan manufacturers build these side drifts of steel.

The use of self-contained full-housed steel fans has been rare heretofore, but with the introduction of the "Sirocco" fan they have come into more general use on account of the smaller diameters used. With this fan the ratio of inlet diameter to the diameter of the fan is 80 per cent., whereas with previous types 50 per cent. to 60 per cent. was the maximum.

Another prominent feature introduced with the "Sirocco" fan is that the clear area of inlet exceeds 90 per cent. of the calculated area, whereas with fans having arms, spiders, tee or angle irons on the backs of the vanes to form obstructions in the inlet, the clear area of inlet is reduced from 60 per cent. to 80 per cent. of the calculated area. From this it will be noted that with the same area of inlet as fans of previous types, the diameters of "Sirocco" fans are very much less, allowing of the use of full steel-plate housings with far less vibration when in operation, and of greater stability, due to the lesser distance of the bearings above the foundation.

The general method of driving mine fans is by direct-connected steam engines, and this has had a great influence in limiting the speed of fans in use. In the anthracite fields of Pennsylvania it has been usual to limit the speed to 75 to 90 revolutions per minute. These extremely slow speeds are not as necessary with modern methods of fan manufacture as was formerly the case, and it is obvious that old style unwieldy fans, revolving at 70 to 90 revolutions per minute on partly true shafts are not and cannot be mechanically equal to smaller and less weighty fans running about 150 revolutions per minute.

In the bituminous coal fields, direct-connected units are operated at from 125 to 175 revolutions per minute, depending upon the type of engine and the size of the fan. As mine operators have generally found that there is more or less trouble to be had with belts, there is a very strong prejudice against indirect drives. However, they will come into more general use with the tendency to use smaller high-speed fans and with the use of electric motors for driving fans. The particular advantage in steam-driven fans is that the fan can be run 24 hours a day, with Sundays and holidays included, without regard to the operation of any other apparatus, but if the fan is electrically driven it becomes necessary to run the generator continuously, which is not always possible or desirable.

Gear drives have been found inapplicable to fan drives, because owing to shot firing in the mine and the opening and closing of doors, sudden slight changes in the speed of the fan take place, causing heavy backlashes. In England and Germany where a great many high-speed fans are installed, the popular drive is the English system of ropes. That is, a number of separate ropes. Fans direct-connected to motors have not met with large success in this country up to the present time, largely for the reason that the size of the motor required is too great. With many types of fans, if sufficient area of inlet is provided, it makes the fan of such large diameter that it cannot run at a very high number of revolutions, and if a direct-connected motor is used it must be very large to develop the necessary power at the slow speed. In this connection, the

Fig. 4

"Sirocco" fan has an advantage, as with the same area of inlet the fan is much smaller in diameter and can, therefore, run at a much higher speed. In both the case of direct-connected engines and direct-connected motors, it is to be noted that there is a great advantage in their favor, viz., the loss of power in an indirect drive would ordinarily amount to about 10 per cent. of the power delivered to the fan; and the possibility of trouble with an indirect drive is eliminated by direct connecting.

THE NEW ILLINOIS MINING LAW

Written for Mines and Minerals

The legislature of Illinois at the session recently adjourned, passed several mining bills changing the mining law of the state that has been in force for a number of years.

Practically the Law Recommended by the Mining Investigation Commission

These bills were presented to the legislature as the result of the work of the Mining Investigation Commission authorized by the Forty-Sixth General Assembly. This commission was composed of three coal operators, three coal miners and three disinterested persons who had no connection with the coal industry of the state. The laws as passed by the legislature are practically those recommended by the commission, with two exceptions, which have to do with the inspection service rather than directly with the safety features of the law. The report of the commission recommended an increase of salary for the State Mine Inspectors from $1,800 to $3,000. This was not allowed. The report also suggested the substitution of deputy inspectors instead of county inspectors but this was not agreed to by the legislature, and Illinois still has the dual system of inspection consisting of state inspectors appointed by the governor, and county inspectors appointed by the county boards.

The essential features of the new law are the following: The general direction of mining matters is placed in the hands of the State Mining Board, consisting of two practicing coal miners, one practicing coal-mining hoisting engineer, and two coal operators. Under the previous law the board was similarly constituted, but it was mainly a board for examining applicants for state certificates. The new law gives the board specific authority and direct supervision over the inspection service and the mining practice of the state for a period of two years, the president must be a coal operator and the secretary a coal miner. The compensation of the members of the board is five ($5) dollars per day and the number of days of service per annum is limited to 100. In addition to the supervision of mining matters, the gathering of statistics and the publication of a coal report is placed under the board. A chief clerk to the board at a salary of $2,000 per annum is provided.

The requirements in regard to qualifications of and examinations for inspectors, mine managers, mine examiners and hoisting engineers are practically the same as in the previous law, excepting that a knowledge of rescue work and first-aid work is required upon the part of mine inspectors and mine managers. It is also a new requirement that all examination papers shall be kept on file, and the applicant may secure a copy of his paper within a reasonable time by paying a suitable copy fee. The requirements for the employment of certificated managers, examiners, and hoisting engineers, are essentially the same as under the old law. The number of mine inspectors has been increased from 10 to 12 and provision made so that inspectors shall have a better allowance for expenses than has been heretofore possible. This is a much-needed improvement, as heretofore not only have the salaries been too small but the allowance for expenses has been so meager that it was frequently exhausted before the end of the fiscal year and any subsequent inspection had to be done at the personal expense of the inspector. Definite provision has also been incorporated in the law authorizing and directing the State Mining Board to furnish each inspector with the necessary instruments needed in carrying out his work.

Every mine in which marsh gas has been detected in quantities, which in the judgment of the mining board is dangerous, must be examined at least once in six months by the state inspector. The state board may require the state inspectors to examine any and all mines when they deem it necessary. Inspectors are required to measure the air in the last cross-cut in each pair of entries and in the last room of each division in longwall mines and must supply a written report of each inspection to the State Mining Board.

As heretofore, the inspectors continue to be sealers of weights, although it seems to be an unnecessary burden and one that is in no way connected with the safety of the miners.

The requirements in regard to maps are essentially as heretofore, excepting that the data on maps must now include the location and depth of all holes drilled for oil, gas, or water, that penetrate a workable coal seam. In order that the engineer of the coal company may have the data necessary for carrying out this provision, a separate law was enacted requiring the driller of every such hole to file with the county clerk of the county in which the hole is drilled a description and map showing the location of every such hole, which map and description become a part of the title record of the tract of land.

Mines to be abandoned or indefinitely closed are required to be mapped before such abandonment. The state inspector of mines may order additional maps to those required by law, if he has any reason to believe that the maps furnished him are inaccurate he may have another map made at the expense of the operator.

The new law has definitely placed the operation of shaft sinking under the inspection service and requires a safe and substantial structure to support the head-sheaves placed at a height of not less than 15 feet above the landing place.

A platform must also be arranged to prevent material from falling into the shaft. No material can be hoisted from the bottom, when men are in the shaft, excepting in a bucket or on a cage. Provision is made for properly attaching a rope to the engine drum, and for providing the engine with a brake. Not more than four persons can be lowered or hoisted in the bucket at one time and it is unlawful to ride on a loaded bucket. In sinking, all shots must be fired by electricity and the hoisting engineer must be a certificated man.

The second shaft, commonly known as the escapement shaft, shall be not less than 500 feet nor more than 2,000 feet from the hoisting shaft, excepting that in mines employing 10 men or less this distance may not be less than 250 feet. The escapement shaft, if equipped with a cage for hoisting men, must conform to all of the requirements of the hoisting shaft in reference to the hoisting and lowering of men.

Several material changes have been made in the regulations about the shaft bottom and provision is now made for a passageway from one side of the cage to the other, "free from obstruction and dry as possible and not less than 3 feet wide and for the use of men only; animals or cars shall not be taken through such passageway while the men are passing or desirous of passing through such passageway."

A refuge place or places for men coming out at the close of the day's work shall be provided off the main bottom, at such place or places and of such size as approved by the mine inspector. These places shall be not more than 400 feet from the hoisting shaft.

No inflammable structure can be erected or reerected on the surface within 100 feet of any hoisting shaft or escapement shaft excepting in mines employing 10 men or less, and no oil or similarly inflammable materials can be stored within 100 feet of any hoisting or escapement shaft or within a mine.

The regulations in regard to safety lamps have been considerably extended and at every mine in the state there must now be kept at least two safety lamps and as many more as may be required by the state mine inspector. The regulations for the use of the safety lamps are also quite stringent. No man is allowed to use one until he has shown that he understands the proper use thereof and the danger of tampering with it. In any mine where locked safety lamps are used it is made a misdemeanor for an unauthorized person to have in his possession any means for unlocking a safety lamp or any matches or other means of producing a fire.

To reduce the danger from excessive shooting in closed

places the following provision is made: "When undercut or sheared, the entry, cross-cut and room neck may be advanced concurrently, but not more than one cutting shall be shot in the room neck until the cross-cut is finished; and after the entry has advanced 15 feet beyond the location of the new cross-cut, only one shot shall be fired in the entry to two in either or both the cross-cut and room neck at the same shooting time.

"When not undercut or sheared, the entry and cross-cut may be advanced concurrently, but no room shall be opened in advance of the last open cross-cut, and after the entry has advanced 15 feet beyond the location of a new cross-cut only one shot shall be fired in the entry to two in the cross-cut at the same shooting time.

"Not more than three shots shall be exploded at one shooting time ahead of the last open cross-cut."

Provision is made for staggering the cross-cuts between rooms.

The inspector is required if he finds men working without the proper amount of air to notify the mine manager to increase the amount of air, and in case of the refusal of the manager to act properly and in all cases where men are endangered by such lack of air, the inspector shall at once order the men affected out of the mine.

Provision is made for keeping the roads clean so as not to endanger the drivers in jumping off their trips.

The regulations in regard to illuminating oils are completely changed and all illuminating oils are required to conform to specifications provided by the mining board. All oil must be stamped or branded upon the original package, showing that it has been tested and found to conform to the specifications as prescribed by the mining board. Any person selling or offering for sale oil not complying with the specifications, and any owner, operator or employe, who knowingly uses in a mine or who knowingly permits to be used any oil forbidden by the act, is guilty of a misdemeanor. The state mine inspectors are given authority to sample all oil used for illuminating purposes or kept on sale for use and it is made their duty to send to the State Mining Board samples of any oil that they have reason to suspect does not conform to these standards or specifications.

The amount of powder kept by any one workman is increased from 25 to 35 pounds. The reason of the change being made is that according to the old law, if a miner had a part of a keg of powder left at night, or not enough to run him through the following day, he was prohibited from bringing in a new keg until after the part of the keg has been used.

The following regulations in regard to dead holes have been incorporated:

"No person shall drill or shoot a dead hole as hereinafter defined. A 'dead hole' is a hole where the width of the shot at the point measured at right angles to the line of the hole is so great that the heel is not of sufficient strength to at least balance the resistance at the point. The heel means that part of the shot which lies outside of the powder.

"In solid shooting, the width of the shot at the point, in seams of coal 6 feet or less in height, shall not be greater than the height of the coal, and in seams of coal more than 6 feet in thickness the width of the shot at the point shall, in no case, be more than 6 feet.

"In undercut coal, no hole shall be drilled 'on the solid' for any part of its length."

Mixed shots are prohibited. Holes are required to be tamped full and no coal dust or inflammable material or any material that can create a spark can be used for tamping Missed shots cannot be withdrawn excepting with the use of copper tipped or wooden tools.

The duties of the mine manager, mine examiner, hoisting engineer, are essentially the same as in the previous law, but the wording has been changed so as to be specific, and an effort has been made to avoid ambiguity so that duties cannot be shirked through a lack of definite provisions in the law.

The special rules have been made somewhat more stringent. It has been made a misdemeanor to enter or work in or about a mine or mining building, tracks or machinery connected therewith, while under the influence of intoxicants.

A miner is required to sound and examine the roof of his working place before commencing work, and if he finds loose rock or other dangerous conditions he is forbidden to work in such room under such conditions excepting he make the room safe. No one is allowed to enter or leave a mine without indicating the fact of entering or leaving by some suitable checking system. It is made a misdemeanor to change or alter in any way any car check.

Provision is made for keeping bore holes in advance when abandoned workings are approached. In case there is a map of the old workings, bore holes shall be kept within 50 feet of the old works, such bore holes being 10 feet in depth, one being in the face and one in each rib. The advance working shall not be more than 20 feet wide. If there is no map of the old workings, bore holes will be maintained from the time the new workings reach the point within 100 feet of where the old workings are supposed to be.

By separate act the use of powder has been regulated and the following specifications have been provided for black powder:

"(*a*) It shall have a specific gravity of not less than 1.74 nor more than 1.90.

"(*b*) It shall have a moisture content of not to exceed 1 per cent. at the time when shipped by the manufacturer or his agent.

"(*c*) Said powder shall be sold for use in coal mines only in seven sizes of granulation to be determined as follows:

"CCC shall be powder which shall pass through a screen having round-hole perforations of $\frac{40}{64}$ of an inch in diameter and remain on a screen having round-hole perforations of $\frac{32}{64}$ of an inch in diameter.

"CC shall be powder which shall pass through a screen having round-hole perforations of $\frac{36}{64}$ of an inch in diameter and remain on a screen having round-hole perforations of $\frac{24}{64}$ of an inch in diameter.

"C shall be powder which shall pass through a screen having round-hole perforations of $\frac{27}{64}$ of an inch in diameter and remain on a screen having round-hole perforations of $\frac{18}{64}$ of an inch in diameter.

"F shall be powder which shall pass through a screen having round-hole perforations of $\frac{20}{64}$ of an inch in diameter and remain on a screen having round-hole perforations of $\frac{12}{64}$ of an inch in diameter.

"FF shall be powder which shall pass through a screen having round-hole perforations of $\frac{14}{64}$ of an inch in diameter and remain on a screen having round-hole perforations of $\frac{7}{64}$ of an inch in diameter.

"FFF shall be powder which shall pass through a screen having round-hole perforations of $\frac{9}{64}$ of an inch in diameter and remain on a screen having round-hole perforations $\frac{1}{64}$ of an inch in diameter.

"FFFF shall be powder which shall pass through a screen having round-hole perforations of $\frac{5}{64}$ of an inch in diameter and remain on a screen having round-hole perforations of $\frac{2}{64}$ of an inch in diameter.

"In testing powder for size of granulation as herein required, it shall be permissible for a given size to contain not to exceed $7\frac{1}{2}$ per cent. of weight of grains of the size next larger and $7\frac{1}{2}$ by weight of grains of the size next smaller."

All black powder must have plainly stamped on the case the letters showing the sizes of granulation as given above. Failure to conform with these requirements is made a misdemeanor punishable by a fine of not less than $100, or imprisonment for 90 days or both. The state inspector is authorized to sample blasting powder for granulation in conformity with

the sizes given above, and for other tests he is authorized to sample powder and send the same to the State Mining Board.

A separate act prohibits the drilling of oil or gas wells nearer than 250 feet to any opening of a mine used as a mine's egress for persons employed therein or as an air-shaft. As noted above, it is incumbent on the driller of the oil or gas well to file a record of the depth in the county clerk's office so that it may become a part of the title of that property.

Before the casing shall be drawn from any well the hole must be securely stopped or blocked.

The act requiring fire-fighting equipment, passed by the special session of the legislature which convened after the Cherry disaster, has been amended so as to do away with the gongs near the face of the workings. The fire-fighting equipment required of the small mines has also been reduced by eliminating the requirements for chemical fire-fighting extinguishers in mines employing less than 10 men. The provision in regard to alarm gongs has been changed, since it seems to be the opinion of both operators and miners that such gongs are objectionable and are probably causes of panic rather than an efficient means of warning. To increase the means for warning, however, additional telephones are now required.

The law in regard to the mine-rescue stations remains practically unchanged excepting with a few verbal amendments to permit the commission to more efficiently operate the rescue cars which were not provided in the original act of incorporation. The recommendation of the Rescue Commission for increased salary for the assistants at the stations was not agreed to by the legislature. An appropriation of $30,000 per annum was made for the support of the stations.

A bill authorizing the University of Illinois to establish miners' and mechanics' institutes throughout the state was passed but will not be operative because no money was appropriated to carry out the provisions of the act. An act was also passed continuing the Mining Investigation Commission, or rather providing for the appointment of another commission along the lines identical with those upon which the previous commission acted, the previous commission having gone out of existence when it filed its report to the governor and legislature.

In the appropriations for the University of Illinois, the allowance for maintenance of the mining department given by the previous session of the legislature was doubled and is now $7,500 per annum. An item of $5,000 per annum was appropriated calling for cooperative investigations between the United States Bureau of Mines, the Illinois Geological Survey, and the Department of Mining Engineering. An appropriation of $25,000 was also made for equipment for the Department of Mining Engineering, and an appropriation of $200,000 was made for a new engineering building in which the Department of Mining Engineering will have quarters.

INDIA'S MINERAL RESOURCES

The petroleum production of India and Burma in 1897 was only 19,000,000 gallons; in 1908 it had increased to 176,500,000 gallons, and in 1909 amounted to 233,500,000 gallons.

It is well known that India possesses enormous untapped supplies of iron and copper, which only require the necessary enterprise and capital for their exploitation. The Tata Iron and Steel Co. is about to introduce a new era into the mineral and metallurgical life of India, and the time is apparently not far off when this country will be in a position to smelt its own ores with its own coal and fluxes, and not only produce sufficient to supply its own needs, but to export the surplus to other lands. The Tata iron and steel plant is really an American conception and is being pushed to completion under the supervision of American engineers and mechanics and will be equipped largely with machinery made in the United States.

ALABAMA INSPECTORS' WARNING CARD

James Hillhouse, Chief Mine Inspector of Alabama, and his associates, Robert Neill and J. W. Dickinson, got up the following warning card last November and had it posted in conspicuous places at the coal mines. It was so instrumental in diminishing accidents that the consuls and vice-consuls of Austria-Hungary, Russia, and Italy have adopted it for translation, publication, and distribution among miners of those nationalities in other states.

WARNING!

TO ALL MINE EMPLOYES

The subject of mine accidents is attracting a great deal of attention throughout this state. From the loss of life incurred thereby, the suffering cannot be measured. The desolation and want that follow the killed or crippled bread winners call forth much sympathy. In some of these accidents the danger is unforeseen, but a large majority could be avoided if the necessary precautions are taken. This office suggests the following:

That every coal mine worker familiarize himself with the mining laws which are posted at all coal mines.

THE MINER

Don't forget to sound the roof on entering your place to begin a day's work.

Don't forget to sound the roof after each blast.

Don't use short fuse, "skinner backs."

Don't use two different explosives in same hole.

Don't use coal for tamping.

Don't conclude the roof is safe in spite of drummy sound.

Don't take lighted pipe or lamp to your powder box.

Don't fire two holes at the same time.

Don't bore your holes beyond the mining.

Don't hurry in order to get out early.

Don't risk your life to save labor.

Don't forget the miner is responsible for the safety of the laborer.

THE LABORER

Don't go into working place until the miner has examined it and pronounced it safe.

Don't fire blasts for the miner nor in his absence.

Don't disregard the orders of the miner.

Don't forget to retreat to place of safety when blasting.

DRIVERS

Don't ride between cars.

Don't ride on side of cars.

Don't run cars on grades until you know the road is clear below.

Don't forget to call driver boss' attention to bad track.

Don't take door boy away from his post to drive your mule.

DOOR BOY

Don't leave your door.

Don't allow your door to remain open longer than necessary.

Don't play around or run after trip of cars.

ALL EMPLOYES

Don't ride up and down the slope.

Don't pass over danger signals.

Don't roam through old workings.

Don't travel the slope; take the manway.

Don't smuggle dynamite in mines; and report any one you see doing it.

THE MINE FOREMAN

See that every working place is free of gas before employe starts work.

See that all apparatus affecting safety of employes is in good and safe condition.

See that the above suggestions are enforced.

THE TESTING OF MINERS' OIL

*By C. E. Scott, Chief of Testing Department, Fairmont Coal Co.**

In the West Virginia State Mining Laws, there is an act which reads in part as follows: "That only animal, vegetable, or paraffin oil, or other oil as free from the evolution of smoke as a standard cottonseed oil, when burned in a miner's lamp shall be used in any open lamp or torch for illuminating purposes in any coal mine in this state and that kerosene or blackstrap oil or a mixture of kerosene and blackstrap shall not be used in miners' torches for illuminating purposes in any coal mine in this state."

Legal Requirements. Testing for Smoke—Forms of Lamps. Testing Safety Lamp Oil

The law further states the test the oil must stand, giving in a general way the method of making it. The test is based on the specific gravity of the oil, which must not exceed 24° B. scale, and although some of the physical properties of illuminating oils bear a close relation to the specific gravity, thus allowing it to be taken as an indicator, it is necessary to make further tests to determine the efficiency and relative values of various oils. This fact, together with the desirability of establishing specifications for the purchase of oils led to a series of tests both in the laboratory and the mine.

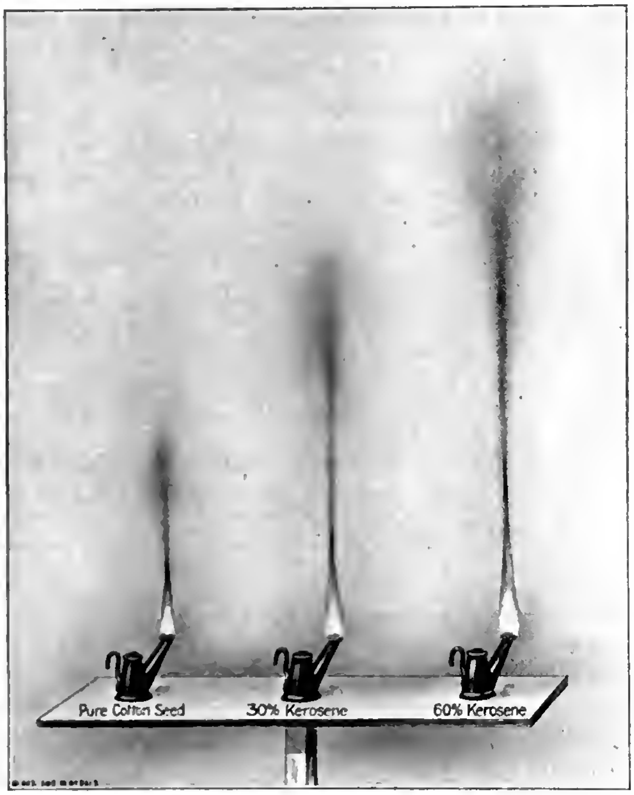


Fig. 1. Effect of Kerosene in Lamp Oil

Samples from mine stores over the entire Fairmont region were tested and analyzed and in the majority were found to be within the state law requirements. Samples of oil being sold by other than coal company stores throughout the Fairmont region were also tested, and in most cases these oils failed to stand the test. Needless to say, many of these oils are adulterated. Oil adulterants are cheap and unfortunately cannot be readily detected, outside of the laboratory.

The following case illustrates one of the difficulties to be met in attempting to prevent the use of inferior oil: A miner, burning oil which he had purchased at a country store, handed his bottle of oil to the district mine inspector and asked him if that oil was all right. The inspector, after shaking the bottle and looking at the oil, passed the opinion that he thought it was. A sample of this oil was later brought into the laboratory and when tested was found to exceed the state law limit in density and to contain 58 per cent. mineral oil. It was an inferior grade of oil highly adulterated. This was a case where both the district mine inspector and the mine boss were taken advantage of, due to the fact that the inspector had not been provided by the state with the necessary apparatus for making a decisive test, and because of the difficulties in making such a test outside the laboratory.

In this connection Section 2 under Oil Regulations of State Mining Law is here given: "Any person, firm, or corporation either by themselves or an agent or employe, which shall sell or offer for sale for illuminating in any mine in this state, any oil or any mixture or compound of oils which does not comply with the tests as prescribed in Section 1 of this Act, shall be deemed guilty of a misdemeanor, and upon conviction thereof shall be fined not less than $25 nor more than $100 for each offense."

Aside from the necessity of carrying out the requirements of the law, there are advantages in supplying the miners with the best oil obtainable, and prohibiting the use of such oils as are usually sold by miscellaneous dealers.

In making the laboratory tests, a standard No. 2 miners' lamp was used for comparing length of flame, smoke evolution, quantity of oil burned and the candlepower of the different oils. The viscosity, congealing point, density and flash points were also determined on each sample.

The viscosity of an oil is the property of the different particles to hold together and at the same time adhere to other bodies. It may be called the "fluidity" or "thickness" of an oil. Oil of high viscosity does not flow so freely as an oil of low viscosity. Viscosity has been the cause of complaint with miners' oil, otherwise satisfactory, as for instance, an oil which gave the following results upon test: Viscosity, 16.2; density, 22° B.; grams oil burned per minute, .2; height of flame, 4 inches. This oil was the cause of dissatisfaction at several of the mines the complaint being that it did not keep lighted on headings and air-courses where strong air-currents were met with, that it was hard to relight, and that the oil did not feed up the wick fast enough. While the flash point of this oil was somewhat to blame for the trouble, it was principally due to the high viscosity.

The congealing point of oil is an important factor in the winter months. It is the temperature at which the oil will solidify, and if this point is too high it is the cause of considerable trouble both to the miner and storekeeper, which results in a loss of time and waste of oil. Although the temperature inside the mines varies but little, there are many cases where roadmen, drivers, motormen, and others, come in contact with air that is almost as cold as that outside, consequently, when buying winter oil there should be a guarantee that it will not congeal at a temperature above 32° F. and it is advantageous to have this point even lower. The winter oil used by the Fairmont Coal Co. shows a congealing point of 21° F. on an average of 22 samples tested. This allows it to be stored and handled in cold places without freezing.

The flash point is the temperature in degrees Fahrenheit at which the oil will flash or ignite. For instance, an oil with a flash point of 300° is one that will ignite when heated to that temperature, if a flame be applied to it. The flash point decreases as the density becomes less; the lighter oils flashing at lower temperatures. Some companies have 300° to 425° F. established as the limit in their specifications for miners' oil, that is, the flash point must not drop below 300° nor exceed 425° F.; the lower limit being applied as a measure of safety; the higher one as a limit at which the oil will readily burn and relight quickly.

The quantity of oil burned for a given length of time is important from the miner's point of view (although it is hard to make him realize it). In trying to comply with the state law,

* A paper read at the winter meeting of the West Virginia Mining Institute.

and also furnish an oil that will give the miner the best results per unit of cost, it is necessary to take into consideration its efficiency. The grams of oil burned per minute depend on th size of the lamp used, the size of the wick, and the length of the wick beyond the spout, as well as the quality of oil. With the first-mentioned conditions remaining constant, the variation in the grams burned per minute depends to some extent upon the density. With a cottonseed oil of high specific gravity the oil burned per minute in a standard No. 2 mine lamp is about .2 gram, while in an oil highly adulterated with kerosene, or a light, highly volatile oil, and of a low specfic gravity, the figure is about .5 gram.

The candlepower of oil naturally bears a close relation to the quantity of oil burned, and to get a comparative figure on different oils as to this property, the candlepower is calculated to candlepower per gram per minute. This takes into consideration the effect of different rates of burning. In making the test of the candlepower, standard photometric candles (12 to a pound) are used, in an ordinary photometer, and the results check very closely. The present specifications call for an equivalent candlepower per gram per minute at 8.

The height of flame is also taken into consideration and is specified in purchasing oil to be between 4 and 5.5 inches with the wick extending ¼ inch beyond the end of the lamp spout, with 12-strand wick. This height has been found to produce the best candlepower with the least amount of smoke, and if the height of flame is limited, the evolution of smoke is to some extent controlled.

In testing miners' oil there is probably nothing of greater importance than the quantity of smoke it gives off while burning. This smoke is, in a measure, indicative of the quality of the oil, as an inferior oil always makes denser smoke and a greater quantity of it, and it has often been noted that the smoke increases as the oil deteriorates in quality. It is this point which guides the mine inspector in his inspection of the oil used at different mines. If the oil gives off an excessive amount of smoke, he becomes suspicious and forwards a sample to the head of the department for test, and it usually develops that the oil is adulterated and not up to the standard required. Aside from the dust and the smoke of the powder, smoke from open lights is the principal cause of air vitiation in the working places of a mine. This is especially noticeable in places where the velocity of the current is very low, and with oil of low grade. An oil to which kerosene has been added (which is common practice in most regions) is the greatest offender in this direction. Its effect on the quantity of smoke produced is shown in Fig. 1.

This is a sketch from a photograph; beginning on the left, the first lamp contains pure cottonseed oil, the next contains 30 per cent. kerosene and the third contains 60 per cent. While it is true that drivers, motormen, and trappers spend most of their time in an air-current of high velocity, oil can be purchased under most severe conditions, and still come up to standard for them, that will burn with a long flame and hold to the wick specifications, with the exception of the density requirement.

The consumption of oxygen due to the burning of oil in a miner's lamp is very small in amount, since it has been figured that only $\frac{1}{17}$ of the total pure air in a mine is required by the miners, horses, and lamps. It is the quantity of smoke given off that causes the greatest vitiation of the mine air.

Although it is difficult to measure the quantity of smoke given off by any particular lamp, the following method, which is similar to the one used by the state mine inspector, gives comparative results and is sufficient for testing miners' oil. This test is made by observation, using pure cottonseed oil as the standard of comparison. The size of lamp, size and length of wick, and the quantity of oil are the same in each case, and length of the smoke pencil on the standard oil is taken as one, and the length of the smoke pencil on the oil tested is relatively greater or less. The lamps are placed in a box with a glass front, and a white cloth, graduated in inches, serves as a background. The limit placed on the smoke is a pencil about 1.5 greater than that of the standard cottonseed oil. This test, however, is unnecessary quite often when other requirements above mentioned are complied with, as they govern to a very great extent, the evolution of smoke.

Apparently the most difficult requirement to meet is the density, which must not exceed 24° B. scale, which corresponds to .913 specific gravity. Of many samples of miners' oil of different kinds submitted by many companies, there has been only a small percentage to come within the limit. Its adoption as part of the specifications of some companies has been due solely to the fact that the State Oil Regulations require it. Miners' oil of excellent quality can be obtained at very reasonable prices, only a degree over the state law limit, but is not purchased because it exceeds 24° B. This test is made at 60° F., and while certain properties do bear some relation to the density, there is no reason, apparent, why this limit should have been placed as low as 24° B. Very seldom has a miners' oil sample been received in the laboratory that does not exceed this point. We have found oils up to 30° B. that were highly satisfactory for use in miners' lamps, being pure unadulterated oils "as free from the evolution of smoke as a standard cottonseed oil." In fact, they were more satisfactory than standard cottonseed oil. The following tests are an example of this.

	No. 1 Cottonseed Oil	No. 2 Colza Oil
Viscosity	18	7
Density	22.0° B.	30.6° B.
Flash point	532° F.	306° F.
Congealing point	22° F.	20.6° F.
Candlepower	1.19	1.44
Flame height	3.5"	4.5"
Grams oil per minute	.17	.19
Candlepower per gram per minute	5.95	7.58
Smoke	small amount	small amount

The report on these two oils from the test given them in the mine is as follows: "Oil No. 2 seemed to be far superior in every way; it made a bright, clear flame, which did not easily blow off of the wick as does oil No. 1, and it made no more smoke." In addition to this it seemed to have fine lasting qualities, as a standard No. 2 miner's lamp, of the Star pattern, burned 1 hour and 35 minutes with one filling, which is at least 15 minutes longer than the same lamp will burn under the same conditions with oil No. 1."

Again, we have an oil that is highly satisfactory and gave the following results when tested: Density, 24.8° B.; flash point, 409° F.; congealing point, 14° F.; oil burned per minute, .4 gram; flame height, 4.5 inches; smoke, small amount; both this oil and the colza are beyond the 24° B. limit, yet they are pure oils and give little or no more smoke on burning, and due to the difference in price as against the standard cottonseed oil, they could be supplied to the miner at much lower price and would give him a more efficient light in every respect, hence are more desirable than the oil that comes within the present limit of the State Oil Regulations. There is an excellent mixed oil, with cottonseed as its base, recently submitted at a price considerably under that of cottonseed, which gave the following results: Density at 60°, 25.0° B.; flash point, 392° F.; congealing point, 16° F.; grams oil per minute, .18; candlepower, per gram per minute, 7.3; height of flame, 5½ inches; smoke, slight.

A trial test was made extending over several months on standard cottonseed oil, and the reports from 41 mines using it were, in effect, that it was of such high density (low Baume) and high flash point as to be very unsatisfactory. The flame would not hold to the wick, the lamp once out was hard to relight, the light furnished was of low candlepower, and the cost was excessive.

The following table serves to give an idea of what may be expected of a standard cottonseed oil when tested:

Viscosity	Density	Candle-power	Grams Oil (Per Minute)	Height Flame Inches	Flash Point
19.3	22.1° B.	1.23	.15	1¾	560° F.
18.2	21.8° B.	1.25	.17	2¼	580° F.
18.0	21.7° B.	1.32	.19	4	587° F.
20.0	22.0° B.	1.34	.18	3½	552° F.
17.5	22.0° B.	1.34	.17	3½	532° F.
18.0	22.0° B.	1.35	.19	3¾	590° F.
17.1	22.3° B.	1.46	.20	3½	562° F.
16.2	22.0° B.	1.54	.20	4	515° F.

If the limit on density as now stated in the State Oil Regulations were raised to 26° B. there would be advantages to the mine owner and the miner, as well as the oil refiner. The miner could then be supplied with an oil of greater illuminating value at a lower cost and there would not be the tendency to mix light burning oils with it, on the part of the miner. We can see no reason why the act governing this should not be changed to read 26° instead of 24°.

Summing up, the present specifications on miners' oil as adopted by the Fairmont Coal Co. read as follows: Density 24° B. or under; congealing point, 24° F. or under; flash point, 300° to 425° F.; flame height, between 4 and 5.5 inches (in No. 2 miners' Star pattern lamp with ¼-inch wick projection, 12-strand cotton wick); equivalent candlepower per gram per minute, 8 or over; smoke, to be light in quantity.

Fig. 2. Miners' Lamps

When a new oil is to be purchased, samples of it are received and tested and unless it comes up to these specifications it is not considered by the purchasing agent. If a purchase is made of an oil that has been found to be all right, each consignment is tested as it is received at the supply department. If it should here fail to come up to the requirements, as stated, it is rejected. It is a rule to test the oil at every mine at regular intervals as well as each shipment received by the supply department.

Lamps.—Using a standard test oil, different style lamps were tested for efficiency, the points considered being the ability of lamps to hold flame under different velocities of air-currents, the candlepower and the amount of oil burnt per hour. The two lamps which gave the best results are standard-size No. 2 lamps, one with a double-wick spout, the other with a single spout and cup at upper end of spout, which is connected to the bottom of the lamp by means of a wire.

These two lamps, shown in Fig. 2, give equally good results. Both have their merits; the outside casing of the double-spout lamp is a protection to the inside tube, keeping it at a uniformly higher temperature and thereby keeping the oil at a temperature which allows it to feed up the wick sufficiently to give the necessary supply, regardless of the temperature of the surrounding air. The wire connecting the cup to the bottom on the other lamp serves the same purpose, and the cup around the top of the spout catches any overflow of oil and keeps the lamp clean.

Safety Lamp Oils.—The Wolf safety lamp is almost exclusively used in this region. Not only is it admirably adapted to the needs of a working lamp but it is a quick and reliable indicator of the presence of firedamp.

The specifications for the naphtha used in these lamps are those recommended by the manufacturer, as follows:

"The naphtha that should be used in safety lamps is a product of a distillation from crude petroleum and must be rectified safety-lamp naphtha and at 15° C. (59° F.) should have a specific gravity of .690 to .730, and below 60° C. (108° F.) (measured in its vapor) no appreciable quantity should be distilled over. At 150° C. (302° F.) all of the naphtha should distill over.

"Poured upon a flat dish, all the naphtha must evaporate completely at common temperature without leaving any petroleum products. Filter paper saturated with naphtha must become perfectly dry in a very short time without leaving any spots which would make the paper appear translucent. Smooth writing paper which has been saturated with naphtha must also become dry in a very short time, so that one can write on it with ink without the ink running on the paper.

"In order to assure a perfect lighting of the lamp by means of an igniting device there must be a sufficient quantity of low-boiling hydrocarbons present, so that at 100° C. (212° F.) (measured in its vapor) at least 73 per cent. and not more than 87 per cent. of its volume will distill over. Heavy hydrocarbons which boil over 150° C. (302° F.) must not exist, as a bright flame, which is an important feature, would be impossible.

"The naphtha must have an aromatic and not a petroleum odor. The candlepower of the naphtha which is supplied should be such that in a properly cleaned and filled Wolf naphtha lamp, giving a flame about 30 millimeters (1½ inches) long, should give at least $\frac{8}{10}$ candlepower. For the determination of the candlepower, a Bunsen photometer is used and the paraffin candle adopted by the Association of German Gas Engineers, giving a flame 50 millimeters (2 inches) long, is accepted as a unit."

In addition to the Wolf safety lamp, the gas inspector of this company used a Chesneau safety lamp, which indicates the presence of firedamp down to .5 of 1 per cent. This is an alcohol lamp in which liquor of the following composition is used: Methyl alcohol (CH_2OH density .8275), 1 liter; copper nitrate, $[Cn(NO_3)_2]$, 1 gram; ethylene bichloride ($C_2H_3Cl_2$), 1 gram.

FRENCH COAL MARKET

No attempt has been made to import American coal into France by way of Rouen, and for the purpose of enabling American producers to consider this question, and to give them a standard for comparing American coals with those most in demand there, the following analyses of British coal are given, together with prices, etc.:

Constituents	Anthracite		Bituminous	
	Highest Per Cent.	Lowest Per Cent.	Highest Per Cent.	Lowest Per Cent.
Fixed carbon	92.30	89.00	72.70	64.70
Volatile matter	4.30	5.50	18.60	25.30
Sulphur	.80	1.50	2.40	2.00
Ash	1.70	2.70	5.50	6.50
Water	.90	1.30	.80	1.50

Bituminous coal for French use is usually brought as "through" coal—what is termed "mine run" in the United States and "tout venant" (all coming) in France—or broken, according to the following sizes: Large, 2⅓ inches; minimum, up to large blocks; cobbles, 2¼ to 3½ or 4 inches; nut, 1¾ to 2¼ inches; small pea, ⅓ to ¾ inch; large pea, ⅜ to ¾ inch; duff, the dust left after extracting the foregoing.—*United States Consular Report.*

FIRING AMMONIUM NITRATE EXPLOSIVES *

The consumption of ammonium nitrate safety explosives is steadily increasing. It is of importance not only in gaseous mines on account of the safety from gas ignition, but also for other blasting purposes on account of the safety in transportation, storage and use. Professor Will, in a lecture delivered before the Austrian Association of Engineers and Architects in the beginning of April, 1910, stated that the consumption of these explosives in Germany has already exceeded that of dynamite. This marked increase is due in some measure to the fact that the majority of German coal mines contain firedamp, and blasting in them is permitted only with safety explosives.

Slow Action of These Explosives Requires Special Igniters to Insure Proper Detonation

In Austria the estimated yearly consumption of ammonium nitrate explosives (ammonal, dynammon, permissible dynammon, progressite, etc.) is in the neighborhood of 1,100,000 pounds. Of this consumption ammonal constitutes by far the largest percentage, as its enormous potential energy offers many economic advantages. Ammonal has also been found to be reliable for blasting in coal mines without firedamp, and with comparatively small charges produces a good fall of lump coal. The consumption in different coal districts, particularly in the Bohemian brown coal (lignite) districts, has already reached 330,000 pounds per annum. For gaseous mines, another variety of ammonium-nitrate explosives must be employed, which is characterized by low explosive temperature and small flame production.

So far only compounds of the carbonite type can compete with the ammonium nitrate in gaseous localities, but it is apparent that the economic advantage is on the side of the ammonium-nitrate explosives; since in the carbonites, which contain on an average 25 per cent. nitroglycerine, the heat production of the nitroglycerine must be reduced by an admixture with suitable substances which do not add anything, or very little, to its potential energy, whereas, the ammonium-nitrate explosives through their principal and main ingredient produce the low temperature necessary.

The conditions for all ammonium-nitrate explosives have become most favorable since the supply of its chief ingredient is assured. Fears of an inevitable crisis in all modern explosives with the exception of the chlorates and perchlorates, which have arisen recently through the impending exhaustion of the Chilian saltpeter deposits and the consequent increase in price of nitric acid, have fortunately proven to be without foundation, since nitric acid can now be produced in unlimited quantities from the atmosphere. Not only nitric acid, but also nitrates have already been produced in this manner in large quantities and put on the market. This assures a rational production of the ammonium-nitrate compounds for the future.

At first considerable difficulty was experienced in the use of all the ammonium-nitrate explosives, through their failure to explode or through burning out of the charge, and from large collars remaining at the bottom of the drill hole. Even now after these compounds have been in use for years, new consumers, who are not well versed in the use of these explosives time and again complain of misfires. However, careful inquiry always shows that this is not due to deterioration or unevenness in the quality of the explosive, but to the use of too weak or poor blasting caps. Deterioration from dampness on account of the hygroscopic nature of ammonium-nitrate cannot very well occur if the explosives are packed carefully and handled with ordinary care. But not only for civil but also for military purposes where the ammonium-nitrate explosives have been in use for a long time for explosive projectile charges, it has been clearly demonstrated that everything depends on the primary ignition, and that poor ignition will produce practically no result, whereas, good ignition can produce a remarkably high maximum. This is a peculiarity of these explosives, and it depends on the chemical and physical properties of its chief ingredient, ammonium nitrate. Whereas the smallest spark is sufficient to completely ignite and explode any quantity of ordinary black powder, and dynamite and kindred explosives can be readily and completely detonated by a sympathetic explosive wave due to a slight mechanical impulse or by heating it too rapidly, the ammonium-nitrate explosives proper act entirely different.

Ammonium-nitrate explosives proper comprise those which contain the latter as chief ingredient, and not merely as adjuvant admixture, as for example ammonium dynamites. The ammonium-nitrate compounds proper require for complete detonation, even in an enclosed space, a very strong initial impulse, since they resist disintegration through the slowness with which they transmit the explosive wave. The easily detonated compounds may be compared to a pyramid resting on its apex which is readily overthrown by a slight blow of a hammer, the ammonium explosives, however, can be compared to a pyramid resting on its base which requires a strong blow to overthrow it.

How is it possible then to develop the maximum energy from the ammonium-nitrate explosives? A few, as dynammon (composed of ammonium nitrate and charcoal), may be completely ignited under special conditions in an enclosed space, by a small initial charge of black powder. This is, however, merely a rapid combustion similar to that of black powder, and its increased energy over the latter depends solely in the increased volume of gas developed by ammonium nitrate.

A true detonation corresponding to the action of dynamite can be produced only with the assistance of a rapidly detonating compound. As dynammon is also suitable for shooting, an ammonium-nitrate powder charge may be ignited in a gun by a suitable black powder ignition, and very good results are obtained, very much superior to the common black powder. But if the same charge should be ignited by a strong fulminate blasting cap, instead of discharging the projectile, the barrel of the gun would be blown to pieces.

Up to the present time, fulminate blasting caps are used almost exclusively as detonators for the ammonium-nitrate explosives. These are composed either of fulminate with a little potassium chlorate or of a mixture of fulminate and trinitrotoluol. The latter is said to increase the efficiency and at the same time diminish the danger from jar and friction. These caps, which are also employed for setting off dynamite, must be fairly large (at least 1 gram No. 6) for the ammonium-nitrate explosives to secure a complete detonation of the charge.

Too much emphasis cannot be laid on the fact that one should not be saving in the size (strength) of the blasting cap. Even for dynamite a No. 6 cap is at present used quite frequently instead of No. 3 which was formerly employed. In England a No. 7 cap is generally used for the ammonium-nitrate compounds but in Germany a No. 8 cap is employed almost exclusively. The last number (with a 2-gram charge) is to be most recommended; larger sizes offer no additional further advantages. The slightly increased cost of the larger blasting caps (the difference in prices between caps No. 6 and No. 8 is about $.01 per 100) is repaid by the better and more certain results and a slight diminution in the weight of the main charge; for larger charges moreover the cost of the blasting caps constitutes merely a small fraction of the total cost of the charge. Since the blasting caps are markedly affected by moisture, care must be exercised in storing them in dry places. In practice deteriorated caps are found much more frequently than one would expect; these are still efficient for dynamite, but not for the ammonium-nitrate explosives.

There is a very good simple test for the blasting caps described in Heyse's book of "The Explosives."

The blasting caps are placed upright on a piece of soft

* Translated from *Zeitschrift fuer das gesamte Schiess-und Sprengstoffwesen*, etc., by W. H. Blumenstein, Pottsville, Penna., manufacturer of the ammonium nitrate safety explosive, "Kanite."

sheet lead 4 millimeters thick, and ignited with a Bickford fuse. From the caliber of the hole and the force with which the fragments are driven in, a comparative picture of the quality of the blasting cap may be obtained.

Since the initial impact is all important for the ammonium-nitrate explosives, great care must be exercised to see that the uppermost cartridge which contains the ignition cap is absolutely fresh and dry. Even if the main charge should have been damp, it can still be set off by a dry ignition cartridge containing a strong detonating cap, whereas, a moist ignition cartridge or weak detonating cap will fail with a perfect main charge.

Another point which must not be lost sight of with the ammonium-nitrate explosives is that the rapidity of detonation (i. e., the rate at which the explosion wave is transmitted through the explosive) increases markedly, particularly in these compounds, with the diameter of the cartridge, and for this reason drill holes of greater diameter (30 millimeters = $1\frac{1}{4}$ inches to $1\frac{1}{2}$ inches and up) are to be preferred. In drill holes of great length and small diameters portions of the charge frequently fail to explode. The lowermost cartridges may be powerfully compressed and become non-inflammable before they can be detonated. For this reason also an ammonium-nitrate explosive charge in a drill hole can rarely be exploded by placing a dynamite cartridge on top of it. On account of the slowness of the ammonium-nitrate compounds, which increases with their density, it is to be recommended to loosen the charge in the ignition cartridge, if the latter has become hard, by simple pressure with the fingers, and not to tamp the main charge in the drill hole, but to place it loosely.

The method of ignition suggested by the French engineer L'Heure, would offer great advantages for all ammonium-nitrate explosives. It consists of a thin lead tube (a few millimeters in diameter) said to be filled with trinitrotoluol which passes through the entire length of the charge and is ignited at its upper end either within or outside of the drill hole by a detonating cap.

This method produces a more rapid and complete detonation of the charge, than could otherwise, on account of its slowness, be accomplished by detonating it from its upper end alone. According to the reports, this method of ignition is said to offer a saving of 20 per cent. in explosives on account of its increased efficiency, and is therefore economical in spite of the increased cost of the fuse. The partial non-ignition of charges in long drill holes would then be entirely prevented.

In the interest of a more thorough utilization of the ammonium-nitrate explosives a complete development and popularization of this method of ignition would be highly desirable.

Electric ignition also has its advantages for the ammonium-nitrate explosives, particularly for use in mines. The peculiarity of most of these explosives to cause no smoke, is particularly well shown in this way. The smoke from the fuse as a rule obscures the observation of its smokelessness. Dynamite it is true also produces little smoke, but it gives rise to poisonous gases.

Whatever the method of ignition employed, whether Bickford or electric fuse, the strength of the detonation, in other words the initial impulse, is decisive for the result of the blast. In the L'Heure ignition also with the trinitrotoluol core, the selection of the detonating cap is of similar importance, since the trinitrotoluol also requires a strong shock for detonation.

It is of interest to give a brief résumé of the development of the methods of ignition and of the industries connected with it.

There was first ignition by means of a blade of straw, which were generally prepared by children, women, or mine cripples. Then came the Bickford fuse, the invention of a simple miner in Cornwall which was a great blessing for safety in blasting and also initiated a number of flourishing factories in different parts of the world. Dynamite produced the detonating caps which greatly increased the sphere of utility of the already existing rifle ammunition factories. The ammonium nitrate explosives called for the larger caps from No. 6 up.

The electric ignition which is closely associated with all methods; viz., black powder, detonating caps, etc., has also made advances, as it permits the simultaneous discharge of several shots in large mines.

The simple and practical Bickford fuse, however, in all probability will never be entirely displaced in the technic of explosives.

COMPENSATING GRADES

Written for Mines and Minerals, by R. Dawson Norris Hall, E. M., Du Bois, Pa.

Compensation for Mine Railroad Sidings—The Increase or Decrease of Grades on Railroad Curves

In railroad language, compensation is the allowance made in a grade to ease the passage of rolling stock over curved track. The curves cause a resistance, which is dependent in amount on the degree of curvature. Such resistance would seem to be measurable with exactitude. As a matter of fact, the tightness of gauge, the shape of the wheel tread, the shape and condition of the rail, the wheel base of the engine and cars, their stiffness and the elevation of the rail, all have their bearing on the allowance so that it is not easy to get an absolutely correct value for all roads and conditions of running. It is assumed between .03 per cent. and .05 per cent. per degree of curve. That is, it is as hard to overcome gravity in pulling up a .03-per-cent. grade on a straight line as to overcome the resistance of pulling the same load around a 1-degree curve on a level if adequate compensation is .03 per cent. The method adopted to ascertain the value of a "compensated grade" or "grade with compensation" is to add to the total rise of the grade a sum equal to .03 or .05 times the total curvature of the track from end to end, totaling the curvature arithmetically without regard to any consideration of the direction of that curvature, right or left. The result thus obtained is divided by the total length of the line in even station lengths of 100 feet, figuring, of course, only that part of the road which it is desired to keep at the same compensated grade.

Considering a road 12,000 feet long having central curve angles of 15 degrees, 30 degrees, and 45 degrees right and 20 degrees, 50 degrees, and 90 degrees left, and having a total rise of 200 feet, the total equivalent grade will be $200 + .03\ (15 + 30 + 45 + 20 + 50 + 90) = 207.5$ feet.

Dividing by the 120 even stations in 12,000 feet we obtain the compensated grade, or 1.729 per cent.

Thus the straight alinements which, on an uncompensated grade, would slope 1.666 per cent., have that slope increased to 1.729 per cent. when compensation is provided for. Suppose the curve above stated as having a 15-degree central angle is 300 feet long. Then the degree of curvature is 5 degrees. The grade on the curve will be therefore $1.729 - .3 \times 5 = 1.579$ per cent. and the resistance on the curve will equal the resistance on the straight line with a grade of 1.729 per cent. for all cars on ascending the grade.

The purpose of this article is not to discuss compensation as practiced in ordinary railroad work but to show under what conditions it should be reversed. The purpose of compensation, as viewed in general practice, is to aid the climbing of a hill by the rolling stock. The assumption is that the loads will be approximately equal both ways, but in mineral tracks this is not usually the case. Under the tipple the movement of the cars is always one way, and the conditions require an equal force to move the cars steadily forward. If any force acts against the action of gravity, such as the resistance of a curve, the grade should be increased at that point to make the net impelling force a constant. If a 1.75-per-cent. grade is needed above the tipple and a 10-per-cent. curve must be constructed

at that point, the grade should be increased to 2.05 per cent. Nor should the needs for increased gravity impulsion over switches be overlooked. This increase of grade at curves may be appropriately termed inverse compensation. Unless watched, some railroad engineers will follow the "practical" rule and make the 1.75 per cent. on the curve a 1.45 per cent. without considering the natural reversal of conditions. It is true that few in the profession would deliberately arrange such a condition, but when scheming out the run-around tracks, there is a risk that the differentiation between them and the loading tracks may be overlooked. No wonder that curved tracks under a tipple are so freely reprobated.

Where the grade of a railroad is about .4 per cent. it is as easy to pull loads down the hill as to pull empties up. When such an even balance can be effected, it would be a pity to spoil it by a miscalculation of the compensation. But first it will be well to calculate this ideal grade which requires equal tractive power to be expended on descending loads and ascending empties. This can best be done by assuming a special but normal make up of train, as follows:

Weight of engine and tender = 220,000 pounds; weight of 40 mixed wood and steel cars = 1,600,000 pounds; total weight of train = 1,820,000 pounds.

Let the required grade be x per cent. Then the gravity resistance $= \text{weight} \times \frac{x}{100} = 18{,}200\,x$.

If the cars once started will move down a grade of .7 per cent. without steam and without increase of speed on a straight track, then the stiffness and rolling resistance together = weight $\times \frac{.7}{100} = 12{,}740$ pounds. The gross resistance will equal $18{,}200\,x + 12{,}740$ pounds. Going down the hill loaded, assuming the cars to carry 100,000 pounds apiece, they will weigh $40 \times 140{,}000 = 5{,}600{,}000$ pounds. Adding the weight of the engine as before, the weight of the whole rolling load will be 5,820,000 pounds. The gravity assistance to motion would overcome a resistance equal to $58{,}200\,x$. The stiffness and rolling resistance together will equal $58{,}200 \times .7 = 40{,}740$ pounds. The resulting resistance to motion to be overcome by the steam in the cylinders will equal $40{,}740 - 58{,}200\,x$. Equating these resistances $18{,}200\,x + 12{,}740 = 40{,}740 - 58{,}200\,x$

$$76{,}400\,x = 28{,}000$$

$$x = \text{about } .366 \text{ per cent.}$$

Thus the grade which makes resistance equal on a track where the load travels all one way and the empties all the other way is .366 per cent. in favor of the load.

The resistance of an empty train being pulled up the grade on a 1-degree curve will be $18{,}200\,x + 18{,}200 \times .7 + 18{,}200 \times .03$. The resistance of a loaded train going down the same grade on a like curve will be $58{,}200 \times .7 + 58{,}200 \times .03 - 58{,}200\,x$. Equating

$$18{,}200\,x + 58{,}200\,x = (58{,}200 - 18{,}200) \times .03 + (58{,}200 - 18{,}200) \times .7$$

$$76{,}400\,x = 1{,}200 + 28{,}000$$

Hence, $x = .0157 + .366$

The first figure in the result is the result of influence of curvature and it will be seen that with an increase of curvature the balanced grade will increase proportionately. Hence, it will be easy to write a series of values of ideal balancing grades for varying degrees of curvature.

When the straight-line grade is less than .366 per cent. in favor of the load the grade should be increased on the curves, but not to exceed the figures for the balancing grade given in the table. This increase is because, on so gentle a grade, the loads are harder to pull down hill than the empties up hill and the net resistances of the loads approach nearer the capacity of the engine than the resistances of the empties.

When the grade is more than .366 per cent. on a straight line and in favor of the load, then the empties are harder to haul up the ascent than the loads down the descent and the grades should be compensated in favor of the empties, the curve grade being reduced, but not to an amount which will make them less than the figures given in the table as the balancing grades for that degree of curve.

BALANCING GRADES FOR CURVES

Degree of Curvature	Balancing Grade. Per Cent.
0	.366
1	.382
2	.397
3	.413
4	.429
5	.445
6	.460
7	.476
8	.492
9	.507
10	.523

and so on.

Where the grade is against the loads, of course the easing of the grade at curves should be provided for in every case.

The calculations are based on certain assumptions regarding equipment, but it will be found that these assumptions might be varied considerably without changing the balancing grades. The weight of the engine for any given even grade is a certain per cent. of the whole train load. The carrying capacity of the cars is a definite multiple of their light weight, though there is, of course, a variation between wood and steel cars, the latter being the lighter.

EXAMINATION FOR ASSAYERS IN BRITISH COLUMBIA

To practice in British Columbia, assayers are obliged to pass an examination and receive a license.

The examination held April 24, 1911, covered the following subjects:

(*a*) A knowledge of the principles of inorganic chemistry.

(*b*) Sampling of ores and furnace products; preparation of sample for assay, including also the melting of gold dust and sampling of bar for assay.

(*c*) Qualitative determination.

(*d*) Assay of gold; assay of copper and lead-copper bullion for gold and silver; also assay for copper, lead, gold, and silver.

Determination of moisture, volatile combustible matter, fixed carbon, ash, and sulphur in coal.

Fire assays of gold, silver, and lead.

Wet and combined wet and fire assays.

Gold and silver by combined method.

Copper by electrolytic, colorimetric, and volumetric (cyanide or other approved) methods.

Nickel by electrolytic method.

Iron, lead, lime, zinc, sulphur, and silica, by any approved wet methods.

The mineralogical determination of a number of simple substances.

Entrance for examinations must be made in writing to the Secretary of the Board of Examiners at least ten days before the date set for beginning of the examination and must be accompanied by the prescribed fee ($15).

A certificate of efficiency in assaying will upon payment of the prescribed fee ($15) be issued to each successful candidate, which certificate shall be considered as a license to practice assaying in British Columbia, and notice is hereby given that only those holding such certificate of proficiency, or license will be allowed to act as assayers in the province, under penalty as provided by the Act. Graduates of certain Schools of Mines, in Canada, Great Britain, and Ireland, may be exempt from examinations and may upon satisfying the examiners as provided in the Act receive such certificate of competency or license, upon payment of the fee ($15). Any additional information desired may be obtained from Herbert Carmichael, Secretary, Board of Examiners, Victoria, B. C.

ANSWERS TO EXAMINATION QUESTIONS

Answered for Mines and Minerals, by J. T. Beard

Note.—The following questions, selected from different examinations, are here numbered consecutively. The examination from which the question is taken and the number of the question are found at the end of each. (See foot-note).

Selected Questions of the Pennsylvania Anthracite Foreman Examinations, Scranton, April 3-5, 1911

Ques. 1.—How would you direct a miner to restand a post that had been dislodged by a blast at the face of a chamber? Q. 3B.

Ans.—First, carefully examine and test the roof about the place where the post stood, taking down any loose portions. Avoid going under the unsupported roof and do not delay the work of resetting the fallen timber. If, however, the condition is hazardous, come out and wait for the roof to fall. In many cases it will be wise to stand one or more posts temporarily for protection, before resetting the fallen timber. Finally, clean up the floor where the post is to stand, and, having measured the height, set up the timber, with a suitable cap above it, and drive the stick to its place.

Ques. 2.—When is a working place in a mine said to be safe? Q. 4B.

Ans.—When the roof is sound or properly timbered, the coal properly mined, and the place thoroughly ventilated.

Ques. 3.—Name the causes that bring about mine caves. Q. 5B.

Ans.—Mine caves are the result of too large a percentage of coal being taken out in the first working of the seam, or the leaving of inadequate pillars for the support of the overlying strata; or attempting to open out and maintain too large an area before drawing back pillars; or the failure to keep the pillars in a lower seam vertically under those in the seam above when working contiguous seams. Mine caves are also sometimes caused by a squeeze or creep, or by a mine fire destroying the roof supports.

Ques. 4.—If the horsepower of an engine operating a fan be 40, while the horsepower on the air is only 28, what is the percentage of useful effect? Q. 6B.

Ans.— $\frac{28}{40} \times 100 = 70$ per cent.

Ques. 5.—If a water gauge of 2 inches passes 15,000 cubic feet of air per minute, in a certain mine, what water gauge will be required to pass 30,000 cubic feet per minute under the same conditions? Q. 7B.

Ans.—For the same mine or airway the water gauge varies as the square of the quantity of air passing; or, in other words, the water gauge ratio is equal to the square of quantity ratio. Therefore, in this case,

$$\frac{x}{2} = \left(\frac{30,000}{15,000}\right)^2 = 2^2 = 4$$

and $x = 2 \times 4 = 8$-inch water gauge.

Ques. 6.—How would you guard against danger from the accumulation of water in abandoned workings? Q. 8B.

Ans.—Drain the workings and inspect them as often as may be necessary to insure safety. If this plan is impracticable, construct dams in all openings leading to the abandoned workings and make these strong enough to withstand any possible head of water that may arise therein. This latter plan, however, does not remove the danger of water blast that may occur in inclined seams.

Ques. 7.—Explain why the flame will not pass through the gauze of a Davy lamp in a quiet atmosphere. Q. 14B.

Ans.—The wire of the gauze absorbs the heat and so far reduces the temperature of the flame that combustion ceases and the flame is extinguished in close proximity to the gauze. The complete extinction of the flame and prevention of its passage through the gauze is made possible by the outflowing gas being divided into minute streams by the fine mesh of the gauze and thus brought into closer proximity to the cool wires.

Ques. 8.—State fully what action you would take in case you found bad roof at the face of a working place in your district. Q. 15B.

Ans.—Promptly order the men out of the place and immediately proceed to make a careful examination to determine whether the roof can be made secure with proper timbering, or there is loose material that must be taken down or allowed time to fall. The men should then be instructed to set the necessary timbers without delay, or sent out of the mine and the place fenced off and marked dangerous till such time as it can be made safe for work.

Ques. 9.—What is the quantity of air passing through a gangway 12 feet wide and 7 feet 6 inches high where the anemometer registers a velocity of 473 feet per minute? Q. 16B.

Ans.—The sectional area of the gangway is $12 \times 7.5 = 90$ square feet, and the volume of air passing $90 \times 473 = 42,570$ cubic feet per minute.

Ques. 10.—How would you proceed to remove a body of firedamp from the face of a chamber 30 feet wide and 6 feet high? Q. 18B.

Ans.—Before taking any steps to dislodge the gas, withdraw the men working on the return of this current and station trustworthy men to guard each entrance to the same. Begin now at the cross-cut where the air enters the chamber and erect a line of canvas brattice, extending from the outby rib of the opening inward towards the face of the chamber. Advance the brattice slowly, giving the current time to carry off the gas at the face. Use safety lamps only and test the condition of the air behind the brattice; in no case permit any one to carry a lamp beyond the end of the brattice. As the face of the chamber is reached the gas should be brushed from the corners and other lodgments.

Ques. 11.—(*a*) What is flushing, in mining? (*b*) How is it done? (*c*) What precautions would you adopt while flushing? Q. 27B.

Ans.—(*a*) Flushing is the process of washing a large amount of waste material into abandoned portions of a mine for the purpose of filling up the void places so that the pillars in the mine can be removed without serious damage to the overlying surface. (*b*) Flushing is accomplished by arranging a conveyer on the surface to conduct the waste material to the head of the shaft or bore hole where it is fed into a column pipe, together with a continuous stream of water, which carries it in suspension to the point in the mine where it is deposited behind a temporary dam. The water then drains from the deposited material into a ditch, which carries it to a basin or sump, from which it is pumped to the surface to be again used in conveying more material into the mine. (*c*) It is necessary when flushing to watch the operation closely to see that the waste material and the water are fed into the pipe in proper proportions to avoid clogging the pipe; also, to change the position of the discharge from time to time, to secure the successful filling of the void places, as desired.

Ques. 12.—What method would you adopt in order to have a continuous flow of air past the face of each working place? Q. 28B.

Ans.—In the room-and-chamber system, commonly used throughout the anthracite region, the air-current should be deflected by temporary brattices erected in each chamber wherever necessary to cause the air to sweep the face. Doors should be replaced by overcasts on main roads wherever practicable, and double doors should be used at all principal points of the main air-current where a door is required.

Ques. 13.—In considering water-gauge pressure how is the number 5.2 obtained? Q. 29B.

Note.—(*A*) Examination, Districts 1, 2, held at Scranton, April 3-5, 1911. *B*) Examination, Districts 4, 5, held at Scranton, April 3-5, 1911.

Ans.—This number represents the pressure, in pounds per square foot, due to 1 inch of water column. It is, therefore, the weight of $\frac{1}{12}$ of a cubic foot of water; or, $62.5 \div 12 = 5.2$, as shown clearly in Fig. 1.

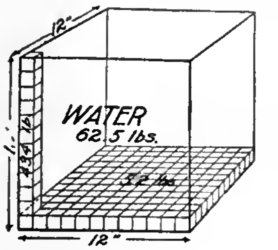


Fig. 1

Ques. 14.—What quantity of air is passing through an airway $6\frac{1}{2}$ feet at the top, $9\frac{1}{2}$ feet at the bottom, and 6 feet high, with an anemometer reading of 300 revolutions per minute? Q. 4A.

Ans.—The sectional area of this airway is $6\,(6.5+9.5) \div 2 = 48$ square feet. The quantity of air passing is, therefore, $300 \times 48 = 14{,}400$ cubic feet per minute, assuming that the reading of the anemometer gives the average velocity of the air-current at this point in the airway.

Ques. 15.—What are the principal causes of mine fires and what precautions would you take to guard against them? Q. 6A.

Ans.—This question is fully answered in reply to Ques. 11, page 734, July, 1910; and Ques. 6, page 52, August, 1910, of Mines and Minerals.

Ques. 16.—(*a*) How many cubic yards of material must be removed in sinking a circular shaft 10 feet in diameter, and 100 feet deep? (*b*) If this shaft was filled with water, how many gallons would it contain? (*c*) What would be the pressure in tons on the bottom of the shaft caused by the weight of the water? Q. 7A.

Ans.—(*a*) The quantity of material excavated would be $100(.7854 \times 10^2) \div 27 = 290\frac{8}{9}$ cubic yards. (*b*) The quantity of water the shaft would hold is $100(.7854 \times 10^2) \times \frac{1728}{231} = 58{,}752$ gallons. (*c*) The weight of the water pressing on the bottom of the shaft is $\frac{100(.7854 \times 10^2) \times 62.5}{2{,}000} = 245.4375$ tons.

Ques. 17.—(*a*) Give a sample report such as you would make after having completed your morning examination of a mine, and finding gas in three places, bad roof in two places, and a fall in an airway sufficient to interfere with the air-current. (*b*) State your duty as mine foreman or assistant mine foreman, after such report is made. Q. 8A.

Ans.—(*a*) Fire boss' report for April 4, 1911, 6:30 a. m.: "I have carefully examined all airways, headings, and chambers in my district and find all working places free from gas and safe, except the following: 1st north, main east, chamber 8, gas; keep out. 1st north, main east, chamber 9, gas; keep out. 1st north, main east, chamber 10, heavy fall of roof and gas; keep out. Also bad roof in chamber 11; keep out. These places are all fenced off and must not be entered. There is also a bad fall at the face of the two blind headings, 5th and 6th north, main east, blocking the cross-cut and checking the air in these headings, which are therefore fenced off. Samuel Jones, Fire Boss." (*b*) It would be the duty of the mine foreman or his assistant, after such report is made, to give his immediate attention to the removal of the dangers mentioned, and to see that the workmen stay out of all places affected till the work is completed and each place has been examined by the fire boss and reported safe. The workmen must be kept out of the return current from the places named while the gas is being removed if much gas is present.

Ques. 18.—What is the ventilating pressure, in pounds per square foot, when the water gauge is $2\frac{3}{4}$ inches? Q. 9A.

Ans.—The corresponding pressure is $2.75 \times 5.2 = 14.3$ pounds per square foot.

Ques. 19.—Explain why the Davy lamp is used by fire bosses and the Clanny lamp by the workmen. Q. 11A.

Ans.—The Davy lamp is sensitive to gas, does not give a good light for working, and is an unsafe lamp in the hands of an inexperienced workman. The Clanny lamp gives a good light, does not flame as quickly as the Davy when exposed to gas, and is or can be protected by a bonnet against strong air-currents.

Ques. 20.—If the run of a mine is 18 cars of coal per keg of powder used, what allowance per car would you make when it is possible to cut only seven cars to the keg; the price of powder being \$1.50 per keg? Q. 12A.

Ans.—The average cost for powder under the two conditions is $\$1.50 \div 18 = 8\frac{1}{3}$ cents per car; and $\$1.50 \div 7 = 21\frac{3}{7}$ cents per car. Hence, the allowance on account of powder to be made is $21\frac{3}{7} - 8\frac{1}{3} = 13\frac{2}{21}$ cents per car, or \$2.75 for every 21 cars of coal mined.

Ques. 21.—In case a fire occurred on the main intake of a mine, endangering the lives of the workmen, by suffocation, state how you would endeavor to rescue them. Q. 14A.

Ans.—Notify the men, at the earliest possible moment, to get out quick by the return airways as far as possible. Reverse, or shut down, or at least slow down the fan, or short circuit the air-current so as to prevent or hinder the smoke and gases of the fire from reaching the men. Ample provision should be made in every mine beforehand to enable the men to reach a safe exit under any conditions that may arise. The danger exists chiefly in the failure to foresee and provide for what may possibly happen in the daily operation of the mine, and the risks assumed are often too great where hundreds of lives are at stake.

Ques. 22.—Name the two gases most commonly found in our mines and state how and where you would expect to find each gas. Q. 3B.

Ans.—Marsh gas or methane (CH_4), and carbon dioxide (CO_2). Marsh gas should be sought with the safety lamp, using some improved type of lamp that will indicate small percentages of the gas and show exactly how much is present in the air. The test for marsh gas should be made at the roof, or at or near the face of rise headings where it may accumulate if the air-current is deficient; also well up on falls and in the accessible portions of disused headings and abandoned workings, and in return airways of gaseous mines. Carbon dioxide may be sought with an open light in low places and dip workings, and in abandoned parts of the mine where the ventilation is slack or not sufficiently strong to carry off the blackdamp as it is formed.

Ques. 23.—What units of work will be absorbed by the frictional resistance of an airway 6 feet square and 1,000 feet long, when the quantity of air passing is 10,800 cubic feet per minute? Q. 11B.

Ans.—The sectional area of the airway is $6 \times 6 = 36$ square feet. The velocity of the air-current is $10{,}800 \div 36 = 300$ feet per minute. The rubbing surface of the airway is $4 \times 6 \times 1{,}000 = 24{,}000$ square feet. The work of friction is then $u = k\,s\,v^3 = .00000002 \times 24{,}000 \times 300^3 = 12{,}960$ foot-pounds per minute.

Ques. 24.—How many cubic yards of coal are there underlying a lot that measures 175 feet long and 55 feet wide, when the vein has an average thickness of 5 feet 9 inches? Q. 12B.

Ans.—The surface area of the lot is $175 \times 55 = 9{,}625$ square feet. Assuming the vein is level its cubical contents underlying the lot is $9{,}625 \times 5.75 \div 27 = 2{,}049.77$ cubic yards.

Ques. 25.—(*a*) What are the dangers attending the use of electricity in coal mines? (*b*) How would you guard against such dangers? Q. 13B.

Ans.—This question is fully answered in reply to Ques. 14, page 53, Mines and Minerals, August, 1910.

來 來

The power required to ventilate a mine varies as the cube of the volumes of air passing. If it requires 25 horsepower to circulate 100,000 cubic feet of air through a mine, what horsepower will be required to pass 200,000 cubic feet? Solving the proposition: $(100{,}000)^3 : (200{,}000)^3 : : 25 : x$, then $x = 200$ horsepower.—*From Jeffrey Mfg. Co. Catalog.*

Catalogs Received

In writing for catalogs, please mention Mines and Minerals

The Toledo Pipe Threading Machine Co., a catalog of 48 pages describing and showing by photographs the possibility of using Toledo pipe-threading devices under conditions where it would be difficult or impossible to use other pipe-threading tools.

Abenaque Machine Works, Westminster Station, Vt., Gas and Gasoline Engines, 28 pages; Abenaque Gas or Gasoline-Driven Direct-Connected Air-Compressor Outfits, 12 pages.

Barco Brass and Joint Co., 56 N. Jefferson St., Chicago, Ill., The Barco Flexible Joint, 16 pages.

Crocker-Wheeler Co., Ampere, N. J., Bulletin No. 128, Form W Motors for Rolling Mills, 12 pages; Bulletin No. 129, Direct Current Crane and Hoist Motors, 20 pages; Bulletin No. 132, Form I Machines, 16 pages.

John Davis & Son (Derby), Ltd., 110 West Fayette St., Baltimore, Md., Bulletin No. 113A, Anemometers, Water Gauges, Hygrometers, Barometers, Engineering and Meteorological Instruments, 44 pages.

De Laval Steam Turbine Co., Trenton, N. J., Catalog "B," De Laval Centrifugal Pumps, 96 pages.

The Excelsior Drill and Mfg. Co., 630-631 Empire Building, Denver, Colo., The Excelsior Airometer, 3 pages.

The Economic Machinery Co., Denver, Colo., The Fairchild Magnetic Separator, 11 pages.

Falkenburg & Laucks, Seattle, Wash., Testing Laboratory Bureau of Inspection, 12 pages.

International Instrument Co., Cambridge, Mass., Catalog C 1910, Centrifuges, 50 pages; Bulletin No. 49, Highway Laboratory Testing Machines, 4 pages.

Keuffel & Esser Co., Hoboken, N. J., Standard Profile Plan Paper, 1 page.

Reinhold Betterman, Johnstown, Pa., The Betterman Standard Boiler Flue Plug, 8 pages.

National Electric Lamp Association, 4411 Hough Ave., Cleveland, Ohio, Bulletin 13A, "Mazda" Multiple Lamps, 16 pages.

Pilley Mfg. Co., 608 Third St., St. Louis, Mo., Pilley's Electric Miners' Lamp, 8 pages.

The National Pulverizer Co., 1738 Broadway, Denver, Colo., The National Pulverizer, 12 pages.

The Wm. Powell Co., Cincinnati, Ohio, Catalog No. 10, Dependable Engineering Specialties, 287 pages.

Scranton Steam Pump Co., Scranton, Pa., Catalog No. 12, Pumping Machinery, 44 pages.

Smith, Emery & Co., Inc., San Francisco, Cal., Assayers and Metallurgists, 8 pages.

Sauveur & Boylston, Cambridge, Mass., Third Edition Apparatus for Metallographic Laboratory, 100 pages.

Starrett Pump and Mfg. Co., Salt Lake City, Utah, A Modern Method of Pumping, 32 pages.

Weber Subterranean Pump Co., 90 West St., New York, N. Y., Weber Subterranean Pump for Artesian Wells, 4 pages.

L. J. Wing Mfg. Co., 90 West St., New York, N. Y., Bulletin No. 7, "Typhoon" Turbine Blowers, 20 pages; Bulletin No. 9, Feed Water Regulator, 8 pages.

The Western Metallurgical Co., 1854 California Street, Denver, Colo., Mineralogist's Pocket Reference, 46 pages.

The Henry E. Wood Ore-Testing Co. (Inc.), 1734 Arapahoe St., Denver, Colo., Ores Tested in Carload Lots, 8 pages.

The J. C. Ulmer Co., Cleveland, Ohio, The Lucas Chain Tape, 6 pages.

Allis-Chalmers Co., Milwaukee, Wis., Bulletin No. 1210, Universal Bolter With Vibromotor Drive, 12 pages; Bulletin No. 1524, Price List of Repair Parts for Auxiliary Air Brake Apparatus, 16 pages.

Pulsometer Steam Pump Co., 17 Battery Place, New York, N. Y., Catalog 17, The Pulsometer, 27 pages.

General Engineering Co., Salt Lake City, Utah, Circular Describing Office and Testing Plant, 4 pages.

E. I. DuPont de Nemours Powder Co., Wilmington, Del., Trap-Shooting, 32 pages; DuPont Lesmok, 8 pages; Infallible Smokeless, 12 pages; Circular Describing Sporting Powders.

Merck & Co., New York, N. Y., 1911 Price List, 80 pages.

Gardner Crusher Co., 550 W. 34th St., New York, catalog describing Gardner Crusher, Disintegrator, and Pulverizer, 12 pages.

Saeger Engine Works, Lansing, Mich., Bulletin 39, describing Olds Mine Hoist, 12 pages.

Pennsylvania Flexible Metallic Tubing Co., 1305 Arch St., Philadelphia, Pa., Interlocking Metal Hose for Highest Pressures, 24 pages.

New Inventions

Complete specifications and drawings for any of the following patents can be obtained from the Commissioner of Patents, Washington, D. C., at the following rates:

Single copies	5 cents each
Copies by subclasses	3 cents each
Copies by classes	2 cents each
An entire set of patents	1 cent each

PATENTS PERTAINING TO MINING ISSUED MAY 2 TO MAY 30, 1911, INCLUSIVE

No. 991,017. Hydraulic coal breaker, William Pill, Cowanporth, England.

No. 990,969. Coaling apparatus, Ysidoro Ferrer, New York, N. Y.

No. 990,859. Apparatus for igniting miners' safety lamps, Ernest Arthur, Hailwood Morley, England.

No. 990,860. Apparatus for testing miners' safety lamps, etc., Ernest Arthur Hailwood, Morley, England.

No. 991,317. Ore crusher or grinder, George Johnston, Chihuahua, Mexico.

No. 992,079. Coking furnace, Leland L. Summers, Chicago, Ill.

No. 992,078. Coking oven, Leland L. Summers, Chicago, Ill.

No. 991,667. Hammer drill, Albert H. Taylor, Easton, Pa.

No. 991,677. Ore feeder, Enos A. Wall, Salt Lake City, Utah.

No. 992,092. Ore jig, George H. Williams, Chicago, Ill.

No. 991,695. Apparatus for electrochemical treatment of ores, Tullio Bubola, Luipaardsvlei, Transvaal.

No. 992,684. Coke oven, Laurence P. Palmer, Uniontown, Pa.

No. 992,492. Coke-oven door, Ludwig Carl Flaccus, New York, N. Y.

No. 992,456. Hammer drill, Albert H. Taylor, Easton, Pa.

No. 992,629. Apparatus for dewatering and classifying ores, Randall P. Akins, Denver, Colo.

No. 993,474. Coal-cutting machine, Louis Thomas, Ans, near Liege, Belgium.

No. 993,783. Crusher head, Volney W. Mason, Jr., New York, N. Y.

No. 993,731. Dumping apparatus, Samuel B. Stine, Osceola Mills, Pa.

No. 993,513. Means for withdrawing dust and fumes from zinc or similar furnaces, Emile Dor Delattre, Liege, Belgium.

No. 993,593. Mining machine, William Guernsey, Prosser, Wash.

No. 993,481. Ore and coal jig, Enos A. Wall, Salt Lake City, Utah.

No. 994,023. Ore crusher, Charles O. Michaelsen, Omaha, Neb.

No. 993,451. Treatment of auriferous and argentiferous ores, Alfred Arthur Lockwood, London, England.

No. 993,424. Throttle valve for rock drills, Daniel S. Waugh, Denver, Colo.